Fruta del Norte Project

Ecuador

NI 43-101 Technical Report on Feasibility Study

Prepared by:
Ignacy (Tony) Lipiec, P.Eng., Amec Foster Wheeler
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Simon Allard, P.Eng., Amec Foster Wheeler
Charles Masala, P.Eng., Amec Foster Wheeler
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Alejandro Sepúlveda, RM CMC, NCL
Tony Maycock, P.Eng., MM Consultores
David Ross, P.Geo., Roscoe Postle Associates

Prepared for:
Lundin Gold Inc.

Effective Date:
30 April, 2016

Project Number:
189565

CERTIFICATE OF QUALIFIED PERSON

I, Ignacy (Tony) Lipiec, P.Eng., am employed as a Director, Process Engineering with Amec Foster Wheeler Americas Limited (Amec Foster Wheeler).

This certificate applies to the technical report titled “Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 30 April 2016 (the “technical report”).

I am a Professional Engineer (P.Eng.) in the Province of British Columbia (#23976). I graduated from the University of British Columbia with a B.A.Sc. degree in Mining & Mineral Process Engineering, in 1985.

I have practiced my profession for 31 years, and have previously been involved with metallurgical design and process engineering for precious metals, base metals, and disseminated sulphide projects in North America and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Fruta del Norte Project (the “Project”).

I am responsible for Sections 1.11, 1.17, 1.21, 1.22, 1.27, 1.28; Sections 2.3, 2.5, 2.6.1; Sections 3.1, 3.2, 3.3; Section 13; Section 17; Sections 21.1.1, 21.1.2, 21.1.3, 21.1.5, 21.1.6, 21.1.7, 21.2.1, 21.2.3, 21.2.5, 21.2.6, 21.3; Sections 24.2, 24.3; Sections 25.5, 25.9, 25.13, 25.14, 25.17, 25.18, Section 26.3.4; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

“Signed and sealed”

Ignacy (Tony) Lipiec, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

I, Juleen Brown, MAusIMM CP., am employed as a Mining Sector Lead – Environment with Amec Foster Wheeler Australia Pty Ltd (Amec Foster Wheeler).

This certificate applies to the technical report titled "Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 30 April 2016 (the "technical report").

I am a Member of the Australasian Institute of Mining and Metallurgy (MAusIMM) and a Chartered Professional (CP) with the AusIMM (#201809). I graduated from the University of Queensland in 1999 with a Bachelor of Engineering degree in mining, and obtained a Master's degree in Environmental Management from the same institution in 2006.

I have practiced my profession for 14 years since graduation. I have been directly involved in environmental management of mining operations, environmental input and planning into feasibility studies for mining projects, environmental and social due diligence of mining operations and projects, mine closure input and reviews, and managing large multidisciplinary Environmental and Social Impact Assessments to IFC standards. My experience includes coal and metalliferous mines in Australia and the Asia-Pacific.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Fruta del Norte Project (the "Project").

I am responsible for Sections 1.19.1, 1.19.8, 1.19.9, 1.19.10, 1.27, 1.28; Sections 2.3, 2.5, 2.6.1, 2.6.2; Sections 3.1, 3.2, 3.3, 3.4; Sections 20.1, 20.8. 20.9, 20.10, 20.11.1, 20.11.6, 20.11.7, 20.11.8, 20.11.9; Sections 25.11.1, 25.11.7, 25.11.8, 25.11.9, 25.17, 25.18; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

"Signed"

Juleen Brown, MAusIMM CP.

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CERTIFICATE OF QUALIFIED PERSON

I, Simon Allard, P.Eng., am employed as a Principal Consultant and Study Manager with Amec Foster Wheeler Americas Limited (Amec Foster Wheeler).

This certificate applies to the technical report titled "Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 30 April 2016 (the "technical report").

I am a registered Professional Engineer (P. Eng.) in the Province of British Columbia. I graduated from Université Laval in 2004 with a Baccalauréat coopératif en génie des mines et de la minéralurgie Degree.

I have practiced my profession for 12 years. I have been directly involved in cash-flow modelling, risk evaluation, real-options valuation, financial analysis, marketing studies and financial review of mines located in Africa, Mongolia and North and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Fruta del Norte Project (the "Project").

I am responsible for Sections 1.1, 1.20, 1.23, 1.24, 1.27, 1.28; Sections 2.3, 2.5, 2.6.1; Sections 3.1, 3.2, 3.3, 3.5, 3.6, Section 19, Section 22, Sections 25.12, 25.15, 25.18; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

"Signed and sealed"

Simon Allard, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Charles Masala, P.E. P.Eng., am employed as an Associate Water Resources Engineer with Amec Foster Wheeler Americas Limited (Amec Foster Wheeler).

This certificate applies to the technical report titled “Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 30 April 2016 (the “technical report”).

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (# 28339) and Washington Association of Professional Engineers (# 45248). I graduated from the University of Zambia with a Bachelor of Engineering degree in 1989, from the University of New Castle Upon Tyne with a Master of Science degree in 1990 and from the University of British Columbia with a Master of Applied Science degree in 1995.

I have practiced my profession for 22 years. I have been directly involved in studies relating to mine water management, including water quality and characterization, surface water management and design, and water treatment in North and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Fruta del Norte Project (the “Project”) between April 1 to April 2, 2015.

I am responsible for Sections 1.19.7, 1.27, 1.28; Sections 2.3, 2.4, 2.5, 2.6.1, 2.6.2; Sections 3.1, 3.2, 3.3; Section 18.7; Sections 20.7, 20.11.5; Section 24.3; Sections 25.11.6, 25.17, 25.18, Section 26.3.2; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June 2016

“Signed and sealed”

Charles Masala, P.Eng.

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CERTIFICATE OF QUALIFIED PERSON

I, Stella Searston, RM SME, am employed as a Principal Geologist with Amec Foster Wheeler E&C Services Inc (Amec Foster Wheeler).

This certificate applies to the technical report titled "Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 30 April 2016 (the "technical report").

I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM #111778), a Member of the Australian Institute of Geoscientists (MAIG #2406) and a Registered Member of the Society for Mining, Metallurgy and Exploration (RM SME #4168111).

I graduated from James Cook University in Australia in 1987 with a Bachelor of Science degree in geology, and from the University of Tasmania in 1999 with a Master of Economic Geology degree.

I have practiced professionally since graduation in 1987. In that time I have been directly involved in generation of, and review of, mineral tenure, surface and other property rights, geological, mineralization, exploration and drilling data, geological models, sampling, sample preparation, assaying and other resource-estimation related analyses, quality assurance-quality control, databases, resource estimates, risk analyses, preliminary economic assessment, pre-feasibility, and feasibility studies, and due diligence studies in Australia, Southern Africa, the Pacific and North and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Fruta del Norte Project (the "Project").

I am responsible for Sections 1.2, 1.3, 1.4, 1.5, 1.6, 1.27, 1.28; Sections 2.1, 2.2, 2.3, 2.5, 2.6, 2.7; Sections 3.1, 3.2, 3.3; Section 4; Section 5; Section 23; Sections 25.1, 25.2, 25.18; Section 26.1; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have no previous involvement with the Project.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

"Signed and stamped"

Stella Searston, RM SME

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Klohn Crippen Berger

CERTIFICATE OF QUALIFIED PERSON

I, Bryan D. Watts, P.Eng, am employed as the Chairman and Principal with Klohn Crippen Berger Ltd. (KCB). This certificate applies to the technical report titled “Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 30 April 2016 (the “technical report”).

I am a registered Professional Engineer (P.Eng.) in the Provinces of British Columbia (13125) and Alberta (24608), and am registered as a Professional Geoscientist (P.Geo.) in the Province of British Columbia (13125). I graduated with a Bachelor of Applied Science degree from the University of British Columbia in 1974, and with a Master of Science in Civil Engineering degree from the University of Alberta in 1981.

I have practiced my profession for 42 years since graduation. I have been directly involved in geotechnical engineering in tailings and water dam design and construction, including construction monitoring, field investigations, design engineering, and participation on review boards for a wide variety of projects. I have worked in Canada, USA, PNG, Peru, Spain, Jamaica, Guyana, Chile, Honduras and Brazil.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Fruta del Norte Project (the “Project”) between 16 to 18 October, 2015.

I am responsible for Sections 1.19.4, 1.21, 1.22, 1.27, 1.28; Sections 2.3, 2.4, 2.5, 2.6.1, 2.6.3; Sections 3.1, 3.2, 3.3; Section 18.5; Sections 20.4, 20.11.3; Sections 21.1.2, 21.1.7, 21.2.1, 21.2.4, 21.3; Section 24.3; Sections 25.11.4, 25.13, 25.14, 25.17, 25.18; Section 26.3.5; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since 2015 as part of preparation of the feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June 2016

“signed and sealed

Bryan D. Watts., P.Eng.

CERTIFICATE OF QUALIFIED PERSON

I, Alejandro Sepúlveda, RM CMC, am employed as a Principal, Project Director, with NCL Ingeniería y Construcción SpA (NCL).

This certificate applies to the technical report titled “Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 30 April 2016 (the “technical report”).

I am a Registered Member of the Comisión Calificadora de Competencias en Recursos y Reservas Mineras (Chilean Mining Commission; RM CMC). I graduated from the School of Mine Engineering, Universidad de Chile.

I have practiced my profession for 35 years. I have been directly involved in the mine design and planning of the Fruta del Norte Project.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Fruta del Norte Project (the “Project”) between 20–23 June, 2015.

I am responsible for Sections 1.14, 1.15, 1.16, 1.19.2, 1.19.3, 1.19.5, 1.19.6, 1.21, 1.22, 1.26, 1.27, 1.28; Sections 2.3, 2.4, 2.5, 2.6.1, 2.6.5; Sections 3.1, 3.2; Section 15; Section 16; Sections 18.2, 18.3; 18.4, 18.6, 18.14; Sections 20.2, 20.3, 20.5, 20.6, 20.11.2, 20.11.4; Sections 21.1.1, 21.1.2, 21.1.3, 21.1.5, 21.1.6, 21.1.7, 21.2.1, 21.2.2, 21.2.6, 21.3; Sections 24.2, 24.3; Sections 25.7, 25.8, 25.11.2, 25.11.3, 25.11.5, 25.13, 25.14, 25.17, 25.18; Section 26.3.3; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since 2014 as part of preparation of the feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

“Signed”

Alejandro Sepúlveda, RM CMC

MM Consultores

CERTIFICATE OF QUALIFIED PERSON

I, Anthony (Tony) R. Maycock, P.Eng, am employed as the Senior Consultant with MM Consultores Limitada (MM Consultores).

This certificate applies to the technical report titled “Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study” that has an effective date of 30 April 2016 (the “technical report”).

I am a registered Professional Engineer (P.Eng.) in the Province of British Columbia (#13275). I graduated with a Bachelor of Science (Honours) degree in 1969 from the University of London, England.

I have practiced my profession for 47 years since graduation. I have been directly involved in mineral processing plant operations, design and commissioning and project management for large mining projects and associated infrastructure. I have worked in Zambia, Canada, USA, Chile, Argentina, Peru and Brazil.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Fruta del Norte Project (the “Project”) between 7 and 8 April 2016.

I am responsible for Sections 1.18, 1.21, 1.22, 1.25, 1.27, 1.28; Sections 2.3, 2.4, 2.5, 2.6.1, 2.6.4; Sections 3.1, 3.2, 3.3; Sections 18.1, 18.7, 18.8, 18.9, 18.10, 18.11, 18.12, 18.13, 18.14; Sections 21.1.1, 21.1.2, 21.1.3, 21.1.4, 21.1.5, 21.1.6, 21.1.7, 21.2.1, 21.2.4, 21.2.6, 21.3; Section 24; Section 25.10, 25.13, 25.14, 25.16, 25.17, 25.18; Sections 26.2, 26.3.6; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as independence is described by Section 1.5 of NI 43–101.

I have been involved with the Project since 2014 as part of preparation of the feasibility study.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

“Signed and sealed”

Anthony R. Maycock, P.Eng.

MM Consultores Limitada
El Chagual 2010, Las Brisas de Chicureo
Colina RM
Chile
Tel: 56 99 419 5242

CERTIFICATE OF QUALIFIED PERSON

I, David A Ross, P.Geo, am employed as Director, Resource Estimation, and Principal Geologist with Roscoe Postle Associates Inc. (RPA).

This certificate applies to the technical report titled Fruta del Norte Project, Ecuador, NI 43-101 Technical Report on Feasibility Study that has an effective date of 30 April 2016 (the "technical report").

I am registered as a Professional Geoscientist (P.Geo.) in the Province of Ontario (#1192). I graduated from Carleton University, Ottawa, Ontario, Canada, in 1993 with a Bachelor of Science degree in Geology and from Queen's University, Kingston, Ontario, Canada, in 1999 with a Master of Science degree in Mineral Exploration.

I have worked as a geologist for a total of 20 years since my graduation. My relevant experience for the purpose of the technical report is:

- Mineral Resource estimation work and reporting on numerous mining and exploration projects around the world.
- Exploration geologist on a variety of gold and base metal projects in Canada, Indonesia, Chile, and Mongolia.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 Standards of Disclosure for Mineral Projects (NI 43–101).

I visited the Fruta del Norte Project (the "Project") between 7 and 9 April, 2016.

I am responsible for Sections 1.7, 1.8, 1.9, 1.10, 1.12, 1.13, 1.27, 1.28; Sections 2.3, 2.4, 2.5, 2.6.1, 2.6.6; Sections 3.1, 3.2, 3.3; Section 6; Section 7; Section 8; Section 9; Section 10; Section 11; Section 12; Section 14; Sections 25.3, 25.4, 25.6, 25.18; Section 26.3.1; and Section 27 of the technical report.

I am independent of Lundin Gold Inc. as is described in Section 1.5 of NI 43–101.

I have been involved with the Project since 2015 as part of preparation of the feasibility study and I have previously been a co-author on a technical report entitled:

- Evans, L., Ross, D., and Scholey, B., 2014: Technical Report On The Mineral Resource Estimate, Fruta Del Norte Project, Ecuador: technical report prepared by RPA for Fortress Minerals Corp., effective date 21 October, 2014.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: 15 June, 2016

(Signed and sealed) "*David A. Ross*"

David A Ross, P.Geo.

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for Lundin Gold Inc. (Lundin Gold) by Amec Foster Wheeler Americas Ltd and Amec Foster Wheeler E&C Services Inc (collectively Amec Foster Wheeler), Klohn Crippen Berger Ltd (KCB), MM Consultores Limitada (MM Consultores), NCL Ingeniería y Construcción SpA. (NCL), and Roscoe Postle and Associates Inc. (RPA), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Lundin Gold subject to the respective terms and conditions of its contracts with the individual Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1.0 SUMMARY

1.1 Key Outcomes

Table 1-1: Key Outcomes

Project economics at a gold price of US$1,250/oz and a silver price of US$20/oz			
Item	**Units**	**Pre-tax**	**After-tax**
Net present value at a 5% discount rate (NPV_5)	US$ million	1,283	676
Internal rate of return (IRR)	Percent	23.8	15.7
Capital payback after commencement of production	Years	3.7	4.5

Cashflow (US$ million)	**2020**	**2021**	**2022**	**Average Years 1–10**	**Life-of-Mine**
Doré revenue	62	121	151	133	1,669
Concentrate revenue	117	247	314	280	3,631
Total revenue	179	368	465	414	5,301
Operating costs	107	151	149	147	1,961
Operating profit	72	216	316	267	3,339
Taxes and royalties	16	(6)	16	59	914
Capital cost estimate	139	16	11	28	975
Changes in working capital	46	8	11	6	—
Cash flow (after tax)	(129)	198	279	174	1,449

Sensitivity analysis (US$100/oz variation from the base case gold price; silver held at US$20/oz)				
Item	**Units**	**US$1,150 oz Au**	**Base Case US$1,250 oz Au**	**US$1,350 oz Au**
NPV_5	US$ million	506	**676**	844
IRR	Percent	13.4	**15.7**	17.8
Payback	Years	5.0	**4.5**	4.2

1.2 Introduction

Amec Foster Wheeler Americas Ltd. (Amec Foster Wheeler) was requested by Lundin Gold Inc. (Lundin Gold) to compile an independent NI 43-101 Technical Report (the Report) for the Fruta del Norte (FDN) Project (the FDN Project or the Project) located in Ecuador.

The firms and consultants who are providing Qualified Persons (QPs) responsible for the content of this Report, which is based on a feasibility study completed in 2016 (the 2016 FS) and supporting documents prepared for the 2016 FS, are, in alphabetical order, Amec Foster Wheeler Americas Ltd and Amec Foster Wheeler E&C Services Inc (collectively Amec Foster Wheeler), Klohn Crippen Berger Ltd (KCB), MM Consultores Limitada (MM Consultores), NCL Ingeniería y Construcción SpA (NCL), and Roscoe Postle and Associates Inc. (RPA).

The FDN Project is wholly-owned by Lundin Gold, through its indirectly-held subsidiary, Aurelian Ecuador S.A. (Aurelian).

1.3 Terms of Reference

The Report will be used in support of Lundin Gold's press release dated 6 June, 2016, that is entitled "Lundin Gold Announces a Positive Feasibility Study for the Fruta del Norte Project and Conference Call".

Currency is expressed in US dollars unless stated otherwise; units presented are metric units, such as metric tonnes, unless otherwise noted.

Calendar years are used in some sections of the Report, in relation to the proposed mine plan and execution plan. The years shown are for illustrative purposes; the actual timing may vary. Formal approval is required from the Lundin Gold Board for mine construction, and additional permits need to be granted by the Government of Ecuador.

1.4 Project Setting

The nearest city to the Project area is Loja, the fourth-largest city in Ecuador. The Project is situated about 139 km east–northeast of Loja.

Vehicular access from Loja to the FDN site is via a 150 km long paved highway (Highway 45) to the town of Los Encuentros. A 40 km long gravel road connects Los Encuentros to the Project site.

The Cordillera del Cóndor consists of heavily dissected, steep ridges that rise from the Zamora River and Nangaritza River valleys (about 850 masl) to sharp ridges and flat-topped mesas, up to 2,400 masl, which lie along the border with Peru. The majority of the Project area, including the La Zarza concession, lies in the highlands south of the Zamora River, and east of the Nangaritza River, both of which flow into the Amazon River drainage system. Tropical rain forest canopies most of the region except where cleared for agriculture in the river valleys and adjacent slopes.

As a result of the Project location near the equator and moderate elevation of 1,400 masl, daily average temperatures are fairly constant at approximately 16ºC. Annual precipitation is about 3,000 mm. Lower average daily temperatures and higher monthly rainfalls prevail at higher elevations such as the La Zarza concession. Currently some exploration activities may be curtailed during the rains.

Lundin Gold expects that any future mining activity will be conducted year round.

1.5 Mineral Tenure, Surface Rights, Royalties and Agreements

Lundin Gold's mineral tenure holdings currently comprise 31 mining concessions that cover an area of approximately 74,855 ha. The FDN deposit is hosted in the La Zarza concession. In addition to the mining tenures, Lundin Gold holds two construction materials concessions that total about 237 ha.

Lundin Gold's concessions in Ecuador are held in the name of Aurelian. The concessions were originally issued under Ecuador's old mining laws with a 30-year term. With the reformation of the country's mining laws in 2009, Lundin Gold's concessions were registered in the Mining Registry and now have different expiry dates, ranging from 21–23 years from the date of registration.

Under the current Mining Law, a concession's term is divided into two stages: exploration and exploitation. The exploration stage is further subdivided into shorter phases based on the achievement of stipulated milestones. Obligations that must be met to retain the concessions include payment of annual conservation fees, completion of annual reports on exploration completed, and proposed investment plans. Any failure to achieve these milestones and successfully advance to the next stage by the deadline can result in a forfeiture of the concession.

In the final stage of exploration, referred to as the economic evaluation stage, an application for exploitation can be made to the Government of Ecuador. If successful, a concessionaire can then enter into an exploitation agreement with the Government of Ecuador, and the concession term is the one negotiated under the agreement.

The majority of the concessions form a large contiguous block that extends from the Nangaritza River eastward to the international border with Peru.

Surface rights must be obtained to support mining project development either through the land acquisition or by an easement (agreed with the land titleholder or imposed by the Ministry of Mining). To date, 60 public deeds for required surface rights have been signed, which cover a collective area of approximately 4,118.5 ha. At the Report effective date, one public deed remains in negotiation for an area of approximately 40 ha. Lundin Gold has advised Amec Foster Wheeler that there are sufficient surface rights currently acquired to support construction and development of the planned mining-related infrastructure.

Seven land easements have been concluded; these cover areas including the access road and construction of surface infrastructure to support mining activities. The term granted is equivalent to the duration of the La Zarza concession term, or the Definitive Exploitation Agreement terms and its extensions (see next section).

One concession easement agreement has been concluded with Cóndor Gold S.A. (Cóndor Gold), to support construction and operation of the access road. The easement agreement is valid for as long as the underlying mining concessions held by Cóndor Gold remain current.

Lundin Gold holds seven water rights under a number of water tenures that collectively allow for 97.25 L/s of extraction. Six rights were granted for exploration purposes, and one water right allows for human consumption. The 2016 FS envisages that Lundin Gold will not be applying for an overall water permit for industrial usage for mining activities, since the water that is proposed to be used will be from secondary, not primary, sources.

A 1% net revenue royalty is payable in perpetuity on production from Lundin Gold's current mining concessions, including the La Zarza concession, under a royalty agreement dated November 16, 2007 among Lundin Gold's subsidiaries (Aurelian Resources Inc., Aurelian Resources Corporation Ltd., and Aurelian Ecuador S.A.) and two individuals, being Keith M. Barron and Patrick F.N. Anderson.

There are no other third party royalties, back-in rights, payments, or other encumbrances in favour of Lundin Gold or Aurelian.

1.6 Definitive Exploitation and Other Agreements

During 2015, Lundin Gold, through Aurelian, and the Government of Ecuador worked collaboratively to establish the fiscal terms and conditions for the development of the FDN Project. At the start of 2016, Lundin Gold announced that it had completed negotiations with the Government of Ecuador and had settled the Definitive Exploitation Agreement terms for the Project. Key features from Lundin Gold's perspective include:

- The right to develop and produce gold from the FDN Project for 25 years; this right can be renewed
- An advance royalty payment of US$65 million to the Government of Ecuador, with $25 million being due upon execution of the Definitive Exploitation Agreement
- A royalty equal to 5% of net smelter revenues from production, payable to the Government of Ecuador
- An extraordinary revenue tax (the Windfall Tax) will be calculated in the event that market prices exceed a stipulated base price for gold and for silver; the Windfall Tax will not apply until Aurelian has recouped all of the cumulative investment in the development of the FDN Project since its inception plus the present value of the actual cumulative investment incurred from signing of the Definitive Exploitation Agreement until the start of production

- The Government of Ecuador's share of cumulative benefits derived from the FDN Project will not be less than 50%. To the extent that the Government of Ecuador's cumulative benefit falls below 50%, Aurelian will be required to pay an annual sovereign adjustment. Each year, the benefits to Aurelian will be calculated as the net present value of the actual cumulative free cash flows of the FDN Project subsequent to the signing of the Definitive Exploitation Agreement, net of the cumulative investment incurred in the development of the Project from its inception until the date of the Definitive Exploitation Agreement. The Government of Ecuador's benefit will be calculated as the present value of the cumulative sum of taxes paid including corporate income taxes, royalties, Windfall Tax, labour profit sharing paid to the State, non-recoverable value-added tax (VAT; IVA in the Spanish acronym), and any previous sovereign adjustment payments.

Lundin Gold expects Aurelian will sign the Definitive Exploitation Agreement within six months of the FDN Project being approved to move to the Exploitation Stage under the mining legislation.

Coincident with the signing of the Definitive Exploitation Agreement, Lundin Gold expects Aurelian will enter into an investment protection agreement with the Government of Ecuador, the objective of which is to provide legal and fiscal stability and protection to Aurelian for its investment in the Project. At the Report effective date, Lundin Gold was in negotiations with the Government regarding the terms of this agreement.

1.7 Geology and Mineralization

The FDN deposit is located within a 150 km long copper–gold metallogenic sub-province located in the Cordillera del Cóndor region. The deposit is hosted by andesites of the Misahuallí Formation and feldspar porphyry intrusions.

The FDN deposit is an intermediate-sulphidation epithermal gold–silver deposit measuring approximately 1,670 m along strike, 700 m down dip and generally ranging between 150 m and 300 m wide. The top of the deposit is located beneath approximately 200 m of post-mineralization cover rocks of the Suárez Formation and Hollín Formation. The eastern and western limits of the deposit are defined by two faults that together form part of the Las Peñas fault system that is thought to control the gold–silver mineralization. The southern limit of the mineralization along the fault system has not been fully defined by exploration activities.

Mineralization is characterized by intense, multi-phase quartz–sulphide ± carbonate stockwork veining and brecciation over broad widths, typically between 100–150 m wide in the coherent central and northern parts of the system where the gold and silver grades are highest. Mineralized shoots are typically present within dilatant zones developed along inflections of vein strike or dip where the geometry permits maximum opening at the time of mineralization.

The mineralogy of FDN consists of chalcedonic to crystalline quartz, manganese-carbonates, calcite, adularia, barite, marcasite, and pyrite, as well as subordinate sphalerite, galena, and chalcopyrite, and traces of tetrahedrite and silver sulphosalts. The bulk of the gold is microscopic and associated with quartz, carbonates and sulphides. Much of the gold is free milling, but the mineralization is moderately refractory, with approximately 40% of the gold locked in sulphides. However, coarse visible gold is commonly observed. Individual gold grains range from discrete specks less than 0.1 mm in size to broccoli-like, arborescent crystals >10 mm across. Visible gold occurs in all mineralized zones, in quartz or carbonate, as well as within pyrite or silver sulphosalt clusters.

Exploration has delineated a number of additional epithermal-style targets and prospects.

In the opinion of the QPs, the knowledge of the deposit settings, lithologies, mineralization style and setting, ore controls, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

1.8 History

The Cordillera del Cóndor was first explored by Spanish conquistadors in the 1500s. There is evidence that pre-Columbians mined both hard rock and alluvial gold in the area. Spanish mining activity ceased about 1620, following conflict with local Indian tribes that had been enslaved to work in the mines. Artisanal alluvial miners began to prospect the Cordillera del Cóndor as early as 1935, both in Peruvian and Ecuadorian territory.

Companies involved prior to Lundin Gold's Project interest included Minerales del Ecuador S.A. (Minerosa), from 1986–1992; Amlatminas S.A. (Amlatminas) from 1996–2002; Minera Climax del Ecuador (Climax), a subsidiary of Climax Mining Ltd. of Australia from 1996–1998; Aurelian Resources Corporation Ltd. (Aurelian Resources) from 2003–2008; and Kinross Gold Corporation (Kinross) from 2008–2014.

Completed activities have included stream sediment, rock chip, grab, soil and trench sampling, reconnaissance exploration, geological and structural mapping, ground and airborne geophysical surveys, genesis and modelling studies, core drilling, metallurgical testwork, Project design studies, and preliminary marketing assessments. Kinross completed a pre-feasibility study in 2009 (2009 Kinross PFS), and a feasibility study in 2011 (2011 Kinross FS). Lundin Gold undertook a feasibility study in 2015–2016 (2016 FS), the results of which are documented in this Report.

No commercial production has occurred from the Project; however, there have been periods of active artisanal mining within the Project boundaries.

1.9 Drilling and Sampling

Drilling completed within the Project area to 1 December 2015 totals 479 core holes (171,831.03 m). Within these programs, the drill campaigns completed on the La Zarza concession between 1997 and 1 December 2015 consisted of 438 holes (162,200 m) completed at the FDN deposit, on areas with potential to host infrastructure, and on a number of exploration prospects within the La Zarza concession. A total of 284 holes (126,708 m) were completed at the FDN deposit. No drilling occurred on the Project between 1 December 2015 and 25 April 2016.

A new exploration drill program that commenced on 26 April 2016 is focused on key exploration targets outside of the La Zarza concession, and is provisionally envisaged as consisting of 20–30 drill holes (7,500–10,500 m), depending on results. As of 24 May 2016, six core holes had been completed on exploration targets within the Princesa and Emperador 1 concessions, for a total of approximately 2,217 m. Assay results are pending.

Drilling has been by core methods. Core sizes drilled include HQ (63.5 mm core diameter) and NQ-sized core (47.6 mm) for exploration purposes, and lesser diameter HQ3–NQ3 (for geotechnical purposes), NTW (56 mm) and BTW (42 mm) core sizes.

Following arrival at camp, the core was photographed, recovery was measured, and the core was geotechnically logged. Lithological logging followed with the geologist recording a detailed description of the lithology, texture, alteration, mineral assemblage and intensity and level of oxidation/weathering. A graphic log column with a sketch of the geology was also included.

Drill recoveries were acceptable. Lower recoveries during the 2015 drilling (with respect to previous programs) may in part be due to the number of the 2015 drill holes drilled to the west of the FDN deposit, and others drilled outside the FDN deposit to better define known fault zones where lower core recovery and drill hole problems could be expected.

During the 2005 to 2007 drill programs, drill hole collars were located by professional Ecuadorian surveyors using a Total Station survey instrument. During the same programs, the existing Climax drill collars were surveyed, where they could be located. Drill holes completed since 2009 were surveyed by Aurelian–Kinross or Lundin Gold personnel using Total Station survey instruments. As part of the quality assurance and quality control (QA/QC) process, at the end of the Lundin Gold drill program the local engineering firm Leiva was contracted to survey the drill collars using differential double frequency GPS equipment.

Core holes from the Climax programs were surveyed by either acid tests or Tropari tests. The initial Aurelian–Kinross programs used a Sperry Sun or Tropari single shot survey instrument taking a measurement every 50 m, or a Flexit digital multi-shot survey instrument with a reading every 30 m down the drill hole. Later programs used Flexit and Reflex digital multi-shot survey instruments. For the 2015 Lundin Gold drilling program, a Reflex EZ-TRAC digital down hole survey instrument was used.

The deposit was systematically drilled out on 50 m to 100 m sections between lines 2500N and 3900N. The grade and mineralization intensity characteristics clearly delineated zones of high grade and high tonnage mineralization in the north versus more disperse, albeit locally high grade mineralization, in the south. Infill drilling on 50 m centres was focused over 300 m of strike between 3300N and 3600N. The drilling tactic typically involved fan drilling from the pad collar to facilitate between 50 m and 25 m infill drilling before stepping out across strike to define the up or down dip geometry. Even though the majority of Aurelian core holes were drilled with an easterly (approximately 090°) azimuth and the dominant dip of the mineralized system is west, no single method or percentage adequately describes the complex relationship between down hole (core) length and the true width of the intersected mineralized zones. Drill hole inclinations vary significantly (from -45° to -84°) and the mineralized zones have variable dips from moderate to steep westerly to steep easterly dips. Therefore, most drill holes intersect the mineralized zones at an angle, and the drill hole intercept widths reported for the Project are greater than true widths.

The density determination methodology consisted of the water-displacement method. Measurements were made from every hole at an interval rate of approximately 50 m in unmineralized rock and every 20 m in the mineralized system. Rock density is relatively constant within specific lithologies and shows only minimal variation between different lithological groups.

During the Climax drill program, core was sawn in half and sampled at 2 m intervals, regardless of geology. Each sample consisted of 2 m composites of half core, with the exception of the first and last intervals in each hole. Aurelian–Kinross and Lundin Gold core was sampled using the following criteria:

- Maximum sample length of 2 m in un-mineralized lithologies
- Maximum sample length of 1 m in mineralized lithologies
- Smaller samples may be selected around high-grade, visible gold-bearing veins
- Minimum sample length of 20 cm.

Drill core was split along the long axis using core saws. Areas of very soft rock were cut using a machete and sections of very broken core were sampled using spoons. The right hand side of the core was always sampled.

A number of independent laboratories were used during the core drilling exploration and delineation phases. Sample preparation facilities included ALS Quito, Inspectorate Quito, and SGS Santiago; analytical facilities included ALS Vancouver, ALS Lima, Inspectorate Lima, SGS Toronto, and SGS Antofagasta.

Sample preparation included drying the sample, crushing to initially >70% passing -2 mm, and later changed to 90% passing, then pulverizing to better than 85% (90%) passing -75 µm in the initial programs, which changed to pulverizing to better than 90% passing 100 µm.

For the ALS primary laboratories, gold was determined by 30 g or 50 g fire assay with an inductively-coupled plasma (ICP) atomic emission spectroscopy (AES2) finish, method code AU-ICP21 (AU-ICP22). The principal gold determination method was changed to method code Au-AA24 from drill hole CP-07-98 to BLP2130e01 (end of 2012), which applied an atomic absorption spectroscopy (AAS) finish following a 50 g fire assay. If gold assays greater than 10 g/t were detected using either of the above techniques, then over-limit re-assays were completed using a 50 g fire assay with a gravimetric finish, method code AU-GRA22. Multi-element analysis was performed on samples from 2006 to 2012 using method code ME-ICP41, which resulted in a 34-element package, including silver, with a nitric aqua regia acid digestion, and ICP-AES2 finish. Over-limit re-assays were run on selected drill holes for silver, zinc, lead and copper if Ag >100 ppm, Zn >10,000 ppm, Pb >10,000 ppm or Cu >10,000 ppm. Over-limit re-assays were completed using an aqua regia acid digestion and AAS 3 finish (method code AA46).

The quality control (QC) program implemented has varied considerably over time in terms of the frequency of insertion and the source of the certified reference materials (CRMs) or standard reference materials (SRMs). Programs typically included submission of blank samples, CRMs/SRMs, field and reject duplicates and pulp check assaying. Ongoing monitoring of the program was performed by the operators, with spurious results being investigated and changes implemented when required.

The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs conducted by Aurelian–Kinross and Lundin Gold are sufficient to support Mineral Resource and Mineral Reserve estimation. Sample collection, sample preparation, analytical methods and sample security for all Aurelian–Kinross and Lundin Gold drill programs are in line with industry-standard methods for epithermal gold–silver deposits and can support Mineral Resource and Mineral Reserve estimates.

1.10 Data Verification

At the end of the 2009 and 2010 infill programs, Kinross–Aurelian site personnel compiled and checked all certificates against the database for all elements. The comparison showed no errors. Kinross also did a manual 5% check of the 2010 drill assay data on site in June 2010. No errors were identified.

RPA performed database audits in support of Mineral Resource estimates in 2009, 2014, 2015, and 2016, and in support of compilation of a technical report in 2014. Data verification activities included detailed reviews of the standard operating protocols, drill hole spacings, core diameters used, how the final collar coordinates were determined, down hole surveying procedures, drill core logging protocols, core recovery, collection of bulk density data, sample layout, sample preparation and sample security procedures, and QA/QC protocols. During site visits in 2009, 2014, and 2016, RPA reviewed drill core from numerous drill holes and compared observations with assay results and descriptive log records made by Aurelian and Kinross geologists. In addition to reviewing core, RPA examined outcrops, drill rigs, sampling procedures and other general exploration protocols.

RPA is of the opinion that database verification procedures for the Project comply with industry standards and are adequate for the purposes of Mineral Resource and Mineral Reserve estimation.

1.11 Metallurgical Testwork

Metallurgical testwork commenced in 2006. Initial testwork and Project design by Aurelian and Kinross focused on a pressure oxidation (POX) flowsheet. Prior to the 2015–2016 metallurgical programs, Kinross conducted a metallurgical program to assess the potential of a flowsheet to produce a saleable concentrate in conjunction with the production of doré from cyanidation of a gravity concentrate and flotation tailings. This work assessed the differences between a gravity, flotation, leach (GFL) versus a gravity, leach, flotation (GLF) flowsheet. The outcome of the testwork indicated that the GFL flowsheet was the preferred option due to improved metal recoveries and lower capital and operating costs.

Amec Foster Wheeler reviewed the Kinross data and, due to the capital costs associated with a POX plant, concurred with the GFL flowsheet approach. As a result, much of the initial POX-related testwork is not relevant to the current design; however, a description of the work is included in Section 13 for completeness.

During the 2016 FS, the Early MET, FDN MET1 (MET1) and FDN MET4 (MET4) testwork programs were carried out under the supervision of Amec Foster Wheeler. Metallurgical testwork programs were completed at SGS Minerals S.A. in Santiago, Chile for Met 1 and at SGS Lakefield in Ontario, Canada, for Early MET and MET4 programs. The results of each testwork program were independently reported by each SGS laboratory. While the Early MET program provided early confirmation of the GLF flowsheet, the MET1 and MET4 programs provide the basis of the new design.

Physical characterization testwork was carried out on selected drill core intervals for both the MET1 and MET4 programs. The characterization work included semi-autogenous grind (SAG) comminution (SMC) testing, and Bond ball mill work indices (BWi). In total, 24 MET1 and 14 MET4 samples were submitted for SMC testing and representative samples of each MET1 composite were submitted for Bond ball mill work indices. Based on the individual SMC results, the orebody can be classified as moderately hard in comparison to the Julius Kruttschnitt Mineral Research Centre (JKMRC) database. These results remain consistent with the previous testwork programs and historical data on the deposit.

Both MET1 (composite and variability) and MET4 samples were submitted to gravity concentration using laboratory scale Knelson concentrators. The Knelson concentrator feed size was approximately 150 µm for both MET1 and MET4 programs. The amount of gold that potentially can be recovered by gravity in this deposit is considered high, as supported by the global recovery results of the gravity testwork and automated scanning electron microscopy (QEMScans) of the head feed. Of note is the additional recovery of silver, suggesting that a large proportion of free gold is in the form of electrum. Leaching characteristics of the gravity concentrates were also investigated. Gold extraction rates were found to be consistent with industry norms. MET1 and MET4 composite samples tested achieved between 94% and 98% leached gold recovery from the gravity concentrates produced.

A sulphide flotation test program was developed for the production of a gold- and silver-rich concentrate, knowing that flotation tailings would be subsequently cyanide leached. The objective of the flotation circuit was to recover fine free gold and gold associated sulphides to produce a saleable concentrate. During the MET1 program each variability sample was subjected to an open circuit (OCT) flotation test to determine the optimal flotation conditions. Subsequently, the MET1 composite sample and MET4 sample were submitted to locked cycle test (LCT) at the optimal conditions, using the same flowsheet. All samples tested reported only moderate gold recoveries. The overall flotation process requires lengthy residence time and relatively high reagent dosage as a result of the middlings gold being a combination of sulphide and quartz associations. Analysis of the flotation tailings indicates, fine free gold, gold associated sulphide and gold associated quartz occlusions, which cannot be recovered by conventional sulphide flotation. Final concentrates showed reasonable gold and silver grades, with mid-level impurities. Overall, the concentrates produced are considered suitable for sale to a smelter for further processing.

Bottle roll leaching tests were performed on each variability and composite sample (including MET4). During the MET1 composite testing, kinetic studies were carried out using air and oxygen injection methods. In addition a pre-oxidation stage was tested to determine the optimal leaching conditions. Kinetic testing of each composite showed negligible difference between using air, oxygen and pre-oxidation. Ultimate leach recoveries between 51.3% and 64.4% were obtained after 24 hours of leaching. The MET4 program leaching results confirmed the ultimate recoveries obtained in the MET1 program. Cyanide consumption during the leach tests was low due to the recovery of sulphides to the concentrate during the flotation stage.

Additional testwork in support of the plant and process design included cyanide detoxification testing, using the Inco SO_2/air process, and settling testwork on detoxified MET1 tailings composite samples to determine the optimal flocculant dosage and corresponding settling rate.

The metallurgical testwork completed to-date is based on samples which adequately represent the variability of the proposed mine plan.

Gold recovery relationships were developed for the flotation circuit (grade/recovery curves) and for the total number of gold units reporting as doré (via gravity recovery and flotation tailings carbon-in-leach (CIL)). All recovery relationships are bounded by the condition of the Au–S ratio of the flotation feed ≤10 g/t Au:% S. The boundary was checked against the monthly reported grades and resulting Au–S ratios of the feasibility study mine plan. All monthly values reported in the mine plan were found to fit within this boundary.

Current recovery estimates are based on the MET program testwork results. The life-of-mine (LOM) plan (LOMP) average gold metallurgical recovery is set at 91.7%. Actual gold recoveries are expected to range between 91.4–92.1%, peaking in 2023 when high-grade ore is processed, then reducing until 2031. The recovery projections increase again for the last two years of operations.

The two products of the plant, the concentrate and the doré, are considered saleable without major penalties. The level of arsenic and mercury in the flotation concentrate is expected to be able to be maintained at acceptable levels.

1.12 Mineral Resource Estimation

A total of 246 drill holes support the estimate. There was no drilling on the project in the years 2013 to 2014 inclusive. Assay results from the 2015 drilling were not available at the time of the resource estimate update. Therefore, the most recent drill holes used to estimate Mineral Resources were drilled in 2012, and the effective date of the current Mineral Resource model is 1 December, 2015.

Logged rock types were grouped into one of 13 lithological units. These units were then divided into four main geological domains based on lithology, alteration and grade criteria. Each domain is distinctive in mineralogical, textural and geochemical character as well as in gold distribution.

The four zones are believed to represent distinct hydrothermal events starting with the Xp_Ip domain, which is associated with late porphyry events. This was followed by the silica–(arsenopyrite)–marcasite alteration associated with hydrothermal brecciation (Xh) in the up-flow zone centred on section 3400N and “mushrooming” out below the Suárez Formation unconformity. The later-stage quartz–carbonate phase (Vn) appears to have formed in the northern section of the deposit, wrapping partially around a flexure in the feldspar porphyry contact. Xh and Vn were grouped together for resource domaining purposes.

Four models were constructed:

- SRK produced a new lithological model following the 2015 geotechnical and hydrological drilling program using Leapfrog Geo software. The lithological modelling program was an iterative process conducted concurrently with updating the structural model. As far as possible, observed fault kinematics and offsets were preserved within the lithological model. The interpretation was based on the integration of topographic, and drill hole data, and on direct observations made during two site visits. For the interpretation, Lundin Gold supplied SRK with assay and lithological data for 440 core holes, geophysical data, and light detection and ranging (LiDAR) topographic data
- The current fault model is a revision of a fault model produced by SRK for the FDN deposit in 2013. Information supporting the model includes site observations, selected drill core data (assay and lithological data; oriented geotechnical data; observations of core box photos), a Leapfrog geological model (FDN6) created by Kinross and maintained by Lundin Gold, and LiDAR topographic data. A total of 14 faults were modelled and the fault model was used to define fault blocks for subsequent lithological modelling. Several iterations were constructed to refine the structural and lithological models
- A degradation 3D model was constructed using Leapfrog Geo software. In general, degraded zones are contained within the Suárez Formation conglomerate and its interbedded tuff beds. Degradation also occurs to a lesser extent in the siltstone beds in the upper part of the Suárez Formation
- The alteration model was constructed from 18,930 infrared spectra collected from 148 drill holes at a 1.5 m interval for 2015 core holes and one spectrum per box for historical drill holes, and three-dimensional (3D) modelling of the key infrared-active alteration minerals and features.

Leapfrog and GEMS software were used to build the wireframe models representing the domains. Given the selected block size of 4 m by 10 m by 10 m, a 2 m composite was selected for grade interpolation purposes.

The FDN metal capping review consisted of disintegration analysis of the composite values in conjunction with histogram, log probability, and mean variance plots. In order to preserve the grades within the high-grade zones with intense veining of domain Xh_Vn, composites were left uncapped, and instead a restricted search for gold values greater than 60.0 g/t was applied. A capping value was applied to the silver grades for this domain.

The resource database includes 3,511 density measurements. Density data were reviewed by lithology and alteration type. The average values were assigned to the block model to convert volumes to tonnes.

Variography was carried out by Kinross within a 450 m long segment of the deposit with closely-spaced drilling, between northings 9,583,300N and 9,583,750N. Drilling and domains within this segment have not changed significantly since that time, and therefore the variogram models remain current.

Grade interpolations for gold and silver were performed using the ordinary kriging (OK) algorithm and using search strategies individually adapted to each domain. The search ellipses generally have the same orientations, striking north–northeast, dipping west, and plunging north–northeast. A two-pass approach was used, with the first pass search ranges approximately equivalent to the variogram ranges at 80% of the sill. The first pass used a minimum of two drill holes. The second pass used a larger search with a one drill hole minimum. Both hard and soft boundaries were used, based on various contact analyses and the geological interpretation. Pass 1 applied a hard boundary between domains. Pass 2 used a soft boundary between domains. The interpolation parameters for silver were similar to those for gold.

Two block models were constructed in Gemcom GEMS Version 6.7.1 for the interpolation process consisting of a partial model set (percentage model) and a consolidated model. Model validation was performed.

Mineral Resources were classified into the Indicated or Inferred categories based on drill hole spacing and the apparent continuity of mineralization. Variography has suggested a range of 35 m at 75% of the total sill. Infill drilling in 2010 was designed at 35 m spacing. In general, areas of 35 m spacing or shorter were classified into the Indicated category. Other factors that were taken into consideration include the search distance to the nearest composite, estimation by the first pass search ellipse, visual examination and general considerations of drill fan spacings. Parts of the Xh_Vn and Xp_Ip domains were classified as Indicated Mineral Resources. All of the M_South domain was classified as Inferred Mineral Resources. Due to the lack of exposures of mineralization for inspection on the surface or underground, there are no Measured Mineral Resources at this time.

Mineral Resources were reported at a block cut-off grade of 3.5 g/t Au. Silver was not included in the cut-off grade calculation due to its relatively low grade and small contribution to the value of the mineralization.

1.13 Mineral Resource Statement

Mineral Resources are summarized in Table 1-2, and have been classified using the 2014 Canadian Institute of Mining and Metallurgy (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards). The QP for the estimate is Mr. David Ross, an RPA employee.

The Mineral Resource estimate has an effective date of 31 December, 2015. Mineral Resources are reported inclusive of Mineral Reserves at a block cut-off grade of 3.5 g/t Au, assuming underground mining methods. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Factors which may affect the Mineral Resource estimates include: metal price and exchange rate assumptions, changes to the assumptions used to generate the cut-off grade value, changes in local interpretations of mineralization geometry and continuity of mineralization zones, density and domain assignments, changes to design parameter assumptions that pertain to stope designs, changes to geotechnical, mining and metallurgical recovery assumptions, assumptions as to the continued ability to access the site, retain mineral and surface rights titles, obtain environmental and other regulatory permits, and obtain the social licence to operate.

1.14 Mineral Reserve Estimation

The resource block model was provided by RPA (November 2015) and consisted of density, grades, rock types (geometallurgical resource domains), resource category and other impurities.

The geotechnical block model was developed by SRK Consulting (Canada) Inc. (SRK). It utilized assessments of lithology, alteration and structure to model three domains that encompassed Poor, Fair-Poor, and Good–Fair rock mass conditions. This model was built in Leapfrog.

The Mineral Reserve block model was prepared by combining the resource block model and the geotechnical block model.

The models were imported by NCL via ASCII files into DESWIK software. Validation was carried out with 99.9% of the original block model data for Indicated and Inferred Resources in terms of tonnes, gold ounces and silver ounces. The Inferred Mineral Resources grades were set to zero for the purposes of Mineral Reserves estimation.

The mining methods for FDN will be long-hole transverse stoping (TS) in Fair to Good ground, and drift-and-fill (D&F) stoping in Poor ground. Dilution was applied following the geotechnical recommendations. The shape optimizer from DESWIK was used to determine practical mining shapes. The deposit was divided into horizons that were classified both vertically and by mining method.

Table 1-2: Summary of Mineral Resources Inclusive of Mineral Reserves

Category	Tonnage (Mt)	Grade (g/t Au)	Contained Metal (Moz Au)	Grade (g/t Ag)	Contained Metal (Moz Ag)
Indicated	23.8	9.61	7.35	12.9	9.89
Inferred	11.6	5.69	2.13	10.8	4.05

Notes:

1. The Qualified Person for the estimate is Mr. David Ross, P.Geo., an employee of RPA. The estimate has an effective date of 31 December, 2015.
2. Mineral Resources are reported inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are reported at a cut-off grade of 3.5 g/t Au; which was calculated using a long-term gold price of US$1,500/oz.
4. Mineral Resources are constrained within underground mineable shapes that assume a minimum thickness of 2 m; metallurgical recovery of 94%; total operating costs of US$145/t milled (mining cost of US$60/t milled; process costs of US$35/t milled; G&A costs of US$15/t milled; surface infrastructure costs of US$28/t milled; concentrate transport and treatment costs of US$7/t milled); royalties of US$71/oz and selling costs of US$65/oz.
5. Numbers may not add due to rounding.

The top D&F horizons (DF-1245 and DF-1270) will be the crown pillar areas. The D&F south area at elevation 1170 is a high-grade zone within a Poor rock quality domain. The upper part of horizon TS-1080 (stopes between elevation 1155L and 1170L) will be a sill pillar with 15 m high stopes. The bottom horizons (below 1080L) will be mined as D&F. The recommended dimensions for TS are 12 m wide x 20 m long x 25 m high. D&F was designed with vertical cuts of 4 m each.

The dilution material for the TS primary stopes was estimated using the resource block model; dilution material for lateral stopes was assumed to be zero grade on one side and the grade from the resource block model on the other side. The total maximum dilution reaches 16.9% (sill pillar starting stope); for scheduling and reporting purposes the waste dilution is applied (a maximum of 7.7% in sill pillar lateral stopes) so as not to duplicate tonnage because of the stope arrangement. The grade dilution factor applied for TS stopes is a factor by which grades are adjusted because of dilution; in this case the waste reduces the grades because it adds no content for the following elements: gold, silver, mercury, lead, sulphur and antimony. The D&F dilution estimate includes the primary, secondary and tertiary drifts. A grade dilution factor of 95.3% was used for D&F.

Overall, in primary TS stopes, the total mining losses are estimated to be 8.8%, resulting in a mining recovery factor of 91.2%. In secondary TS stopes, the total mining losses are estimated to be 11.9%, resulting in a mining recovery factor of

88.1%. Sill pillar recovery is assumed to be 50%, and for D&F, recovery was assumed to be 100%.

The final LOMP weighted-average dilution applied in the estimation (including TS, D&F and development) is 5.63%. This material is backfill material and is applied at zero grade.

The final LOMP weighted-average mining recovery applied to the estimate is 90.9%.

The grade dilution factor (the factor by which the drop in grade from insitu to diluted and mining recovered is reported) varies depending on the mining method and extraction sequence. The weighted-average grade dilution factor applied (the weighted-average dilution of 5.63% and weighted-average mining recovery of 90.9%) is 95.14%. This is estimated using the ratio of:

(Diluted & mining recovered)/(Insitu).

Two different cut-off grades (COG) have been used, the breakeven COG (BECOG) and the mill COG (MCOG). The BECOG is one of the key parameters needed for mine and stope design. The estimate of BECOG considers mining, processing, royalties and overhead operating costs. The MCOG is applied after the stopes and the accesses are defined, because at this stage there could be some low-grade material that has to be mined and hauled to surface. A decision has to be made whether to send this material to the process plant or to the waste dump. If the material has sufficient grade to pay for processing and other surface costs, it is assumed to be sent to the process plant (the mining cost is considered a sunk cost). A BECOG of 4.7 g/t Au was used for TS and an elevated BECOG of 6.8 g/t Au was used for D&F. A MCOG value of 2.7 g/t Au, excluding the mining costs, was used where production development was already built.

1.15 Mineral Reserve Statement

Mineral Reserve estimates were prepared by Mr. Alejandro Sepúlveda, RM CMC, an NCL employee.

Mineral Reserves have been classified using the 2014 CIM Definition Standards and have an effective date of 30 April 2016.

Mineral Reserves are summarized in Table 1-3.

Table 1-3: Probable Mineral Reserves Statement

Material Source	Tonnes (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)
Long-Hole Stope	8,404	8.97	2,423	10.4	2,813
Drift & Fill	5,533	11.15	1,984	16.9	3,003
Development >4.7 g/t Au	1,158	9.70	361	11.6	434
Development >2.7 g/t Au	394	3.72	47	7.4	94
Total	***15,490***	***9.67***	***4,816***	***12.7***	***6,344***

1. The Qualified Person for the Mineral Reserve estimate is Mr. Alejandro Sepúlveda, RM CMC an NCL employee.
2. Mineral Reserves have an effective date of 30 April 2016. All Mineral Reserves in this table are Probable Mineral Reserves. No Proven Mineral Reserves were estimated
3. Mineral Reserves were estimated using a US$1,250/oz gold price. Mining cost assumptions for transverse stoping (TS) US$61.0/t; mining costs for drift-and-fill (D&F) stoping US$80/t. Other costs and factors common to both mining methods were process and other costs US$75.80/t, dilution factor of 10%, concentrate transport and treatment charges of US$6.70/t. A royalty of US$71.10/oz/t Au and a gold metallurgical recovery of 93.9% was assumed.
4. Gold cut-off grades were 4.7 g/t for TS and 5.3 g/t (elevated to 6.8 g/t) for the D&F.
5. Silver was not used in the estimation of cut-off grades but is recovered and contributes to the revenue stream. The average silver metallurgical recovery is 81.6%. The silver price assumption was US$20/oz.
6. Tonnages are rounded to the nearest 1,000 t, gold grades are rounded to two decimal places, and silver grades are rounded to one decimal place. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces
7. Rounding as required by reporting guidelines may result in summation differences.

Factors that may affect the Mineral Reserves include: long-term commodity price assumptions, long-term exchange rate assumptions, and long-term consumables price assumptions. Other factors that can affect the estimates include changes to: Mineral Resources input parameters, constraining stope designs, cut-off grade assumptions, geotechnical and hydrogeological factors, metallurgical and mining recovery assumptions, and the ability to control unplanned dilution. Operations will require grant of the exploitation licence.

1.16 Mining Methods

The following key considerations influenced the mine design:

- The Project is located in an environmentally sensitive area. Although an open pit mining method or a caving method might be possible, the subsequent impacts were assessed not to be feasible. Hence, selective underground mining was considered for the 2016 FS
- The host rock for the deposit appears competent but the resource zone is less competent with a small portion in Poor rock (less than 10%). Geomechanically, the rock mass quality varies from Poor to Fair (RMR range 40 to 55) with the intact

rock strength averaging 60 MPa. The deposit is also relatively close to surface (within 140 m of surface in some locations)

- Given the variable conditions likely to be encountered, a range of methods and or support regimes was considered appropriate for FDN. The primary methods of extraction selected are TS in the better ground conditions and D&F in the more challenging areas
- Incorporation of backfill to reduce the risk of geotechnical failure and maximize extraction
- Consideration of dewatering requirements and proximity of the Machinaza River.

1.16.1 Geotechnical Considerations

The faults present in the 2015–2016 structural model form a complex network of west–northwest- to northeast-trending, moderate dipping to sub-vertical faults that variably truncate and offset lithology and gold mineralization. Faults generate the widest zones of gouge and breccia where they cross the Suárez Formation. In comparison, faults have well defined margins where they cross the Misahuallí Formation. The West, Central, and portions of the East Fault are significant fault structures that represent a risk to the stability of an open stoping method and subsequently these areas are considered suitable only for a limited man-entry mining method such as D&F where conditions can be well controlled.

Degradation of Suárez Formation conglomerate results in difficult mining conditions that can be mitigated through extraordinary ground support (full shotcrete lining and invert) which will be a high mining cost with slow advance rates. The mine layout has been optimized to avoid intersecting the Suárez Formation.

Stress measurements are not currently available for FDN. In the absence of this information, a stress regime based on SRK's evaluation of the structural geological setting and the World Stress Map have been used to provide a range of estimates. The ground stress is relatively low based on the shallow depth, and rock damage due to higher mining induced stress concentrations is only anticipated in high extraction or sequence closure areas, and weaker rock mass areas. However, reduction in the mining stresses around excavations is likely to adversely affect the stability of large open span areas. Tensile failure and gravity induced unravelling are foreseen as the main failure mechanisms.

The FDN deposit is in a structurally complex, clay-altered, porphyry environment, adjacent to a river. Rock mass conditions in the infrastructure and production areas vary from Poor to Fair quality (RMR 20 to 60) with the poorest conditions present within major structures that run longitudinally through and bound to the deposit. Outside of these fault areas, rock mass conditions are generally Fair (RMR 40 to 60;

intact rock strength 50 to 70 MPa); however, localized zones of Poor ground potentially associated with secondary structures or locally elevated alteration intensity are present throughout the planned mining area.

Excavation stability assessments were completed using industry-accepted empirical relationships, with reference to analogue operational mines where possible. The rock mass conditions in the Poor to Fair and Good domains are considered suitable for open stoping mining methods. The ground conditions within the Poor domain (and crown pillar area) are considered suitable only for a limited man-entry method.

Ground support design considers industry-standard empirical guidelines and SRK's experience in variable ground conditions. Compromises have been made in the extraction sequence as a result of the need to balance grade and production profiles, extraction of wide orebody areas, and other geotechnical constraints. Ultimately several aspects of the sequence may not be geotechnically optimal, and additional design may be required.

1.16.2 Groundwater

Groundwater is expected to inflow into the underground mine from the fractured bedrock around the mine itself and from geological structures. The total groundwater inflow will not be large compared with many other mines around the world, and could be dealt with by in-mine pumping, but the combination of the water with poor ground conditions and the mining methods could have an influence on mining productivity. Rock within the mining area is potentially acid-generating (PAG); hence, water that flows through the mine is assumed to need treatment before being discharged to the environment.

Groundwater inflow risks and potential effects will be managed in multiple ways, including cover and probe drilling, localized grouting, dewatering wells, and underground drainage galleries. As mine development proceeds, the groundwater system will start to drain down, but since the geological units only have moderate hydraulic conductivity and flow will be fracture controlled, it is expected that drainage performance will be highly variable over different parts of the mine. The combination of dewatering wells and drainage galleries with drain holes provides flexibility and some degree of redundancy to reduce the risk of areas not being sufficiently dewatered prior to production mining.

1.16.3 Water Management

All the water flow generated in the mine (infiltrated, industrial and paste fill water) will be managed in a single dewatering system. The system assumes that water flows running on ramps, declines and drifts is collected by gravity in a sump on each production level. Where gravity flow is not possible, a sump pump will be used to conduct water to the sump.

1.16.4 Mine Designs

SRK recommended TS where there is no Poor domain rock quality. The recommended dimensions for TS are 12 m wide x 20 m long x 25 m high.

For excavations within the Poor ground a D&F method is recommended. Dimensions for this method are 4.0 m wide x 4.0 m high.

The crown pillar will be from the 1240 L (south area of the mine) to the 1270L (north area of the mine). Because of instability risk associated mainly with the rock quality, the mining method for these areas will be D&F.

A sill pillar was included between the TS horizons 1080L and 1170L at 1155L, which allows for earlier production. The mining method for this sill pillar will be TS with a stope height of 15 m (instead of the 25 m to be used in the regular stopes).

Twin declines will be constructed, and will use a spiral to gain depth to maximize the distance from the surface, so that a vertical distance of approximately 155 m below the Machinaza River can be obtained. The mine ramp will be located central to, and will be approximately 50 m offset from, the main workings to the east of the deposit. The ramp configuration will enable haulage trucks to achieve higher average haul speeds and maintain safety standards. The ramp will be developed nominally at a -15% gradient.

Levels will be developed to access the strike extents of the deposit and connect the development to the return air raise (RAR in the north) and fresh air raise (FAR in the south) in order to establish flow-through ventilation.

Stope cross-cuts are required to access sill development from the haulage drifts, as well as connecting sill development within a given stope line separated by waste. Development will be centrally located within a given stope. The top development in a stope will initially serve as the drill horizon for the stope below, and then as the mucking horizon for the stope above. The bottom development in a stope will serve as the mucking horizon for the stope above.

1.16.5 Mine Operating Assumptions

An experienced, qualified mining contractor will be required to develop the declines. Contract mining will continue until the critical underground infrastructure has been constructed. The contractor will then demobilize. There will be a transition period as Owner mining equipment is introduced when access to additional ventilation and the mineralized zone is reached. Owner mining will eventually operate both development and production equipment.

1.16.6 Ventilation

The ventilation system proposed is a mechanical exhaust ventilation system (pull) where fresh air will enter by suction. The mine ventilation system will consist of the FAR and RAR. The raises will have a diameter of 5 m; the RAR will have an overall length of 290 m, and the FAR will have an overall length of 345 m. The remaining sections of the mine ventilation system will consist of the two declines, the mine ramp and the internal raises connecting levels.

1.16.7 Production Plan

Criteria and assumptions used in preparing the production plan include:

- The production plan has been developed on a monthly basis from Year 2017 to Year 2022 and annually thereafter
- The mine will operate 360 d/a with five days allowed for delays due to weather conditions
- The plant is scheduled to operate 365 d/a
- Production will be a combination of TS and D&F methods
- The process plant is designed to treat 3,500 t/d.

1.16.8 Backfill

The following backfill capacities and strength targets were set:

- The paste plant has been designed to cater for a nominal throughput of 70 m^3/h and will operate at an average utilization rate of approximately 60%
- The nominal design production rate of the CRF plant is 180 m^3/h
- Main pour target strength of 300 kPa after 14 days with a plug pour target strength of 434 kPa after three days
- CRF target strength of 3 MPa to 5 MPa after seven days.

The paste plant will be a batch-type backfill plant. All tailings leaving the process plant will be thickened to about 55%m solids. When no paste fill is required underground, the entire tailings stream will be pumped to the TSF. When paste fill is scheduled for underground, approximately half of the tailings stream will be pumped 3.4 km to the paste plant for further dewatering. Excess process water will be pumped back from the paste plant to the process plant using a second pipeline.

1.16.9 Underground Infrastructure Facilities and Services

It is proposed to keep material handling as simple as possible, relying on mobile equipment for transport instead of permanent infrastructure and facilities. Minimal storage will be developed underground.

Haul trucks will be repaired in a surface maintenance facility. Load–haul–dump vehicles (LHDs), drills, explosive carriers and scissor trucks will be repaired/maintained underground or driven/hauled to the surface shop for major work. Most of the mobile equipment, trucks and LHDs, and vehicles parked on surface will be fuelled from the surface facility. The rest of the fleet will be fuelled by the fuel/service vehicles or at the underground service facility.

The radio communication system is based on laying leaky cable feeder antenna. A fibre-optic network will provide a communication highway for control systems and data management inside the mine.

The air compressor system will consist of two compressors in operation and one on standby.

Explosive and detonator magazines will be located on a selected level underground.

1.16.10 Mining Equipment

Mine operations will use the same equipment for development for TS and for D&F. Drilling, support, loading and hauling equipment are the same for both methods. Different equipment is required for loading for production because TS is 5 m wide x 5 m high and D&F is only 4 m wide x 4 m high.

A maximum of four 10 yd^3 LHDs, four 12 yd^3 LHDs and nine 45 t trucks will be required for production and development. Additional equipment will include a rammer-jammer, jumbos and explosive loaders. Support equipment will include a scissor lift, crew and rescue vehicles, shotcrete sprayer and transmixer, jacklegs, scaler, boom truck, telehandler, core drill, Kubota tractors, rock breaker, dozer, grader, fuel and lube truck, and a front-end loader.

1.17 Recovery Methods

The Fruta del Norte process plant feed will contain gold in the following forms:

- Fine free gold
- Coarse free gold
- Gold contained in sulphides (refractory)
- Gold contained in other forms (e.g. silicates).

The GFL flowsheet was selected for the Project because of the nature of the gold in the plant feed. The up-front gravity circuit is essential to recover the coarse free gold and small amounts of fine free gold. The gravity circuit will reduce spikes in coarse gold content in the feed, ensuring that the flotation feed grade stays relatively uniform. The flotation circuit is capable of recovering the gold associated in sulphides (pyrite). The flotation circuit will reduce spikes in sulphide gold grade and provide a consistent feed to the CIL circuit. Typically, CIL circuits function best on a uniform feed; this can be provided by the combined gravity and the flotation circuits.

Run-of-mine (ROM) ore will be transported to ROM stockpiles. Feed will be reclaimed from the pile, transferred to an apron feeder and conveyed to feed the primary SAG mill. Oversize from the SAG mill discharge will be recycled back to the SAG feed. The SAG circuit product will be fed to a cyclone cluster which will be in closed circuit with the gravity concentrators and ball mill. Oversize from the gravity concentrator feed screen will be fed into the ball mill discharge which is pumped to the cyclone feed. Undersize will feed the gravity concentrators. Gravity concentrate will report to the intensive leach reactor and the gravity concentrator tailings will return to the cyclone feed.

The intensive leach reactor (ILR) will produce pregnant solution which will be directed to electro-winning cells to produce a gold–silver precipitate. After washing, the barren slurry will report to the flotation regrind circuit.

The overflow from the grinding cyclone will report to the flotation circuit. The flotation circuit will consist of three stages of flotation and regrind. Rougher and scavenger concentrate combined with ILR barren slurry will be directed to a regrind mill in closed circuit with a cyclone cluster. Final concentrate from the third cleaning stage of the flotation circuit will be thickened, filtered and bagged as product. Overflow from the concentrate thickener will be recycled to the process water tank.

Flotation tailings will be thickened and then report to the leach circuit while the thickener overflow will be recycled to the process water tank. The slurry will continue through pH conditioning before reporting to a series of CIL tanks where the slurry is leached with cyanide. Discharge from the leach train will report to cyanide destruction.

The loaded carbon generated from the CIL tanks will be pumped to the carbon elution and regeneration circuit. Once gold has been eluted, the carbon will be sent to regeneration. After quenching and screening to remove small particles, the reactivated carbon will be reintroduced to the CIL circuit.

Gold eluate will be sent to electro-winning cells using stainless steel cathodes to produce a gold–silver sludge. This is combined with sludge from the separate ILR electro-winning cell, filtered and dried. It is then mixed with fluxes and smelted to produce gold-silver doré.

Slurry discharged from the CIL tanks will report to cyanide destruction. A two-stage Inco SO_2/air process will be employed with the addition of lime. Sulphur dioxide will be provided as sodium metabisulphite. Slurry discharged from cyanide destruction will report to the tailings thickener. Underflow from the thickener will be sent to the tailings storage facility (TSF) and/or the paste backfill plant. Overflow from the thickener will be recycled back to the process water tank.

The process control system (PCS) will have redundancy and will allow dependable, simple and effective control of the plant processes. The PCS will monitor and act over continuous analogue loops, on/off valves, motors, variable frequency drives and programmable logic controllers (PLCs). The PCS will also signal alarms for abnormal conditions and store process data.

The surface operation areas will be maintained by an in-house maintenance crew. The maintenance team will be shared by the surface operations areas including the process plant, paste backfill plant, cemented rock fill plant, water treatment plants, tailings storage facility (TSF) and operations buildings.

The concentrate production rate is expected to be 160 t/d at a feed rate of 3,320 t/d and 140 t/d at a feed rate of 3,500 t/d. The actual concentrate quality could vary from month to month based on ore variability, mine planning and sequencing as well as the geometallurgy.

The total gold expected to be produced as doré varies from 90 koz to 145 koz per year during steady state, and is 1,323 koz during the LOM. The doré is expected to contain above 98% precious metals with the remainder made up of base metals and impurities. The precious metals portion is expected to contain approximately 70% gold and 30% silver.

1.18 Project Infrastructure

1.18.1 Access

The planned route to access the FDN site is by the Troncal Amazonica road to Los Encuentros and from this point to the Project site by a new main access road (a section of public road near the El Pindal village, and another section of road through the Ecuadorean jungle). After km 15, the new 22 km road will be used exclusively for access to the Project site. The access control facility will be located at km 15.

The main port for international cargo arrival will be Guayaquil. The Port of Bolivar may be used as an alternative.

1.18.2 On-site Infrastructure

On-site non-process services such as the camp, greenhouse, sewage treatment plant and mobile equipment will support the operation. There will be fresh water, domestic water and process water systems and a fire detection and protection system.

The utilities and services include compressed air supply and distribution, process control system, closed circuit television (CCTV) system, supervisory control and data acquisition (SCADA) system, waste management systems and fuel storage and distribution.

Mobile equipment for maintenance, operations services and transportation includes tractors and loaders for stockpile rehandling, mobile cranes, buses and utility vehicles.

1.18.3 Camps and Accommodation

The permanent camp facilities will be located close to the main access control to the Project site, at an altitude below 1,500 masl. The permanent camp will have a peak accommodations capacity of approximately 830 persons. The temporary camp will be located alongside the permanent camp and will provide 1,184 beds in tents. The temporary camp will have only accommodation, toilets, and showers; other services will be provided from the permanent camp. The temporary camp will be closed when operations manpower reaches steady-state.

1.18.4 Off-site Facilities

In order to reduce the impact of the Project footprint, some support facilities are planned be located off the main Project site. For the purpose of the 2016 FS, the off-site facilities are considered to be located 12 km from the main Project site; however this location is conceptual, and the final site location will be determined during future Project stages. These facilities include a guard house, light vehicle shop; warehouse and laydown area; and an office building.

Lundin Gold has established administration offices in Quito and in Los Encuentros. These existing offices provide administrative and logistics support to the Project, and are not part of the Project capital costs.

1.18.5 Power

The Ecuadorian electrical system is based on a high quality electricity service matrix, the distribution system is called the Sistema Nacional de Distribución (SND, National Distribution System). The SND is controlled by CELEC EP Transelectric, a government institution in charge of power transmission and distribution.

The Project site is located within the supply concession area of the Empresa de Energia Regional del Sur (EERSA, Regional Electric Company of the South). SND has no substation near to the Project with sufficient capacity or reliability to feed the Project. Lundin Gold is participating in a public infrastructure investment to reinforce the SND matrix in the area, and is contributing to the installation of a transmission line between Taday and Bomboiza.

The overall Project power requirements are expected to be met via a 230 kV double circuit transmission line from the Bomboiza substation. The contract for this substation has been awarded and it will be built at the same time as the Taday–Bomboiza transmission line. The Bomboiza substation will be situated approximately 50 km away from the FDN site. A transmission line will be built from the Bomboiza substation to a new substation at El Pindal, near Los Encuentros. This system will be a public transmission line and substation, owned and operated by CELEC EP Transelectric, with installation paid for by Lundin Gold.

From the El Pindal substation, a single-circuit, 230 kV dedicated transmission line will be built to feed the Project. It is planned to build the FDN main substation on the process area platform. This substation will step down the power to 13.8 kV, and will distribute power throughout the plant site at this voltage.

The annual average power demand is estimated to be approximately 222,000 MWh.

1.18.6 Communications

The communications system for the Project will consist of a fibre-optic network infrastructure, telephony system, radio communications, mobile telephony, and satellite communications. The data management system will be connected to the communications systems.

1.19 Environmental, Permitting and Social Considerations

1.19.1 Baseline Studies

The physical (abiotic), biotic, social, economic, and cultural baseline has been characterized for the Project using primary information gathered in the field, and secondary information from official sources such as Government records. Field studies and data gathering for the baseline studies were undertaken between 2008 and 2016.

1.19.2 Mine Waste Stockpile Design

As part of the underground development at FDN approximately 2.03 Mt of waste material will be generated. Of this, approximately 1.29 Mt (64%) will be returned underground as part of the backfill management strategy. The remaining 0.74 Mt of material will need to be permanently stored on surface.

An area to the south of the process plant has been allocated to accommodate waste from the underground mine. Two different types of waste will be produced:

- Potentially acid rock drainage (ARD) or potentially acid generating (PAG)
- Non-potentially acid generating (NAG).

1.19.3 Ore and Low Grade Stockpiles

There are three types of stockpiled material based on grade:

- High grade (>7 g/t Au): almost never stockpiled
- Medium grade (4.7 g/t Au to 7 g/t Au): maximum 30,000 t
- Low grade (2.7 g/t Au to 4.7 g/t Au): maximum 170,000 t late in the mine life (Year 2033).

The area allocated for these stockpiles is close to the crusher station at the process plant. Stockpiled material will be consumed by the time the mine closes.

1.19.4 Tailings Storage Facility

The facility will be located in the uppermost portion of the valley, to minimize the catchment area and to maximize the separation distance from the Zarza River downstream.

The tailings dam will be an earth-and-rock-fill structure constructed with a maximum dam height of 63 m measured at the dam centre line. The ultimate dam will have a crest width of 6 m and a length of 700 m at final grade. A starter dam will be initially constructed to store start-up water for the mill and create sufficient storage for the tailings in the first year of operation, and to safely contain the probable maximum flood (PMF). The TSF dam will be raised continuously throughout the service life until reaching the ultimate elevation. Each dam raise will be completed at least one year before the maximum tailings pond elevation required each year; currently dam raises are contemplated at Years 0, 2, 5, 10 and 14 (ultimate).

A total of 12.15 Mt of GFL tailings will be pumped to the TSF at 55% solids over the mine life. Excess water will be reclaimed to the mill by a floating barge. The sludge produced from the treatment of contact water from the mine at the water treatment plant (WTP) will be delivered at a rate of 4 m^3/h and stored in the TSF. Sediments removed from ponds located in the mine infrastructure area will also be stored in the TSF and will be delivered at a rate of 8 m^3/h.

The TSF design incorporates sufficient dam freeboard at all times during operations to accommodate the sloping tailings beach and to contain the PMF and any excess water volumes in the tailings basin without discharge. Diversions will be constructed on the east side of the TSF catchment to divert non-contact water. These channels will be lined to limit erosion and are designed to convey peak catchment runoff from the 1:100 year storm event.

1.19.5 Hollín Borrow Pit

Lundin Gold will need to exploit a borrow pit to provide granular materials for construction and mine backfill, from construction through to mine closure.

The Hollín Borrow Pit will be operated as an open pit mine. Rock fill material will be delivered direct to the TSF and seepage pond walls; all other materials will be processed through an aggregate plant, which will use screening and crushing/screening to produce the required products.

The mining of the Hollín Borrow Pit is expected to be done in sedimentary rocks, mainly siliceous sandstones belonging to the Hollín Formation, but due to the quantity of material required, will extend through the Hollín Formation sandstones into the underlying intrusive rocks of the Zamora Batholith.

1.19.6 Waste Management

The waste management centre (WMC) was sized to receive waste during operations and manage the waste temporarily until final disposal by an authorized contractor. The WMC is designed to handle waste from one month of operations.

1.19.7 Water Management

Four main types of water will need to be managed during construction and operations:

- Non-contact water: Water (either runoff from precipitation or flowing in natural streams) whose quality is not impacted by the project infrastructure and activities
- Unaffected contact water: Water that is likely to have had a sediment load increase but not subject to chemical/biological impact requiring treatment other than total suspended solids (TSS) removal in order to meet water quality regulations. Requires TSS removal only, prior to discharge to a natural water course; no water treatment plant is required
- Affected contact water: Water that must be sent to a water management pond and a water treatment plant (WTP) prior to being discharged to the environment.
- Neutral water: Groundwater collected above the orebody at the underground mine. Requires TSS removal and/or primary treatment only (depending on the quality parameters) prior to being discharged.

Six water treatment plants are planned, and will include:

- Two domestic water treatment plants: one will be located at the camp site and the other at the process plant
- A sewage treatment plant will be located at the camp site. The process plant sewage will be managed using septic tanks
- A main effluent water treatment plant that will be located at the process plant site and will treat most of the affected contact water from the site
- The Hollín Borrow Pit water treatment plant that will be located close to the aggregate plant and will treat affected contact water from the borrow pit area
- An existing plant at the site will be used during the first year of mine dewatering; it then will be moved to the mine portal area.

Four water management work types are proposed:

- Diversion works: To divert non-contact storm water to prevent it from reaching the site during the construction and operations phases of the project. These comprise riprap interception works, lined channels and creek riprap discharge works. They also include slope drainage systems for mass earthworks
- Contact water works: To manage affected and unaffected water during the construction and operations phases. These comprise sumps, water management ponds, chutes (steep slope conduits), energy dissipaters, water treatment plants, pumping systems and emergency discharge works to natural water courses
- Neutral water works: To deal with groundwater from the dewatering wells above the deposit. These comprise a pumping system, a water management pond and a discharge to the Machinaza River
- Secondary and minor drainage networks: To be located within the facilities for non-contact and contact water, including small sumps, downspouts, and minor collecting pipes. These works have not been designed at the feasibility level.

A water balance model and a water quality model were developed in support of a water management plan (WMP) for the site:

- The purpose of the site-wide water balance model was to simulate the water management plan for the mine site. The model tracks water from the sources, through collection and conveyance systems, usage, storage, treatment and discharge to the environment. The results of the water balance model demonstrate that the proposed water management plan at the site is feasible
- The purpose of the site-wide water quality model was to simulate the water quality elements of the Project, identifying sources of loading, assessing the mixing of different inflows and estimating the resulting water quality concentrations in each

flow. The water quality results determined which water flows met discharge requirements and which flows did not meet discharge requirements and will require water treatment. Water quality parameters requiring treatment in each flow component were identified. The water quality model focused on parameters of concern identified from the surface water quality assessment (aluminium, arsenic, copper, cobalt, cyanide (CN), iron, magnesium, potassium, manganese, lead, selenium and zinc). The results of the water quality model demonstrate that the proposed WMP for the site is feasible and will meet regulatory requirements for discharge to the receiving water bodies.

The general purpose of the WMP was to outline an integrated water management strategy to be followed at the FDN site during the design, construction, and operations phases, and to demonstrate a feasible, rational, sustainable, and environmentally-friendly plan to deal with both surface water and groundwater.

1.19.8 Closure Plan

Closure planning has been undertaken to a conceptual level, and will be continually updated throughout the Project life. The conceptual Closure Plan has been developed in accordance with Article 125 of the Ecuadorian Environmental Regulations for Mining Activities (RAAM) and Title X of the Mining Safety Regulations.

The closure activities will cover closure aspects related to environmental factors such as soil, air and water that are directly related to the community health and safety. Aspects related to economic and cultural dynamics of the communities have not been considered in the current conceptual plan.

The definitive Closure Plan must be presented two years prior to cessation of operations. Under RAAM, mine closure monitoring should last for at least five years after the mining operations are complete.

The closure cost estimate in the conceptual Closure Plan is $28.8 million.

1.19.9 Permitting

Permitting requirements were evaluated by project phase, including before construction (16 permits), during construction (six permits), and before operations (three permits).

The Environmental Design Criteria (EDC), updated through October, 2015, are based on Ecuadorian law, quality criteria and regulations, as well as international standards such as those issued by the International Finance Corporation (IFC), the World Bank, the World Health Organization (WHO), the International Cyanide Management Code (ICMC), the International Network on Acid Prevention (INAP), and the International Council of Mining and Metals (ICMM).

1.19.10 Social Considerations

The Project's indirect influence is expected to extend to some neighbouring communities, including the parish of Los Encuentros and two communities from neighbouring parishes. Los Encuentros is a rural parish located in Yantzaza county, characterized by the existence of one main population centre (the parish seat and home of the parish government) where the population has consolidated. There are also several scattered population centres, known as communities, neighborhoods and sectors.

Some cultural sites have been recorded in the study area, but the Project is not expected to impact any cultural heritage, and strict archaeological protocols are in place in consultation with the National Cultural Patrimony Institute (INPC).

Although perceptions of artisanal mining are low, the community is very supportive of the Project, and the primary concern is access to employment. There is currently no large-scale mining in Ecuador.

A community relations program has been defined based on the Community Development Support Program (PADC, Plan de Apoyo a Desarrollo de la Communidad) which seeks to implement corporate responsibility strategies, to maintain a social licence with the communities, and to comply with socio-environmental legislation applicable to Aurelian's operations. The PADC is based on the principles of community participation, sustainable development and human development.

1.20 Markets and Contracts

Lundin Gold commissioned Cliveden Trading AG (CTAG) to prepare an independent marketing study on the planned products from the FDN Project, which will be a precious metal doré and a gold–silver concentrate.

Information provided to CTAG assumed the following:

- The doré is expected to contain >98% precious metals and <2% base metals and impurities
- The precious metals portion is expected to contain approximately 60% gold and 40% silver.

Gold–silver doré bullion is typically sold through commercial banks and metals traders with sales price obtained from the World spot or London fixes. There are a number of global refineries specializing in refining of doré to bullion; the bulk of refining capacity is centred in Switzerland.

CTAG initially contacted a number of smelters, roasters and metals traders for information regarding gold–silver concentrate sales, and received indicative smelting terms based on the assay data of the concentrate produced from early testwork performed by Kinross to determine the marketability of the concentrate. CTAG subsequently contacted the smelters, roaster and metal traders with more recent assay data of the concentrate produced from testwork performed by Aurelian. Based on the bids received for the gold–silver concentrate and the standard terms for precious metal bullion, the CTAG report concluded that there is significant interest in both products. Potential receivers of the gold-silver concentrate product include the following in general:

- Copper smelters
- Gold roasters
- Gold pressure oxidation plants (POX)
- Bacterial leach plants
- Lead smelters
- Commercial buyers (metals traders).

Most precious metals concentrates are traded on the basis of term contracts. These frequently run for terms from one year to 10 years, although many long-term contracts are treated as evergreen arrangements that continue indefinitely with periodic renegotiation of key terms and conditions. These contracts are easily transacted, and in Amec Foster Wheeler's opinion, would be typical of such contracts within industry.

For the purposes of the Report, CTAG has suggested the following sales distribution for the gold–silver concentrate:

- 40% to western smelters
- 40% to Asian smelters
- Balance in annual sales to traders.

High-grade doré bars are readily marketable, sold through commercial banks and metals dealers, with relatively standard sales prices obtained on spot or London fixes. CTAG expects that the terms of the sales contract would be typical of and consistent with standard industry practices and would be similar to contracts for the supply of gold–silver doré elsewhere in the world.

No contracts are currently in place for any production from the Project.

1.21 Capital Cost Estimates

The methodology used in the development of the capital cost estimate and the level of engineering definition result in the estimate having an accuracy of ±10% to ±15% including the contingency based on the 80% confidence level.

The estimate combined inputs from Amec Foster Wheeler, KCB, Lundin Gold, NCL, and Paterson and Cooke (P&C). The cost estimate was divided into capital costs (direct, indirect and Owner's costs, and contingency) and sustaining and closure costs:

- Capital costs
  - Direct costs: costs for productive works and permanent infrastructure. Includes productive infrastructure, services and equipment required for the extractive process
  - Indirect costs: costs needed to support the construction of the facilities included in the direct costs. Includes engineering, procurement and contract management (EPCM) services, EPCM temporary facilities (infrastructure) and construction management, construction camp and associated services, capital spare parts, freight and logistics
  - Owner's costs: costs associated with Lundin Gold's Project administration, geological studies, support infrastructure, safety and environmental, community relations, administration and finance, human resources and others
  - Contingency: includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site specific conditions. Also accounts for variation resulting from uncertainties that are clarified during detail engineering, when basic engineering designs and specifications are finalized
- Sustaining and closure costs
  - Capital expenditures after the start of operations include costs for the tailings dam wall growth, mine and other equipment replacement and the paste fill plant, plus closure costs. These costs are included in the financial analysis in the year in which they are incurred. The capital cost estimate includes construction activity costs to Q1 2020. Costs after this are classified as sustaining capital.

The initial Implementation Phase capital cost is estimated to be US$668.7 million (Table 1-4). The sustaining capital is estimated to be US$291.9 million. The total capital construction costs are therefore US$960.6 million.

Table 1-4: Implementation Phase Capital Cost Summary by Area

Description	Amount (US$ M)	% of Total
Underground mine	120.5	18.0
Ore handling	7.5	1.1
Process plant	74.3	11.1
Tailings/ reclaim water facilities	30.8	4.6
On-site infrastructure	121.4	18.2
Off-site infrastructure	71.2	10.6
Aggregate borrow pit	0.4	0.1
Indirect costs	126.1	18.9
Owners' costs	49.3	7.4
Contingency	67.3	10.1
Total Cost	***668.7***	***100.0***

Note: Totals may not sum due to rounding

1.22 Operating Cost Estimates

The operating cost estimate was based on Q1 2016 assumptions. The estimate combined inputs from Amec Foster Wheeler, KCB, Lundin Gold, NCL, and P&C, and has an overall accuracy of ±10%.

The operating cost estimate is inclusive of site costs during the operational period (commencing once the C4 commissioning certificates are signed) until site closure. Variable costs were based on a mine plan provided by NCL.

The overall life of mine operating cost estimate is US$118/t, and includes base costs, non-recoverable taxes and leasing. Operating costs are estimated at US$414/oz Au, including all site costs. Mining costs are the greatest contributors to the overall operating cost, followed, in order of contribution, by process, general and administrative (G&A) and surface infrastructure, costs.

A summary of the costs by area is provided in Table 1-5.

Table 1-5: Operating Cost Summary

Area	LOM Total US$ (million)	US$/tonne	US$/oz Au
Mining	934.4	60.30	211.5
Process	516.9	33.40	117.0
Surface Infra.	142.8	9.20	32.3
G&A	234.2	15.10	53.0
Total	***1,828.3***	***118.00***	***413.8***

1.23 Economic Analysis

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this Report include, but are not limited to, statements with respect to future gold and silver prices, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates, unexpected variations in quantity of mineralised material, grade or recovery rates, geotechnical and hydrogeological factors, unexpected variations in geotechnical and hydrogeological assumptions used in mine designs including seismic events and water management during the construction, operations, closure, and post-closure periods, the timing and amount of estimated future production, costs of future production, capital expenditures, future operating costs, costs and timing of the development of new ore zones, success of exploration activities, permitting time lines and potential delays in the issuance of permits, currency exchange rate fluctuations, requirements for additional capital, failure of plant, equipment or processes to operate as anticipated, government regulation of mining operations, environmental, permitting and social risks, unrecognized environmental, permitting and social risks, closure costs and closure requirements, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

The Project has been evaluated using a discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections. Cash outflows include capital expenditures (including the three years of pre-production costs), operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the mid-point of each period.

To reflect the time value of money, annual net cash flow projections are discounted back to the Project valuation date using 5% to produce the base case. The discount rate appropriate to a specific project depends on many factors, including the type of commodity; and the level of project risks (e.g. market risk, technical risk and political risk). The discounted, present values of the cash flows are summed to arrive at the Project's net present value (NPV).

In addition to the NPV, the internal rate of return (IRR) and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project.

The financial model includes consideration of metal prices, transport costs, royalties and taxes, and working capital. An amount of US$430 million of historical costs is considered in the financial model. These historical costs provide a shield against taxes and profit-sharing expenses.

The after-tax NPV at a 5% discount rate over the estimated mine life is US$676 million. The after-tax IRR is 15.7%. The after-tax payback of the initial capital investment is estimated to occur 4.5 years after the start of production. A summary of the financial analysis is presented in Table 1-6, with the base case discount rate highlighted.

The life of mine all-in sustaining cost (AISC) per ounce is US$623 (Table 1-7).

1.24 Sensitivity Analysis

A sensitivity analysis was performed on the base case NPV after taxes to examine the sensitivity to gold price, operating costs, capital costs and labour costs. In the pre-tax and after-tax evaluations, the Project is most sensitive to changes in gold price, less sensitive to changes in operating costs, and least sensitive to capital cost and labour cost changes. Figure 1-1 shows the results of the after-tax analysis.

The gold grade is not presented in the sensitivity graph because the impact of changes in the gold grade mirror the impact of changes in the gold price.

1.25 Execution Plan

The Project schedule entails significant project activity durations, some of which may run concurrently, including a duration of 11 months for the engineering, procurement, contracting and preliminary construction of Early Works (see Section 26 for the recommended work in this program), 12 months for the construction of the access road and bridge over the Zamora River, 34 months for construction of the twin declines, six months to develop the aggregate borrow pit and plant, nine months for the mass earthworks and 20 months for the construction of the process plant and facilities.

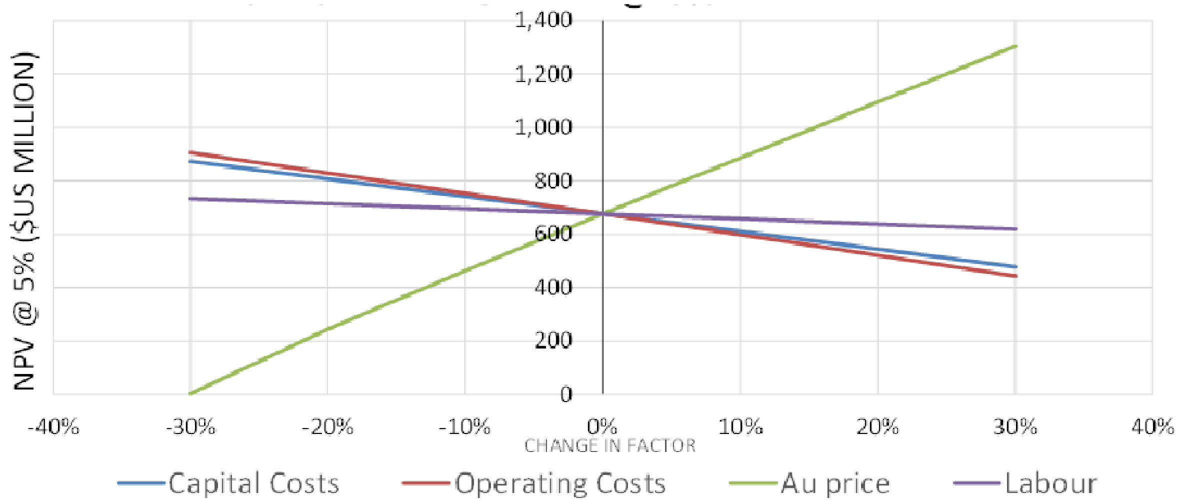
Table 1-6: Financial Analysis Summary (base case is highlighted)

Indicator	Units	LOM Value
Pre Tax		
NPV 4%	US$ million	1,452
NPV 5%	US$ million	1,283
NPV 8%	US$ million	879
NPV 10%	US$ million	675
Payback period from start of production	Years	3.7
IRR	%	23.8
After Tax		
NPV 4%	US$ million	791
NPV 5%	US$ million	676
NPV 8%	US$ million	402
NPV 10%	US$ million	264
Payback period from start of production	Years	4.5
IRR	%	15.7

Table 1-7: Operating Statistics

	Units	Year 1	Year 2	Year 3	Avg. Y1–10	LOM
Metal Production						
Au recovered	koz	149	308	390	345	4,418
Ag recovered	koz	141	329	431	389	5,177
AISC Costs and Profit Margins per oz payable						
Au price	US$/oz	1,250	1,250	1,250	1,250	1,250
Cash cost sub-total (operating cost)	US$/oz	823.82	585.78	473.08	541.78	552.56
Sustaining and closure costs	US$/oz	701.12	63.77	35.86	102.92	70.87
AISC costs/oz Au payable	US$/oz	1,524.94	649.55	508.94	644.70	623.43
Operating Margin/oz Au payable	US$/oz	-274.94	600.45	741.06	585.52	626.57

Figure 1-1: After-Tax Sensitivity Analysis (NPV 5%)



Note: Figure prepared by Amec Foster Wheeler, 2016.

The implementation strategy for the Project is dictated by the duration of the construction of the twin declines, which will provide access to the deposit; the estimated duration for this construction is 34 months. It is possible to build all the surface facilities including the process plant and associated infrastructure during this period. Therefore, the construction of the mine access is the critical path and the Early Works to expedite the construction of the access are also critical. The objective of the Early Works is to build access and platforms for the start of construction of the portals and declines, and to provide support facilities. The Early Works have been given special attention in the execution plan because they will need to start very soon after approval of the 2016 FS, if the proposed Project schedule is to be met.

1.26 Risks and Opportunities

Two Project risk assessment workshops were conducted. The risk structure breakdown was based on the main Project phases and areas. During the review sessions during the feasibility study, mitigation strategies were defined and assigned to the risks. Mitigation can either reduce the probability of the risk occurring and/or can reduce the impact of the risk if it occurs.

Risks that were classified as “high” included:

- Delays in the EIA or receipt of permits for construction would affect the Project schedule for the development of the mine and/or process plan

- Non-availability of sufficient qualified staff to support the construction efforts, which would initially affect the construction phase, but would also have a run-on effect on the Project schedule.

Moderate risks comprised:

- The potential for workforce unrest through non-compliance with local labour laws, as a result of third-party industrial activity, or from technical breaches of safety obligations by contractors; this could affect all stages of the Project if not well managed from the start of construction
- Unrecognized complexities arising from new information obtained on the Project geotechnical and hydrogeological setting during mining operations, which could affect the assumptions made during the 2016 FS and potentially result in changes to the mine plan.

1.27 Interpretation and Conclusions

Under the assumptions discussed in this Report, the Project returns a positive economic outcome. The decision to initiate construction activities rests with Lundin Gold management, and will require formal permit grants by the Government of Ecuador.

1.28 Recommendations

A two-phase work program is proposed for the Project.

Phase 1 is designed to provide information to support the planned Project Early Works and complete the construction phases associated with the Early Works. The main objective of the Early Works is to provide the infrastructure, services and facilities to support the start of construction of the mine twin declines and to reduce the risk for the basic engineering capital expenditure estimate and remaining earthworks.

Phase 2 comprises additional data collection in specific technical disciplines that will support basic engineering and refinement of the capital cost estimates, and additional exploration efforts.

The work phases can be carried out in parallel. As information from a discipline area becomes available, it will be reviewed to determine if any revisions to the existing Project assumptions, Early Works or Project schedule may be required.

The Phase 1 program is estimated at $32.7 million; Phase 2 is provisionally estimated at about $7.9–$13.7 million.

2.0 INTRODUCTION

2.1 Introduction

Amec Foster Wheeler Americas Ltd. (Amec Foster Wheeler) was requested by Lundin Gold Inc. (Lundin Gold) to compile an independent NI 43-101 Technical Report (the Report) for the Fruta del Norte (FDN) Project (the FDN Project or the Project) located in Ecuador (Figure 2-1).

2.2 Terms of Reference

The Report will be used in support of Lundin Gold's press release dated 6 June, 2016, that is entitled "Lundin Gold Announces a Positive Feasibility Study for the Fruta del Norte Project and Conference Call".

The firms and consultants who are providing Qualified Persons (QPs) responsible for the content of this Report, which is based on a feasibility study completed in 2016 (the 2016 FS) and supporting documents prepared for the 2016 FS, are, in alphabetical order, Amec Foster Wheeler Americas Ltd and Amec Foster Wheeler E&C Services Inc (collectively Amec Foster Wheeler), Klohn Crippen Berger Ltd (KCB), MM Consultores Limitada (MM Consultores), NCL Ingeniería y Construcción SpA. (NCL), and Roscoe Postle and Associates Inc. (RPA).

The FDN Project is wholly-owned by Lundin Gold, through its indirectly-held subsidiary, Aurelian Ecuador S.A. (Aurelian). For the purposes of this Report, Lundin Gold is used interchangeably for the parent and subsidiary companies, unless specified.

Aurelian was previously owned and operated by Aurelian Resources Inc. (Aurelian Resources), and by Kinross Gold Corporation (Kinross).

Currency is expressed in US dollars unless stated otherwise; units presented are typically metric units, such as metric tonnes, unless otherwise noted.

Calendar years are used in some sections of the Report, in relation to the proposed mine plan and execution plan. The years shown are for illustrative purposes; the actual timing may vary. Formal approval is required from the Lundin Gold Board for construction, and additional permits need to be granted by the Government of Ecuador.

Figure 2-1: Location Plan

Note: Map sourced from Mappery.com, 2016, and amended by Amec Foster Wheeler

2.3 Qualified Persons

The following serve as the qualified persons for this Technical Report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101F1:

- Mr. Ignacy (Tony) Lipiec, P.Eng., Director, Process Engineering, Amec Foster Wheeler
- Ms. Juleen Brown, MAusIMM CP, Principal Environmental Engineer, Amec Foster Wheeler
- Mr. Simon Allard, P.Eng., Principal Consultant, Amec Foster Wheeler
- Mr. Charles Masala, P.Eng., Associate Water Resources Engineer, Amec Foster Wheeler
- Ms. Stella Searston, RM SME, Principal Geologist, Amec Foster Wheeler
- Mr. Bryan D. Watts, P.Eng., Chairman, Principal, KCB
- Mr. Alejandro Sepúlveda, RM CMC, Principal, Project Director, NCL
- Mr. Anthony (Tony) Maycock, P.Eng., MM Consultores
- Mr. David Ross, P.Geo., Director - Resource Estimation, Principal Geologist, RPA.

2.4 Site Visits and Scope of Personal Inspection

Mr. Charles Masala visited the site from 1 to 2 April, 2015. During the site visit, he inspected several surface water monitoring stations, the weather station, the southern exploration decline portal and the proposed location of the process plant. He also visited the Machinaza, Blanco, and Zarza Rivers, several local streams within the Project area, and examined the drainage design for the camp.

Mr. Bryan D. Watts, visited the Project site from 16 to 18 October, 2015. With local guides, he walked over the proposed tailings site and the process plant site. In the camp, he inspected drill cores from test holes put down in the proposed tailings site, the process plant site, and the quarry location.

Mr. Alejandro Sepúlveda visited the site from 20 to 23 June, 2015. During this visit, he inspected the South Exploration Decline and safety conditions were assessed.

Mr. Tony Maycock visited the site on 7 to 8 April 2016. He reviewed the condition of the temporary access road from the town of Los Encuentros to the existing exploration camp which is planned to be used as a temporary camp for the Early Works program (see Section 26). He also travelled on the exploration road which leads to a bridge over the River Machinaza. This road will be improved and extended to the new North Portal. From the bridge the proposed process plant site was viewed approximately one kilometre in the distance. Closer access was not possible due to the thick jungle vegetation.

Mr. David Ross visited the site from 7 to 9 April, 2016. During the site visit, Mr. Ross examined core from several drill holes (FN3600d01, CP-08-198, CpP-08-204, CP-

07,137, CP-08-196, and CP-08-191) and compared observations to logs, visited several outcrops and former drill sites, and reviewed logging and sampling methods. The procedures outlined by Lundin Gold staff during the site visit were found to meet generally accepted industry practices. As part of the review, Mr. Ross independently resampled four mineralized intervals from two drill holes. Results were acceptable.

2.5 Effective Dates

The Report has a number of effective dates as follows:

- Date of latest information on the ongoing exploration and drilling program: 24 May, 2016
- Date of close-out of database used in resource estimation, date of resource model, and date of Mineral Resource estimate: 31 December, 2015
- Date of Mineral Reserve estimate: 30 April, 2016
- Date of letter regarding taxation considerations that supports the financial analysis: 29 April, 2016
- Date of financial analysis: 30 April 2016
- Date of supply of latest information on mineral tenure, surface rights and Project ownership: 23 May, 2016.

The overall effective date of the Report is taken to be the date of the financial analysis and Mineral Reserve estimate and is 30 April 2016.

2.6 Information Sources and References

2.6.1 Principal Information Sources

The principal information source for the Report was:

- Amec Foster Wheeler, 2016: Aurelian Ecuador S.A., Fruta del Norte Feasibility Study: report compiled for Aurelian, May 2016.

The reports and documents listed in Section 2.6 (Previous Technical Reports), Section 3.0 (Reliance on Other Experts) and Section 27.0 (References) of this Report were used to support the preparation of the Report. Additional information was sought from Lundin Gold personnel where required.

2.6.2 Amec Foster Wheeler

The following provided specialist input to Ms. Juleen Brown:

- Ms. Maria-Cristina Acosta, Manager Environmental and Permitting, Aurelian, and Mr. Fernando Pombo, Senior Environmental Engineer, Aurelian provided supplemental information on aspects of the environmental, permitting and social studies completed in support of the 2016 FS, and future permitting activities
- Mr. Nathan Monash, Vice President, Business Sustainability, Lundin Gold, provided supplemental information on aspects relating to the proposed Social Impact Management Plan.

The following provided specialist input to Ms. Stella Searston:

- Ms Sheila Colman, Vice President, Legal and Corporate Secretary, Lundin Gold, provided supplemental information on aspects of corporate ownership and the Definitive Exploration Agreement.

The following provided specialist input to Mr. Charles Masala:

- Mr. Vikram Khera, P.Eng., Associate Process Engineer/Financial Analyst, Amec Foster Wheeler, provided the evaluation and design of the water treatment plants, as summarized in the technical report
- Mr. Bruce Ott, R.Bio., Principal Scientist-Environmental, Amec Foster Wheeler, provided input and senior review for the water quality analysis and modelling, as summarized in the technical report
- Mr. Felipe Dibarrant., Senior Specialist Engineer, Amec Foster Wheeler, provided analysis and design of the site drainage system including diversion ditches and ponds, as summarized in the technical report
- Mr. Dan Mackie, P. Geo., Principal Consultant Hydrogeology, SRK, provided the groundwater assessment and input data, as summarized in the technical report
- Mr. Patrick Williamson, P.G., Principal Consultant Hydrogeology, SRK, provided the geochemistry assessment and input data, as summarized in the technical report
- Mr. Jose Sanchez, Associate Geotechnical Engineer, KCB, provided hydrological assessment and input data, as summarized in the technical report
- Ms. Maria-Cristina Acosta, Manager Environmental and Permitting, Aurelian, and Mr. Fernando Pombo, Senior Environmental Engineer, Aurelian, provided supplemental information on water management aspects of the environmental and permitting completed in support of the 2016 FS, and future permitting activities.

2.6.3 KCB

The following provided specialist input to Mr. Bryan Watts

- Mr. Leslie Correia, Pr. Eng., Project Engineer, Paterson and Cooke Consulting Engineers (Pty.) Ltd. (P&C), provided information on the design of the cemented rock fill plant (CRF), the paste backfill plant and tailings production rates, as summarized in the technical report
- Mr. Jose Sanchez, Project Manager, KCB, compiled geotechnical data, data interpretations and design for the tailings dam area
- Ms. Lisbeth Pimentel, Water Resources Engineer, KCB, compiled hydrological data and data interpretation site-wide.
- Ms. Karim Espinoza, KCB, Water Resources Engineer, data interpretations and hydraulic design for the tailings dam area.

2.6.4 MM Consultores

The following provided specialist input to Mr. Tony Maycock

- Mr. Marcelo Henriquez, Engineering Manager, Amec Foster Wheeler, provided information on the infrastructure design, as summarized in the technical report
- Mr. Niresh Deonarian, Process Manager, Amec Foster Wheeler, provided information on infrastructure operating costs, as summarized in the technical report
- Mr. Alonso Peraita, Lead Capital Cost Estimator, Amec Foster Wheeler, provided information on infrastructure capital costs, as summarized in the technical report.

2.6.5 NCL

The following provided specialist input to Mr. Alejandro Sepúlveda:

- Mr. Bruce Murphy, FSAIMM, Principal Consultant Rock Mechanics, SRK, provided the geotechnical data and data interpretations as summarized in the technical report
- Mr. Dan Mackie, P. Geo., Principal Consultant Hydrogeology, SRK, provided the hydrogeological data and data interpretations as summarized in the technical report
- Mr. Leslie Correia, Pr. Eng., Project Engineer, Paterson and Cooke Consulting Engineers (Pty.) Ltd. (P&C), provided information on the design of the cemented rock fill plant (CRF) and the paste backfill plant as summarized in the technical report
- Mr. Gino Giubergia, Project Engineer and Dr. Antonio Peralta, Principal Mining Specialist, Amec Foster Wheeler, provided information on the design and operation of the Hollín Borrow Pit as summarized in the technical report.

2.6.6 RPA

The following provided specialist input to Mr. David Ross:

- Mr. Nicholas Teasdale, MAusIMM CP (Geo), Vice President Exploration, Lundin Gold, provided geological and exploration information in support of the Mineral Resources estimate.

2.7 Previous Technical Reports

Lundin Gold, under its former name of Fortress Minerals Corp., filed a Technical Report for the Project as follows:

- Evans, L., Ross, D., and Scholey, B., 2014: Technical Report On The Mineral Resource Estimate, Fruta del Norte Project, Ecuador: technical report prepared by RPA Inc. for Fortress Minerals Corp., effective date 21 October, 2014.

Kinross had previously filed Technical Reports for the Project as follows:

- Henderson, R., 2009: Fruta del Norte Project, Ecuador, NI 43-101 Technical Report: technical report prepared for Kinross Gold Corporation, effective date 31 December 2009
- Hennessey, T., Puritch, E., Gowans, R., and Leary, S., 2008: A Mineral Resource Estimate for the Fruta del Norte Deposit, Cordillera del Cóndor Project, Zamora-Chinchipe Province, Ecuador: technical report prepared by Micon International Ltd. for Aurelian Resources Inc., readdressed to Kinross Gold Corporation, effective date 15 November 2007, amended 21 October 2008.

Aurelian Resources, prior to the take-over by Kinross, had also filed the following Technical Reports on the Project:

- Hennessey, T., Puritch, E., Gowans, R., and Leary, S., 2008: A Mineral Resource Estimate for the Fruta del Norte Deposit, Cordillera del Cóndor Project, Zamora-Chinchipe Province, Ecuador: technical report prepared by Micon International Ltd. for Aurelian Resources Inc., effective date 15 November 2007
- Hennessey, B.T. and Stewart, P.W., 2007: A Review of the Geology of, and Exploration and Quality Control Protocols Used at the Fruta del Norte Deposit, Cordillera del Cóndor Project, Zamora-Chinchipe Province, Ecuador: technical report prepared by Micon International Ltd. for Aurelian Resources Inc., dated December 2006, effective date 9 January 2007
- Hennessey, B.T. and Puritch, E., 2005: A Mineral Resource Estimate for the Bonza-Las Peñas Deposit, Cordillera del Cóndor Project, Zamora-Chinchipe

Province, Southeastern Ecuador: technical report prepared by Micon International Ltd. for Aurelian Resources Inc., effective date 13 January 2005

- Mullens, P., 2003: Geological Report on Exploration at the Cordillera del Cóndor Project, Zamora-Chinchipe Province, Southeastern Ecuador: technical report prepared for Aurelian Resources Inc., effective date 16 December 2003
- Stewart, P. W., 2003: Geological Report on Exploration at the Cordillera del Cóndor Project, Zamora-Chinchipe Province, Southeastern Ecuador: technical report prepared for Aurelian Resources Inc., effective date 16 April 2003.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The QPs have relied upon the following other expert reports, which provided information regarding mineral rights, surface rights, property agreements, royalties, taxation and marketing sections of this Report.

3.2 Mineral Tenure

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Lundin Gold and legal experts retained by Lundin Gold for this information through the following documents:

- Borja, R, 2016: Mineral Concessions Opinion: letter addressed to Mr. Ron Hochstein, Lundin Gold, from Lexim Abogados, 6 May 2016, 18 p.

This information is used in Section 4 of the Report. The information is also used in support of the Mineral Resource estimate in Section 14, the Mineral Reserve estimate in Section 15, and the financial analysis in Section 22.

3.3 Surface Rights

The QPs have not independently reviewed ownership of the Project area and any underlying property agreements, mineral tenure, surface rights, or royalties. The QPs have fully relied upon, and disclaim responsibility for, information derived from Lundin Gold and legal experts retained by for this information through the following documents:

- Borja, R, 2016: Surface Rights Opinion: letter addressed to Mr. Ron Hochstein, Lundin Gold, from Lexim Abogados, 23 May 2016, 20 p.

This information is used in Section 4 of the Report. The information is also used in support of the Mineral Resource estimate in Section 14, the Mineral Reserve estimate in Section 15, and the financial analysis in Section 22.

3.4 Environmental, Permitting and Social and Community Impacts

The QPs have fully relied upon, and disclaim responsibility for, information derived from Lundin Gold and experts retained by Lundin Gold for information related to environmental, permitting, closure planning and social and community impacts through the following documents:

- Aurelian, 2011a: Diseños Definitivos De La Vía Rural Desde La Troncal Amazónica Hasta El Sitio del Campamento FDN
- Aurelian, 2011b. Estudio vía Rural Desde La Troncal Amazónica Hasta El Sitio del Campamento FDN, Sistema de Coordenadas UTM-PSADS56, 23 hojas, Escala H1:1000 V 1:100
- Cardno-Entrix, 2013: Informe de Resultados-Monitoreo Biótico Semestral-Proyecto Minero Colibrí
- Cardno Entrix, 2016: Actualización del Estudio de Impacto Ambiental del Proyecto Minero Fruta del Norte, para la fase de Explotación e Inclusión de las Fases de Beneficio, Fundición y Refinación de Minerales Metálicos en el Área Operativa de la Concesión La Zarza (Cód. 501436) e Infraestructura Complementaria en las Concesiones Colibrí 2 (Cód. 501389) y Colibrí 4 (Cód. 501433), Además de la Explotación de Materiales de Construcción en la Concesión Colibrí 4 (Cód. 501433): report prepared by Cardno Enxtri for Aurelian, March 2016.
- Entrix Inc, 2010a: Actualización del Estudio de Impacto Ambiental para la Fase de Exploración Avanzada, Proyecto “Fruta del Norte” Concesión Minera Colibrí
- Entrix Inc, 2010b: Monitoreo de Fauna en el RVSEZ, Colibrí, FDN y Goldmarca
- Entrix Inc, 2014: Monitoreo Biótico de los Proyectos Mineros Emperador, Colibrí y La Zarza, Primer Semestre 2014
- Entrix Inc, 2015: Rescue and Relocation Plan for Biotic Species - Fruta del Norte Mining Project: report prepared for Aurelian, July 2015
- Walsh, 2011: Estudio de Impacto Ambiental para la Fase de Beneficio, Fundición y Refinación de Minerales Metálicos, Proyecto Minero “Fruta del Norte”, Concesión Minera La Zarza: report prepared for Aurelian Ecuador S.A.
- Rykaart, M., 2016: Fruta del Norte Feasibility Study: Preliminary Mine Waste Rock Closure Cover Design: memorandum prepared for Aurelian by SRK Consulting, 14 March 2016, 11 p.
- SRK Consulting, 2016: Geochemical Compilation Report, Fruta del Norte Project (Draft): report prepared for Aurelian, March 2016, 494 p.

This information is used in Section 20 of the Report. This information is also used in support of the financial analysis in Section 22, the Mineral Resource estimate in Section 14, and the Mineral Reserve estimate in Section 15.

3.5 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Lundin Gold staff and experts retained by Lundin Gold for information related to taxation as applied to the financial model as follows:

- Ernst & Young, 2016: Taxation and Royalty Information in the NI 43-101 Technical Report prepared by Amec Foster Wheeler for Lundin Gold: letter prepared by Ernst & Young LLP for Lundin Gold, 29 April, 2016, 5 p.

This information is used in support of the financial analysis in Section 22, and the Mineral Reserve estimation in Section 15.

3.6 Markets

The QPs have not independently reviewed the marketing or metal price forecast information. The QPs have fully relied upon, and disclaim responsibility for, information derived from Lundin Gold staff and experts retained by Lundin Gold for this information through the following documents:

- Cliveden Trading AG, 2016: Marketing: note prepared by Cliveden Trading AG for Amec Foster Wheeler and Lundin Gold, 11 May 2016, 3 p.

This information is used in Section 19 and in support of the financial analysis in Section 22 and the Mineral Reserves estimate in Section 15.

Metals marketing, global concentrate market terms and conditions, and metals forecasting are specialized businesses requiring knowledge of supply and demand, economic activity and other factors that are highly specialized and requires an extensive database that is outside of the purview of a QP. The QPs consider it reasonable to rely upon Cliveden Trading AG for such information as the company is a global leader in commercial intelligence for the energy, metals and mining industries, and provides independent analysis and advice on assets, companies and markets to these industries.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Property Location

The La Zarza concession, which hosts the FDN deposit, is situated between approximately 9575900N to 9585000N and 781000E to 773000E of UTM zone 17S (PSAD 1956 datum).

4.2 Property and Title in Ecuador

Information in this subsection has been derived from the Mining Law of Ecuador statutes and regulations (2009), the United States Agency for International Development (USAID) land tenure and country profile for Ecuador (2011), Latin Lawyer (de la Torre and Carcelén 2015), and Coronel and Perez (Pino and Ocampo Velez, 2015). This information is in the public domain, and has not been independently verified by Amec Foster Wheeler.

Mining in Ecuador is principally governed by the Mining Law, issued on January 29, 2009, as amended, and the General Regulation of the Mining Law, issued on November 16, 2009, which regulates activity as a whole.

Mining activities under the Mining Act are classified based on production levels as: large-scale mining (underground: >1,000 t/d; open-pit: >2,000 t/d), medium-scale mining (underground: between 301–1,000 t/d; open-pit: between 1,001–2,000 t/d), small-scale mining (underground: <300 t/d; open-pit: <1,000 t/d), and artisanal mining. The Mining Law also sets out classifications for various mining stages over a mining life-cycle as: prospecting, exploration (initial and advanced, and economic evaluation of the deposit), exploitation, beneficiation, smelting, refining, commercialization, and closure.

The key administrative bodies include the Ministry of Mining (MOM) and the Mining Regulation and Control Agency (Arcom, using the Spanish acronym).

4.2.1 Mineral Title

Mining concessions can be obtained through public bidding process (new areas) or public auction (where an area has expired or has reverted to the state). Grant of a concession provides the holder with a mining title that allows an exclusive right to prospect, explore, exploit, beneficiate, smelt, refine, market and sell all existing minerals obtained from a particular area. Concessions can range from 1 ha to a maximum of 5,000 ha in size.

Under the current Mining Law which was implemented in 2009, concessions are issued with a 25-year term. This initial term can be varied by law, however, with each of the sequential mining phases set out in the Mining Law. Under the Mining Law, a concession's duration is divided into two stages: an exploration stage and an exploitation stage. The exploration stage is further subdivided into shorter phases (initial exploration, advanced exploration and economic evaluation) based on the achievement of stipulated milestones. Any failure to achieve these milestones and successfully advance to the next stage by the phase deadline can result in forfeiture of the concession.

The phases of exploration are as follows:

- Initial exploration: maximum period of four years. Once the initial exploration has been completed, and prior to initiating the advanced exploration phase, the Mining Act requires mandatory relinquishment of a portion of the total concession area
- Advanced exploration: maximum period of four years
- Economic evaluation: maximum period of two years, which may be extended for up to an additional two years. On completion of this work segment, an application for exploitation can be made to the Government of Ecuador. A resolution is issued by the Government if the exploitation application is approved.

Within six months of grant of the resolution, the concession holder has to sign an exploitation agreement with the Government of Ecuador, through the MOM. The exploitation agreement defines the terms, conditions, and time periods for the stages of construction and assembly, extraction, transportation and sale of the minerals obtained within the boundaries of the mining concession. If an exploitation agreement is signed, the concession term will be that negotiated under the exploitation agreement.

4.2.2 Surface Rights

Under Ecuadorian laws, surface rights are independent of mineral rights conveyed by mining concessions under the Mining Act. Agreements must be reached with surface-rights owners through acquisitions or easements and appropriate compensation is typically negotiated in the agreement.

4.2.3 Water Rights

Water rights are governed by the 2014 Water Resources and Water Use Organic Law. The Government agency in charge of water is the Secretaría Nacional del Agua (Senagua; Water National Secretariat). All Senagua decisions related to conferment of water use permits must be supported by the Agencia de Regulación y Control del Agua (ARCA; Water Regulation and Control Agency). Water use permits are granted for defined terms, and annual water usage fees must be paid. Permits define specific catchment points for usage monitoring purposes. Once a permit has been issued, a flow meter must be installed at each designated catchment point.

4.2.4 Fraser Institute Survey

Amec Foster Wheeler has used the Investment Attractiveness Index from the 2015 Fraser Institute Annual Survey of Mining Companies report (the Fraser Institute survey) as a credible source for the assessment of the overall political risk facing an exploration or mining project in Ecuador.

Amec Foster Wheeler has relied on the Fraser Institute survey because it is globally regarded as an independent report-card style assessment to governments on how attractive their policies are from the point of view of an exploration manager or mining company, and forms a proxy for the assessment by industry of political risk in Ecuador from the mining perspective.

The Fraser Institute annual survey is an attempt to assess how mineral endowments and public policy factors such as taxation and regulatory uncertainty affect exploration investment.

Overall, Ecuador ranked 92 out of 109 jurisdictions in the survey in 2015.

4.3 Project Ownership

Lundin Gold conducts its business in Ecuador through a number of subsidiaries, including the following five active subsidiaries listed:

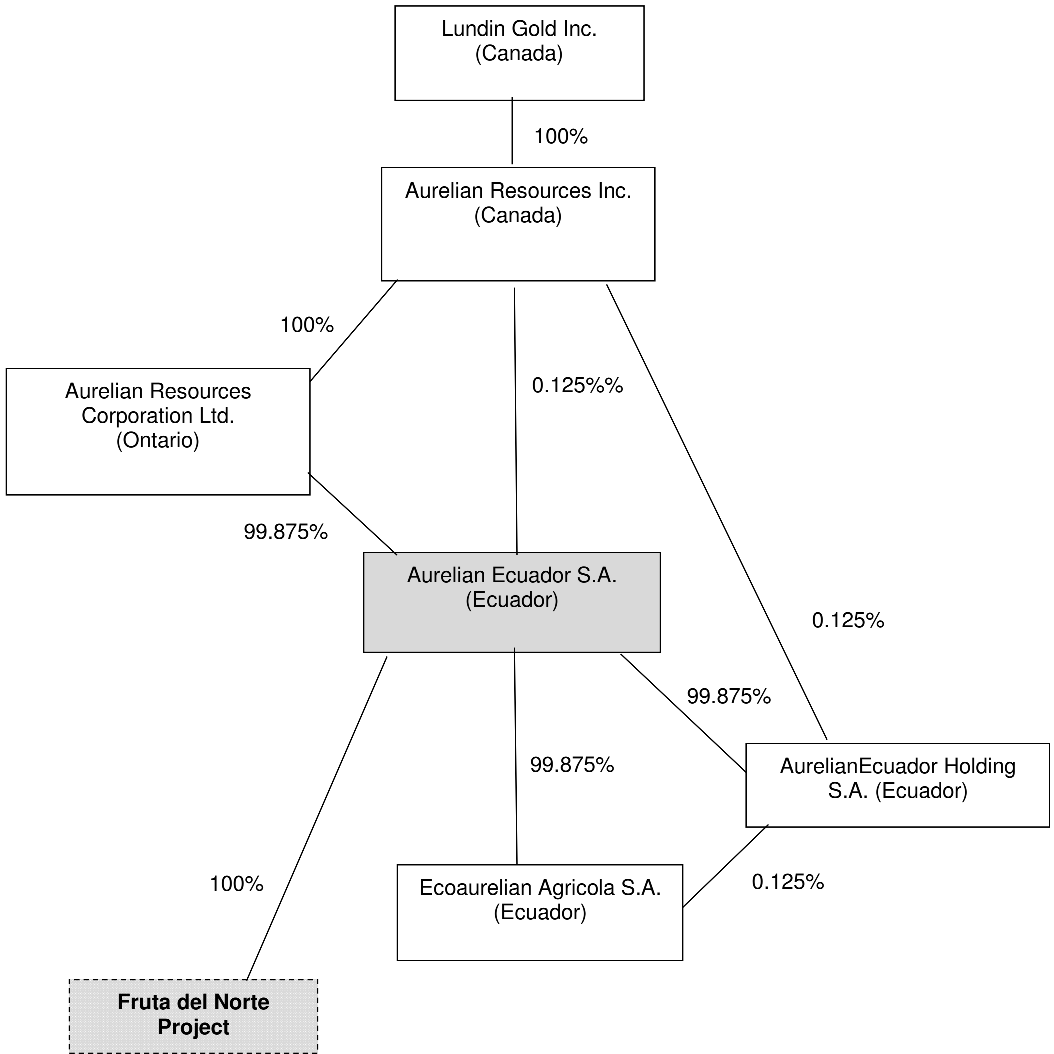
- Aurelian Resources Inc. and Aurelian Resources Corporation Ltd., which own all of the outstanding shares of Aurelian Ecuador S.A.
- Aurelian Ecuador S.A., which is the company's major operating subsidiary in Ecuador and the entity that holds the concessions underlying the FDN Project and the other concessions.
- Ecoaurelian Agricola S.A. which owns the land rights around the FDN deposit, and is a subsidiary of AurelianEcuador Holding S.A. and Aurelian Ecuador S.A.

Figure 4-1 shows the ownership interest.

4.4 Tenure History

In 2001, Patrick Anderson and Keith Barron co-founded Aurelian Resources Corporation Ltd., a Canadian private company, and began compiling a land package in the Cordillera del Cóndor region in the southeast corner of Ecuador through staking. Adding to its growing land position, in 2002 Aurelian Resources Corporation Ltd. purchased the La Zarza concession containing the FDN deposit from Amlatminas S.A. (Amlatminas), a private Ecuadorian company. A small concession, Reina Isabel, was also acquired during the same year. By the end of 2002, Aurelian Resources Corporation Ltd. had acquired an aggregate of 39 concessions.

Figure 4-1: Ownership Interest



Note: Figure courtesy Lundin Gold, 2016.

In 2003, Aurelian Resources Corporation Ltd. became Aurelian Resources and was listed on the TSX-Venture Exchange. It subsequently moved to the TSX in February 2007.

Kinross Gold Corp. acquired 100% of Aurelian Resources via takeover during 2008, and Aurelian Resources was delisted from the TSX in October 2008. Earlier in 2008, the Government of Ecuador imposed a moratorium on mineral exploration pending the development of new mining legislation. The Mining Law was passed at the start of 2009 and Kinross resumed exploration at the Project.

In June 2013, Kinross elected not to continue with the Project, citing unsuccessful negotiations with the Government of Ecuador for the exploitation phase.

In 2014, Lundin Gold (then named Fortress Minerals Corp.) purchased Aurelian from Kinross, thereby acquiring Kinross's land position in Ecuador. At the time of acquisition, Kinross's land position included 39 mining concessions that covered an area of approximately 86,000 ha.

Lundin Gold rationalized the concession holdings during 2016. During the rationalization process, a number of concessions were relinquished, and some concessions were incorporated into the La Zarza concession. An area of the La Zarza concession which overlapped with the Refugio de Vida Silvestre El Zarza (La Zarza Wildlife Refuge) was reduced. Lundin Gold also identified some small artisan concessions that were excised from the overall land holdings.

4.5 Mineral Tenure

Lundin Gold currently holds 31 mining concessions that cover an area of approximately 74,855 ha (Table 4-1 and Figure 4-2). The FDN deposit is hosted in the La Zarza concession.

Lundin Gold's mining concessions have different expiry dates. The expiry dates indicated in Table 4-1 reflect the remaining term of each concession from their registration in the Mining Registry under the current Mining Law. This expiry date assumes that each concession will advance through exploration to exploitation. If an exploitation agreement is signed in respect of a concession, the concession term would change, and the term would be the one negotiated under the agreement. The most imminent deadline to allow each of Lundin Gold's concessions to proceed to the next stage of development is also shown on Table 4-1.

Obligations that must be met to retain a concession include:

- Payment of annual holding fees
- Completion of annual reports on work completed and proposed investment plans.

The majority of the mining concessions form a large, mostly contiguous block that extends from the Nangaritza River eastward to the international border with Peru. Two small artisanal mining leases have been excised from the La Zarza and one from Victoriana concessions (refer to Figure 4-2).

In addition to the mining tenures, Lundin Gold holds two construction materials concessions (Table 4-2) that total about 237 ha.

In Ecuador mining concessions are "map-staked", and boundaries are defined by UTM coordinates.

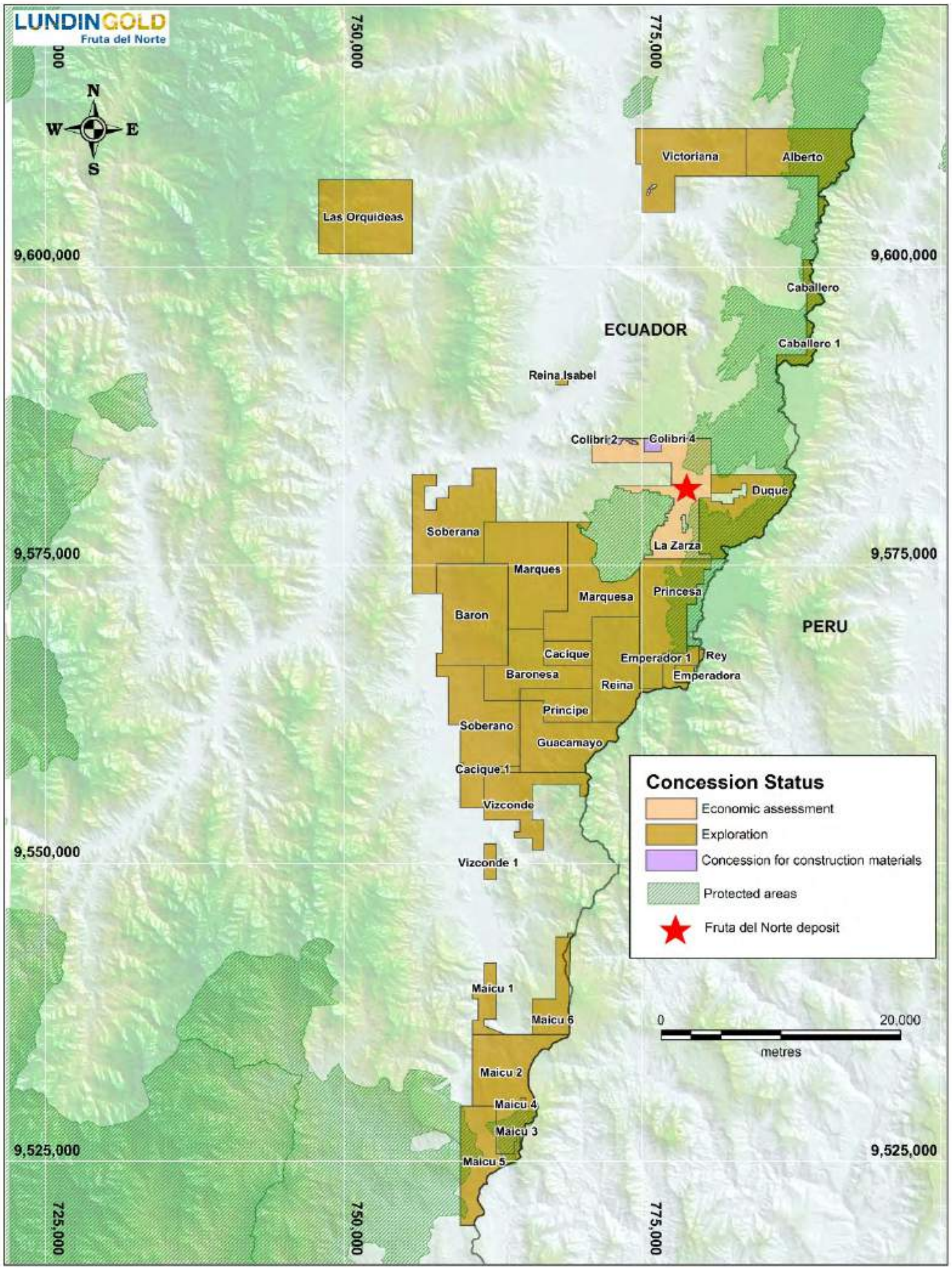
Table 4-1: Mineral Tenure

Concession Name	Holder	Date of Grant	Title Area (ha)	Registration Information	Deadline to Pass to Next Stage	Expiry Date
Alberto	Aurelian Ecuador SA	16 April 2010	3,799.86	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	13 July, 2032
Baron	Aurelian Ecuador SA	15 April, 2010	4,850.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	22 May, 2032
Baronesa	Aurelian Ecuador SA	15 April, 2010	3,000.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	22 May, 2032
Caballero 1	Aurelian Ecuador SA	16 April, 2010	459.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	15 June, 2032
Caballero	Aurelian Ecuador SA	16 April, 2010	376.77	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	7 June, 2032
Cacique 1	Aurelian Ecuador SA	16 April, 2010	150.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	15 June, 2032
Cacique	Aurelian Ecuador SA	16 April, 2010	800.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	15 June, 2032
Duque	Aurelian Ecuador SA	15 April, 2010	3,669.76	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	27 August, 2016	16 June, 2032
Emperador 1	Aurelian Ecuador SA	15 April, 2010	599.14	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	7 November, 2016	8 June, 2031
Emperadora	Aurelian Ecuador SA	15 April, 2010	210.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	7 November, 2016	8 June, 2031
Guacamayo	Aurelian Ecuador SA	15 April, 2010	3,288.82	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	10 December, 2018	14 September, 2032
La Zarza	Aurelian	15 April, 2010	4,661.92	Incorporated with the Third Public Notary of Quito's Protocol 23 April,	17 June, 2016	9 October, 2031

Concession Name	Holder	Date of Grant	Title Area (ha)	Registration Information	Deadline to Pass to Next Stage	Expiry Date
	Ecuador SA			2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010		
La Orquideas	Aurelian Ecuador SA	16 April, 2010	4,898.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	19 July, 2032
Maicu 1	Aurelian Ecuador SA	16 April, 2010	843.84	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	28 February 2034
Maicu 2	Aurelian Ecuador SA	16 April, 2010	3,226.47	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	28 February, 2034
Maicu 3	Aurelian Ecuador SA	16 April, 2010	945.14	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	28 February, 2034
Maicu 4	Aurelian Ecuador SA	16 April, 2010	83.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	28 February, 2034
Maicu 5	Aurelian Ecuador SA	16 April, 2010	2,710.14	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	28 February, 2034
Maicu 6	Aurelian Ecuador SA	16 April, 2010	1,596.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	3 November 2033
Marques	Aurelian Ecuador SA	15 April, 2010	4,900.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	10 December, 2018	12 June, 2032
Marquesa	Aurelian Ecuador SA	15 April, 2010	3,833.90	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	16 June, 2032
Principe	Aurelian Ecuador SA	15 April, 2010	1,320.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	16 June, 2032
Princesa	Aurelian Ecuador SA	15 April, 2010	4,234.37	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	7 November, 2016	12 June, 2032

Concession Name	Holder	Date of Grant	Title Area (ha)	Registration Information	Deadline to Pass to Next Stage	Expiry Date
Reina Isabel	Aurelian Ecuador SA	16 April, 2010	50.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	15 June, 2032
Reina	Aurelian Ecuador SA	16 April, 2010	3,433.44	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	12 June, 2032
Rey	Aurelian Ecuador SA	16 April, 2010	37.81	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	21 May, 2032
Soberana	Aurelian Ecuador SA	15 April, 2010	4,900.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	10 December, 2018	22 May, 2032
Soberano	Aurelian Ecuador SA	16 April, 2010	4,643.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	7 June, 2032
Victoriana	Aurelian Ecuador SA	16 April, 2010	4,446.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	9 December, 2018	13 July, 2032
Vizconde I	Aurelian Ecuador SA	16 April, 2010	300.00	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	7 June, 2032
Vizconde	Aurelian Ecuador SA	16 April, 2010	2,588.33	Incorporated with the Third Public Notary of Quito's Protocol 23 April, 2010; Registered with Agency for Mining Regulation and Control – Zamora Province 18 May, 2010	11 December, 2018	7 June, 2032

Figure 4-2: Tenure Location Plan



Note: Figure courtesy Lundin Gold, 2016.

Table 4-2: Construction Materials Tenure

Concession Name	Holder	Date of Grant	Area (ha)	Registration Information	Term
Colibrí 2	Aurelian Ecuador S.A.	June 17, 2013	83	Incorporated to the Third Public Notary of Quito's Protocol on June 20, 2013; Registered on July 3, 2013 with the Agency for Mining, Regulation and Control, Zamora Province	22 years, 6 months and 16 days since the date of registration in the Mining Registry
Colibrí 4	Aurelian Ecuador S.A.	February 4, 2016	154	Incorporated to the Seventeenth Public Notary of Quito's Protocol on February 10, 2016; Registered on February 19, 2016 with the Agency for Mining Regulation and Control, Zamora Province	22 years, 6 months and 16 days since the date of registration in the Mining Registry

4.6 Surface Rights

An Ecuadorian mining concession is a personal right; distinct and independent from the ownership of land on which it is located, even when both belong to the same person. Surface rights must be obtained to support mining project development.

To date, 60 public deeds for surface rights have been signed, which cover a collective area of approximately 4,118.5 ha. At the Report effective date, one public deed remains in negotiation in support of the construction of the access road from El Pindal to FDN.

Seven land easement agreements have been concluded; these cover areas including the access road and construction of surface infrastructure to support mining activities. The term granted is equivalent to the duration of the La Zarza concession, or the Exploitation Agreement terms and its extensions.

One concession easement agreement has been concluded with Cóndor gold S.A. (Cóndor gold), to support construction and operation of the access road. The easement agreement is valid for as long as the underlying mining concessions held by Cóndor gold remain current.

Lundin Gold holds sufficient surface rights to allow construction and development of the planned mining-related infrastructure. Additional surface rights, if needed for accesses such as easements, could be acquired through negotiation, or by direct request to the Ministry of Mining to impose easements over the required lands.

4.7 Water Rights

Lundin Gold holds seven water rights under a number of water tenures that collectively allow for 97.25 L/s of extraction. Six rights were granted for exploration purposes, and one water right allows for human consumption.

Lundin Gold currently holds three water permits for the La Zarza concession, and one water permit that was originally within the Sachavaca concession, but since the 2016 tenure reorganization, now also covers a portion of the La Zarza concession. Outside the La Zarza concession, Lundin Gold holds a further three water permits.

A summary of the current permits is provided in Table 4-3.

Lundin Gold has applied for an additional water permit for the La Zarza concession for use of water for human consumption from three catchment points, and has requested a 17-year term for this permit.

The 2016 FS envisages that Lundin Gold will not be applying for an overall water permit for industrial usage for mining activities, since the water that is proposed to be used will be from secondary, not primary, sources (refer to Section 20.8).

4.8 Royalties and Encumbrances

A 1% net revenue royalty is payable in perpetuity on production from Lundin Gold's current mining concessions, including the La Zarza concession, under a royalty agreement dated November 16, 2007 among Lundin Gold's subsidiaries (Aurelian Resources Inc., Aurelian Resources Corporation Ltd. and Aurelian Ecuador S.A.) and two individuals, being Keith M. Barron and Patrick F.N. Anderson.

There are no other third party royalties, back-in rights, payments, or other encumbrances in favour of Lundin Gold.

Royalties that may be incurred when the definitive form of the exploitation agreement (the Definitive Exploitation Agreement) for the Project is concluded are discussed in Section 4.9.

4.9 Property Agreements

4.9.1 Government of Ecuador

During 2015, Lundin Gold and the Government of Ecuador worked collaboratively to establish the fiscal terms and conditions for the development of the Project. At the start of 2016, Lundin Gold announced that it had completed negotiations with the Government of Ecuador and had settled the Definitive Exploitation Agreement terms for the Project.

Table 4-3: Granted Water Permits

Permit	Total Volume (L/s)	No. Resolution	Date of Issue	Validity	Number of Catchment Points		Volume by Catchment Point (L/s)	Catchment Point Easting	Catchment Point Northing
Water Industrial Use Permit for Advanced Exploration	8.7	5640-2009-C	06/05/2011	Renewal is in progress	4	La Zarza	1.93	778506*	9583299*
							1.93	778316*	9583433*
							1.93	778201*	9583453*
							2.9	778228*	9583569*
Water Use Permit for Human Consumption	1.15	5181-2010-C	12/01/2015	Indefinite #	1	La Zarza	1.15	778540**	9580693**
Water Industrial Use Permit for Advanced Exploration	1.4	5181-2010-C	12/01/2015	12/01/2019	1	La Zarza	1.4	778553*	9581148*
Permit	**Total Volume (L/s)**	**No. Resolution**	**Date of Issue**	**Validity**	**Number of Catchment Points**		**Code Point**	**Catchment Point Easting**	**Catchment Point Northing**
Water Industrial Use Permit for Advanced Exploration	27	7135-2011-C	23/01/2012	23/01/2016 Renewal is in progress	9	La Zarza	CM5	778409	9585629
							CM6	778341	9585043
							CM7	778146	9584688
							CM8	778135	9584376
							CM9	778367	9584360
						Sachavaca	CM1	779172	9587465
							CM2	779066	9587067
							CM3	778741	9586499
Water Industrial Use Permit for Advanced Exploration	12	0002-SR-DHS-E	03/05/2013	03/05/2017	4	Duque	P01-DUQ	785664,0	9582366,0
							P04-DUQ	785925,0	9582362,0
						Duquesa	P02-DQS	785410,0	9582876,0
							P03-DQS	785836,0	9582636,0

Permit	Total Volume (L/s)	No. Resolution	Date of Issue	Validity	Number of Catchment Points		Code Point	Catchment Point Easting	Catchment Point Northing
Water Industrial Use Permit for Advanced Exploration	20	7117-2011-C	20/01/2012	20/01/2016 Renewal is in progress	5	Emperadora	CE1	778416,0	9565697,0
							CE2	778521,0	9566473,0
						Emperador 1	CE3	777908,0	9566953,0
							CE4	777285,0	9566181,0
							CE5	777065,0	9566319,0
Water Industrial Use Permit for Advanced Exploration	27	003-2012-SR-DHS-E	13/05/2013	13/05/2017	9	Princesa	P05-RB	775908,0	9569970,0
							P06-RB	776001,0	9570055,0
							P07-RB	776293,0	9570055,0
							P08-RB	775697,0	9569960,0
							P09-RB	775717,0	9565730,0
							P10-RB	776163,0	9565409,0
							P11-RB	776127,0	9565285,0
							P12-RB	775977,0	9565846,0
							P13-RB	776667,0	9565431,0

Notes: * co-ordinates using *PSAD56; ** coordinates using WGS84; # under the current water laws, an indefinite tenure must be converted to a term tenure, this conversion is yet to happen.

The key terms of the Definitive Exploitation Agreement are as follows:

- Through its wholly-owned subsidiary in Ecuador, Lundin Gold has negotiated the right to develop and produce gold from the FDN Project for 25 years; this right can be renewed
- Lundin Gold and the Government of Ecuador have agreed to an advance royalty payment of US$65 million, with US$25 million being due upon execution of the exploitation agreement. The balance of the payment will be due in two equal disbursements on the first and second anniversaries of the execution of the exploitation agreement
- Lundin Gold has agreed to pay the Government of Ecuador a royalty equal to 5% of net smelter revenues from production. The advance royalty payment is deductible against future royalties payable. It will be deductible against the lesser of 50% of the royalties payable, or 20% of the total advance royalty payment
- Extraordinary revenue tax (the Windfall Tax) will be calculated in the event that market prices exceed a stipulated base price for gold and for silver. The Government of Ecuador will tax the difference between net smelter revenue and what the revenue would be using the base price at a rate of 70%. The base price, which will be determined on a monthly basis, will be equal to the trailing 10-year average of the daily price of gold or silver, escalated by the US Consumer Price Index, plus one standard deviation
- The Windfall Tax will not apply until Aurelian has recouped all of the cumulative investment in the development of the FDN Project since its inception plus the present value of the actual cumulative investment incurred from signing of the exploitation agreement until the start of production
- The Government of Ecuador's share of cumulative benefits derived from the FDN Project will not be less than 50% (the Sovereign Adjustment). To the extent that the Government of Ecuador's cumulative benefit falls below 50%, Aurelian will be required to pay an annual Sovereign Adjustment. Each year, the benefits to Aurelian will be calculated as the net present value of the actual cumulative free cash flows of the FDN Project subsequent to the signing of the Definitive Exploitation Agreement, net of the cumulative investment incurred in the development of the Project from its inception until the date of the Definitive Exploitation Agreement. The Government of Ecuador's benefit will be calculated as the present value of cumulative sum of taxes paid including corporate income taxes, royalties, Windfall Tax, labour profit-sharing paid to the government, non-recoverable value-added tax, and any previous Sovereign Adjustment payments
- The Government of Ecuador and Lundin Gold have agreed on a mechanism for correcting any economic imbalance to these key terms which are the result of

changes in taxes, laws and regulations as provided under the Definitive Exploitation Agreement.

Lundin Gold expects Aurelian to sign the Definitive Exploitation Agreement within six months of the FDN Project being approved to move to the Exploitation Stage under the mining regulations.

4.9.2 Other Agreements

Coincident with the signing of the Definitive Exploitation Agreement, Lundin Gold expects that Aurelian will enter into an investment protection agreement with the Government of Ecuador, the objective of which is to provide legal and fiscal stability and protection to Aurelian for its investment in the FDN Project. At the Report effective date, Lundin Gold was in negotiations with the Government regarding the terms of this agreement.

4.10 Permits

Permitting considerations for the Project are discussed in Section 20.

4.11 Environmental Liabilities

The environmental status of the Project is discussed in Section 20. Artisanal mining activity in the area of influence of the Project is discussed in Section 6.

4.12 Social Licence

The social licence considerations for the Project are discussed in Section 20.

4.13 Comments on Section 4

In the opinion of the QP:

- Information from legal experts supports that the mining tenure held is valid and is sufficient to support the declaration of Mineral Resources and Mineral Reserves
- Lundin Gold holds sufficient surface rights to allow construction and development of the planned mining-related infrastructure. Additional surface rights, if needed for accesses such as easements, could be acquired through negotiation, or by direct request to the Ministry of Mining to impose easements over the required lands
- Lundin Gold holds water permits; however, the 2016 FS envisages that no overall water permit for industrial usage for mining activities will be applied for, since the water that is proposed to be used will be from secondary, not primary, sources

- The key terms of the Definitive Exploitation Agreement have been agreed upon. Lundin Gold expects that Aurelian will sign the Definitive Exploitation Agreement within six months of the FDN Project being approved to move to the Exploitation Stage under the mining legislation.

- Current permits have allowed exploration and associated supporting testwork to be conducted under appropriate laws. Additional permits are required for Project development, including the execution of the Exploitation Agreement. Lundin Gold is in the process of preparing a Phase Change Application to the Government of Ecuador in support of the phase change approval grant. Delays in the execution of the Exploitation Agreement may impact the Project execution plan as outlined in Section 24

- There are surface disturbances associated with artisanal workings within the Project area (refer to Section 6); there is an expectation that environmental contamination will be associated with these sites

To the extent known, Lundin Gold has advised Amec Foster Wheeler that there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property that have not been discussed in this Report.

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

5.1 Accessibility

The nearest city to the Project area is Loja, the fourth-largest city in Ecuador. The Project area is located about 139 road kilometres (approximately a four-hour drive) east–northeast of Loja. The closest serviced town to the Project is Yantzaza.

Vehicular access from Loja to the FDN site is via a 150 km long paved highway (Highway 45) to the town of Los Encuentros. Currently, a 40 km long gravel road connects Los Encuentros to the Project site.

Loja has daily scheduled air service from the national capital Quito, as well as from Ecuador's largest city and port Guayaquil. Maintained military airstrips at Zamora and Gualaquiza are available for use by chartered airplane and rendezvous with helicopters, for air access to FDN and the nearby Las Peñas exploration camp. The Las Peñas camp is the base for exploration activities at FDN.

Additional information on Project access is included in Section 18.

5.2 Climate

The climate of the Project area is characterized by wet weather throughout the year and was classified into two bio-climate sub-categories: temperate rainy and humid sub-tropical. The annual rainfall around 3,400 mm. The average annual temperature is between 16°C and 18°C, and remains fairly constant throughout the day.

Lower average daily temperatures and higher monthly rainfalls prevail at higher elevations such as the La Zarza concession. Currently some exploration activities may be curtailed during the rains.

Lundin Gold expects that any future mining activity will be conducted year round.

5.3 Local Resources and Infrastructure

The Project is currently isolated from major public infrastructure. Power for domestic purposes is available in San Antonio, and power access was extended to the Las Peñas Camp. Cell phone reception is locally available in the Project area on ridge crests and other high, open sites. Water is currently obtained from surface sources.

Infrastructure assumptions and the proposed infrastructure layout for the Project are discussed in Section 18 of the Report.

5.4 Physiography

The Cordillera del Cóndor is a mountain system situated east of, and parallel to, the axis of the Andes Mountains. It defines the international border with Peru in southeastern Ecuador. The Cordillera del Cóndor consists of heavily dissected, steep ridges that rise from the Zamora and Nangaritza River valleys (about 850 masl) to sharp ridges and flat-topped mesas, up to 2,400 masl in elevation, which lie along the Ecuador–Peru border. The majority of the Project area, including the La Zarza concession, lies in the highlands south of the Zamora River, east of the Nangaritza River, both of which flow into the drainage system of the Amazon River.

Nationally-protected areas are located to the northeast (Bosque Protector Cordillera del Cóndor or the Cordillera del Cóndor Protected Forest) and the southwest (Refugio de Vida Silvestre El Zarza or the El Zarza Wildlife Refuge) of the Project.

Tropical rainforest canopies most of the region except where cleared for agriculture in the river valleys and adjacent slopes. The flat-topped mesas along the international border are covered by low shrub and heath lands. Typically, over half-a-metre of composting vegetation overlies several tens of metres of saprolite.

Saprolite is produced by tropical weathering of bedrock to clay which variably preserves original rock textures. Landslides are common, transporting soil, weathered bedrock and vegetation down slope to locally expose relatively fresh rock on hill slopes. Variably-weathered bedrock is also locally exposed in mountain streams within ravines (quebradas).

5.5 Seismicity

A site-specific seismic hazard assessment was completed by URS Corporation (URS) in 2008, which concluded that “the site will be subjected to future strong ground shaking generated by large earthquakes” (URS, 2008).

The most significant known seismically active fault is about 18 km from FDN, and has a reported maximum magnitude M7.3 earthquake.

The standard US practice is to design mining facilities for approximately 1:500 and 1:2,500 return event periods and this was used as a consideration for infrastructure.

- The design of the surface facilities used the 1:500 and 1:2,500 year return event period, and incorporated design criteria from the Ecuadorian Code and the International Building Code
- Underground excavations were considered to be less vulnerable to earthquakes, as they cannot move independently of the surrounding rock. URS concluded that,

based on a 1:2,500 year return event, the local stress situation, and the approach of installing of ground support throughout the mine, the impact of an earthquake on the underground mining operation will be limited

- The tailings storage facility (TSF) was designed using a 1:10,000 year return event, due to the significant consequences that would result in the event of a failure.

Standard industry practice is to review and update a seismic hazard assessment every 10 years, or after a large earthquake has occurred. Following the earthquake in April 2016 that had an epicenter close to the coast of Ecuador and which caused serious damage in that area, a seismic review was carried out to confirm the design criteria. A high-level assessment of the seismic design was carried out assuming that the peak ground acceleration could increase. It was determined that this would have minimal, if any, impact on the structural design. Seismic design is more critical for structures than for the underground mine or the tailings storage facility (TSF).

Additional seismic deformation analysis will be completed at the detailed engineering stage for critical infrastructure.

5.6 Comments on Section 5

In the opinion of the QP:

- The planned infrastructure, availability of staff, power, water, and communications facilities, the design and budget for such facilities, and the methods whereby goods could be transported to the proposed mine, and any planned modifications or supporting studies are reasonably well-established, or the requirements to establish such, are reasonably well understood by Lundin Gold, and can support the declaration of Mineral Resources and Mineral Reserves

- There is sufficient area within the Project to host an underground mining operation, including mine and plant infrastructure, waste rock and tailings storage facilities

- It is a reasonable expectation that any additional surface rights that would be required to support Project development and operations can be obtained through appropriate negotiation

- It is expected that any future mining operations will be able to be conducted year-round.

6.0 HISTORY

6.1 Early History

The Cordillera del Cóndor was first explored by Spanish conquistadors in the 1500s. There is evidence that pre-Columbians mined both hard rock and alluvial gold in the area. Spanish mining activity ceased about 1620, following conflict with local Indian tribes that had been enslaved to work in the mines. Artisanal alluvial miners began to prospect the Cordillera del Cóndor as early as 1935, both in Peruvian and Ecuadorian territory.

6.2 Work Completed

A summary of the modern exploration activity on the Project is provided in Table 6-1.

Companies involved prior to Lundin Gold's Project interest included Minerales del Ecuador S.A. (Minerosa), from 1986–1992; Amlatminas S.A. (Amlatminas) from 1996–2002; Minera Climax del Ecuador, a subsidiary of Climax Mining Ltd. of Australia (Climax) from 1996–1998; Aurelian Resources Corporation Ltd. (Aurelian Resources) from 2003–2008; and Kinross Gold Corporation (Kinross) from 2008–2014. A location plan showing the prospects discussed in the table is included in Section 7.4.

Kinross completed a pre-feasibility study in 2009 (2009 Kinross PFS), and a feasibility study in 2011 (2011 Kinross FS). Lundin Gold undertook a feasibility study in 2015–2016, the results of which are documented in this Report.

6.3 Production

No commercial production has occurred from the Project.

Artisanal mining activity in the areas surrounding the Project and the Zamora Province in general has been long-standing.

Two areas of the Project, Castillo and Bonza–Las Peñas, were the subject of artisanal mining during the period 1993–1996. A small group of miners led by ex-Minerosa geologist, A. Cardenas, started sluicing alluvial materials from the Quebrada Astudillo area in 1996. Following the discovery of gold-bearing quartz vein float, operations shifted to processing gold-anomalous colluvium and insitu quartz veins. A total of 900 g Au was extracted over eight months of operations at this site (Montes, 1998). This is the only record of the amount of gold from artisanal production for the Project.

Following the departure of Climax, artisanal miners recommenced bedrock operations at Bonza–Las Peñas, and started similar mining operations at Aguas Mesas Norte and Sur. Exploration and exploitation of alluvial deposits on the Zarza, Machinaza and Blanco Rivers continued during Climax's tenure and remain ongoing.

Table 6-1: Exploration History

Company	Active Period	Work Completed
Minerosa	1986–1992	Establishment of a base camp on the east bank of the Blanco River, transportation of equipment to support alluvial mining, stream sediment sampling, and test pits excavated into alluvial terraces. Rock chip sampling, geological mapping and four Acker drill holes (each 15 m to 20 m long) were completed to evaluate primary gold mineralization exposed in the Quebrada Astudillo area, the site of the Castillo prospect (previously called the Ubewdy prospect).
Amlatminas	1996–2002	Generation of a topographic base map, stream sediment sampling (15 samples), rock chip sampling (152 samples), and geological mapping, in and near the Quebrada Astudillo area. Brief field assessments were undertaken by a number of companies in support of potential option agreements over the Project.
Climax	1996	Reconnaissance, leading to signing of an option agreement with Amlatminas.
	1997–1998	Work completed by Climax included gridding (total of 138 line km), geological mapping, stream sediment sampling (208 samples), regional and infill soil sampling (1,380 auger samples), rock chip and grab sampling (480 samples), test pits (658 pits), trenching (total 874 m; 223 samples), adit channel sampling at Bonza (seven adits; 72 samples), Induced polarization (IP) geophysical surveying (73.8 line km of gradient array, 2.15 line km of dipole and 36.5 line km of magnetometer), and core drilling programs (22 drill holes for approximately 3,562 m; 16 at Bonza-Las Peñas and six at Castillo, on the La Zarza concession. Work was primarily conducted over the Castillo, Bonza–Las Peñas, Princesa (Jardin del Cóndor), Rio Negra and Tranca Loma prospects, where precious and base metal geochemical anomalies were defined in areas that displayed features such as quartz veins with pyrite and local silicification and brecciation or clay–silica–pyrite alteration. The IP survey outlined a strong co-incident resistivity and chargeability anomaly above silicified conglomerates of the Suárez Formation. No drill testing was performed, and the concession reverted to Amlatminas in early 1999.
Aurelian	2002	Aurelian purchased the Project from Amlatminas. Confirmation chip sampling (20 grab samples).
	2003–2005	Outcrop examination, gridding, geological mapping, regional geochemical stream sediment sampling, rock chip, channel and grab sampling of outcrop, artisanal workings and trenches, a magnetometer and IP geophysical survey, and core drilling of prospects that either were known previously through Climax's work before 1999, or were discovered by artisanal miners in the period 1999 to 2002.
	2003	14 holes (1,161 m approx.) at Aguas Mesas Sur and Norte prospects*
	2004	34 holes (7,676 m) at the Bonza–Las Peñas, and Aguas Mesas Norte prospects. Nine holes (1,266 m approx.) at Puente prospect. Initial Mineral Resource estimate for the Bonza–Las Peñas area.
	2004–2005	Geological re-interpretation.
	2005	17 holes (3,256 m approx.) at the Bonza–Las Peñas, Tranca Loma and Castillo prospects.
	2006–2008	Core drilling, geological modelling and genesis studies, metallurgical testwork, and initial geotechnical investigations. Discovery of the FDN deposit in 2006. A first-time Mineral Resource estimate was prepared for Aurelian in late 2007. Regional exploration during the same time period comprised additional soil, rock chip and grab sampling, geological and structural mapping, genesis and modelling studies, and geophysical surveys.
	2006	48 core holes (23,579 m approx.) at FDN, Bonza and Las Arenas areas.
	2007	101 core holes (52,020 m approx.) at FDN, Las Arenas and Papaya; 12 core holes (3,730 m) at El Tigre.
	2008	47 core holes (23,609 m approx.) at FDN, Bonza and La Negra.
Exploration Moratorium	2008	On May 6, 2008, the Ecuadorian Government announced a moratorium on mining and exploration activity, pending development of a new mining code. This was lifted in 2009.

Kinross	2008–2009	Desktop studies to support a pre-feasibility study. Kinross also submitted core samples from 58 drill holes that had been completed prior to the imposition of the moratorium, but which had not been analyzed or incorporated into the Project database at the time of the 2007 Mineral Resource estimate.
	2009	Core drilling to support Mineral Resource delineation, assess infill targets, and provide samples for metallurgical testwork; program comprised four exploration drill holes (approx. 2,056 m), and five metallurgical drill holes. Updated Mineral Resource estimate. Completion of a pre-feasibility study (2009 Kinross PFS)
	2010	Core drilling to support updated Mineral Resource estimates, metallurgical and geotechnical drilling. Program consisted of 45 exploration drill holes (18,738 m approx.), four metallurgical drill holes (1,681 m approx.), and 19 geotechnical drill holes (4,142 m approx.).
	2010–2011	Completion of a feasibility study (2011 Kinross FS) based on an updated Mineral Resource and Mineral Reserve estimate.
	2011	Four exploration drill holes (2,457 m approx.) and 19 geotechnical drill holes (1,162 m approx.). This drilling included a long exploration hole to test the west side of the West Fault at depth (FN3490e01; 1,096 m), seven geotechnical holes (total 1,044 m) to provide information in the South Portal area, and three holes (FN 3835d01, FN3835d02, FN4150d01; total 1,356 m) to test for the northern strike extension of the FDN deposit. Results from the west exploration hole and the northernmost exploration hole were negative. However two of the northern step out holes confirmed mineralization in this area (FN3835d02, FN4150d01). The portal geotechnical holes confirmed previously known Bonza mineralization and justified additional exploration drilling in the Bonza North area.
	2012–2013	Underground-based deposit delineation drilling program was planned that focused primarily on the southern portion of the FDN deposit. The decline advanced approximately 600 m (734 m of total development), but no drilling was performed.
	2013	Kinross elected not to continue Project development in June.
Lundin	2014	Acquired Project interest
	2015	Drilling in support of feasibility-level studies; including 11 metallurgical drill holes (5,344 m approx.); 24 drill holes (4,296 m approx.) for geotechnical purposes including portal location, ventilation evaluation, and plant site design; 10 drill holes (1,216 m approx.) for hydrogeology; four structural drill holes (1,674 m approx.); and 15 drill holes (1,374 m) for the Hollin Borrow Pit area. Also completed geophysical surveys (IP), detailed prospecting and mapping of key targets, and initial exploration of those concessions that had limited available mapping and structural data but covered favourable geology. The principal objective was to better define and rank key targets and prepare these for drill testing in 2016. More regional initial exploration work activities focused principally on concessions known to be dominated by Misahuallí Formation volcanic rocks (the host rocks of the FDN deposit) in order to develop additional exploration targets. This work included initial geological mapping and prospecting, as well as soil geochemical surveys to define new target areas.
	2016	Completion of a feasibility study (2016 FS); updated Mineral Resource and Mineral Reserve estimates. Channel sampling completed on two targets (Emperador and Robles) in advance of drilling. Key targets were optimized and prepared for drilling, with drilling planned for May 2016.

Note: The mineral concessions at the time of the Aurelian and Kinross programs included concessions that Lundin Gold is formally relinquishing at the Report effective date. The Aguas Mesas Sur and Norte prospects south of the FDN deposit are not within the current tenure holdings at the Report effective date.

Anecdotal evidence suggests that exploration for and artisanal production from alluvial and/or colluvial sources on drainages within and near the La Zarza concession has continued since 1998, largely by Columbian *mineros informales*. The mining methods used can be two-man dredges or via rudimentary tunnels. The development of bedrock mining operations at Las Peñas, Aguas Mesas South, and Aguas Mesas North indicates that exploration by the *mineros informales*, presumably by panning alluvial and colluvial material, was successful.

Lundin Gold monitors the artisanal mining activity on a quarterly basis. There have been fluctuating levels of formal artisanal mining activity in the Project area, with formalized operations respecting both fiscal and environmental regulations. There has also been illegal artisanal mining activity. In these cases, legal complaints are lodged with the mining regulator (*amparo administrativo*) and the State takes the appropriate actions to ensure that illegal activities cease.

During the RPA site visit in 2016, the RPA QP observed some active artisanal workings.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

The Cordillera del Cóndor region consists of sub-Andean deformed and metamorphosed Palaeozoic and Mesozoic sedimentary and Mesozoic arc-related lithologies that formed between the eastern flank of the Cordillera Real, and west of the flat-lying strata of the Amazon basin.

Intruding the sub-Andean rocks is a composite I-type batholith, the Zamora Batholith, which has an elongate north–northeast axis that parallels the Ecuadorian Andes for over 200 km, extending into northern Peru. The batholith is considered to be the plutonic expression of a Jurassic-aged, subduction-related, continental magmatic arc established on the western margin of the Amazon craton. In the area of the FDN deposit, the batholith consists of phases of monzonite, diorite and granodiorites with local porphyritic and aplitic dikes and breccia zones.

Dominantly andesitic volcanic rocks correlated with the Zamora Batholith intrusive suite are conventionally assigned to the Misahuallí Formation. The Misahuallí Formation is a melange of volcanic, volcaniclastic/epiclastic and intrusive rocks that range in composition from alkali basalt to dacite and crop out as approximately north–south-aligned supra-crustal pendants within the largely contemporaneous Zamora Batholith.

The intermediate to mafic dikes and porphyries that locally intrude the batholith and Misahuallí Formation are conventionally interpreted to be coeval. Felsic dykes occur locally in the south end of the Project within the Princesa and Emperador 1 concessions.

Pre-Andean arc sedimentary and volcanic belts flank, and locally occur within, the batholith. The arc was denuded before Early Cretaceous deposition of alluvial to shallow-water conglomerate and quartz sandstone of the Hollín Formation. Within the Project area, mesa-like outliers of Hollín Formation quartz arenite may be as much as 110 m high, fronted by vertical escarpments.

A marine transgression is indicated by the deposition of overlying Early to Mid-Cretaceous mudstones and limestones. Late Cretaceous to Cenozoic uplift shed voluminous amounts of detritus from the emerging Andes Mountains across the region (Prodeminca, 2000; Quispesivana, 1996).

Jurassic rifting during arc formation is suggested by volcanic- and sediment-filled grabens and half-grabens preserved in the batholith (Prodeminca, 2000). Uplift and denudation of the region exposed large areas of Zamora Batholith before deposition of the Early to Mid-Cretaceous cover (Litherland et al., 1994). The subsequent subduction-related Andean orogeny deformed the sub-Andean units into a back-arc fold and thrust belt. The Cretaceous cover is gently warped around northeast-striking fold axes, although the predominant structures in the region are fault zones. Major drainages commonly follow north- to northeast-striking faults. Cretaceous cover rocks are exposed at variable topographic elevations in the region, and the overall distribution and elevation of the cover is controlled at least in part by north-, east-, northwest- and northeast-striking structures.

The regional Las Peñas fault zone is an important structural control on mineralization in the Cordillera del Cóndor. It strikes north–south and can be traced for at least 80 km. A step-over along the predominantly sinistral strike-slip fault zone led to the development of a pull-apart basin where the FDN deposit developed at the northeastern corner. This basin, termed the Suárez pull-apart basin, is filled with conglomerate-dominated epiclastic and volcaniclastic rocks and lesser lavas that collectively constitute the Suárez Formation.

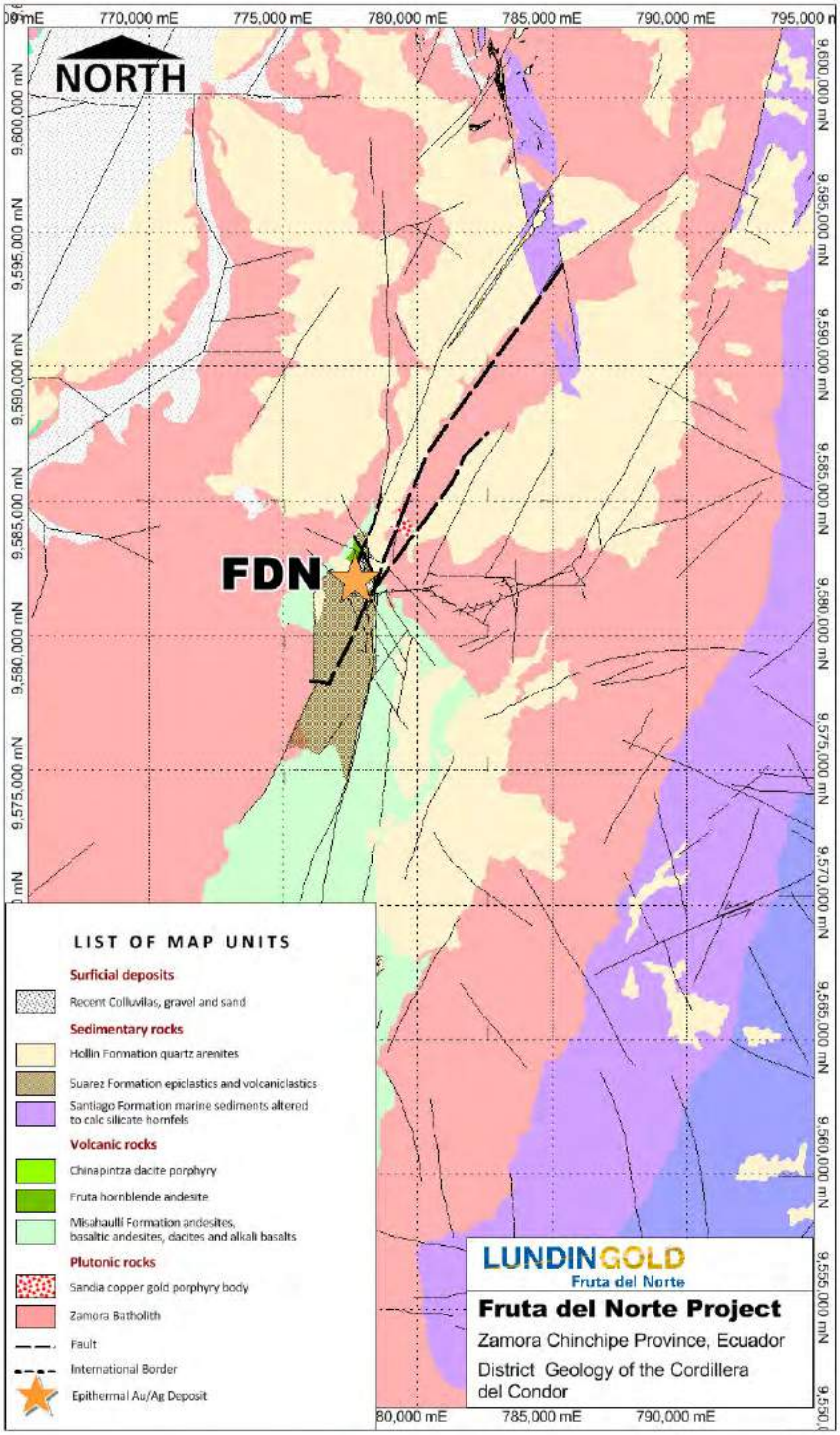
The location of the FDN deposit at the intersection of the north-trending Las Peñas fault zone, the northeast-trending Rio Blanco Fault and other east–west orientated lineaments attests to the distinct structural context of the epithermal system, which is assumed to have been localized along a precursor normal fault during the incipient stages of pull-apart basin evolution. Collectively, the faults that define the pull-apart basin are inferred to have undergone complex histories of normal, reverse and strike-slip motion, although kinematic criteria for the amount, direction and relative history of displacements have yet to be determined. Normal dip-separation of stratigraphy is associated with extension within the pull-apart basin. In particular, post-Cretaceous faulting has displaced the Hollín Formation forming an apparent horst and graben-like relief throughout the Cordillera del Cóndor with a substantial range of stratigraphic height imposed on individual Hollín Formation mesas in excess of 1 km.

A district-scale geology plan is provided in Figure 7-1.

7.2 Fruta del Norte

The FDN deposit is an intermediate sulphidation epithermal gold–silver deposit measuring approximately 1,670 m along strike, 700 m down dip and generally ranging between 150 m and 300 m wide. The top of the deposit is located beneath approximately 200 m of post-mineralization cover rocks. The eastern and western limits of the deposit are defined by two faults that together form part of the Las Peñas fault system that is thought to control the gold–silver mineralization. The southern limit of the mineralization along the fault system has not been fully defined by exploration activities.

Figure 7-1: District Geology of the Cordillera del Cóndor



Note: Figure courtesy Lundin Gold, 2016.

7.2.1 Lithostratigraphic Units

FDN lies buried beneath 130 m to 400 m of Suárez Formation and Hollín Formation cover, essentially burying and preserving the FDN epithermal system; hence, many aspects of the structure, stratigraphy and geological history of the deposit are interpreted from drill core and not from geological mapping.

The deposit is hosted by andesites of the Misahuallí Formation and feldspar porphyry intrusions, and has formed between strands of the Las Peñas fault zone (the East and West fault zones).

The lithostratigraphy is discussed in the following sub-sections, from youngest to oldest. A summary lithostratigraphic column for the general Project area is included in Table 7-1, and a geology plan for the FDN deposit is provided as Figure 7-2.

Hollín Formation

The Hollín Formation (logging code H) is composed predominantly of stacked, cross-bedded quartz sandstones, thinner intervals of interbedded mudstone and sandstone with subordinate shales and associated thin (typically 2–5 cm thick) seams of high-vitrinite coals and dark organic mudstones.

Throughout the Cordillera, Hollín Formation stratigraphy is disrupted by major north- and north–northwest-trending lineaments, and is locally tilted by up to 7º due to regional uplift and residual activity along the Las Peñas fault zone and other fault zones that intersect it.

Hollín Formation stratigraphy typically exhibits a horizontal to sub-horizontal attitude attaining a thickness of between 100 m and 110 m along the mesa highs. A tongue-shaped mesa of Hollín Formation separates the known extent of the FDN deposit from the Bonza–Las Peñas deposit to the south.

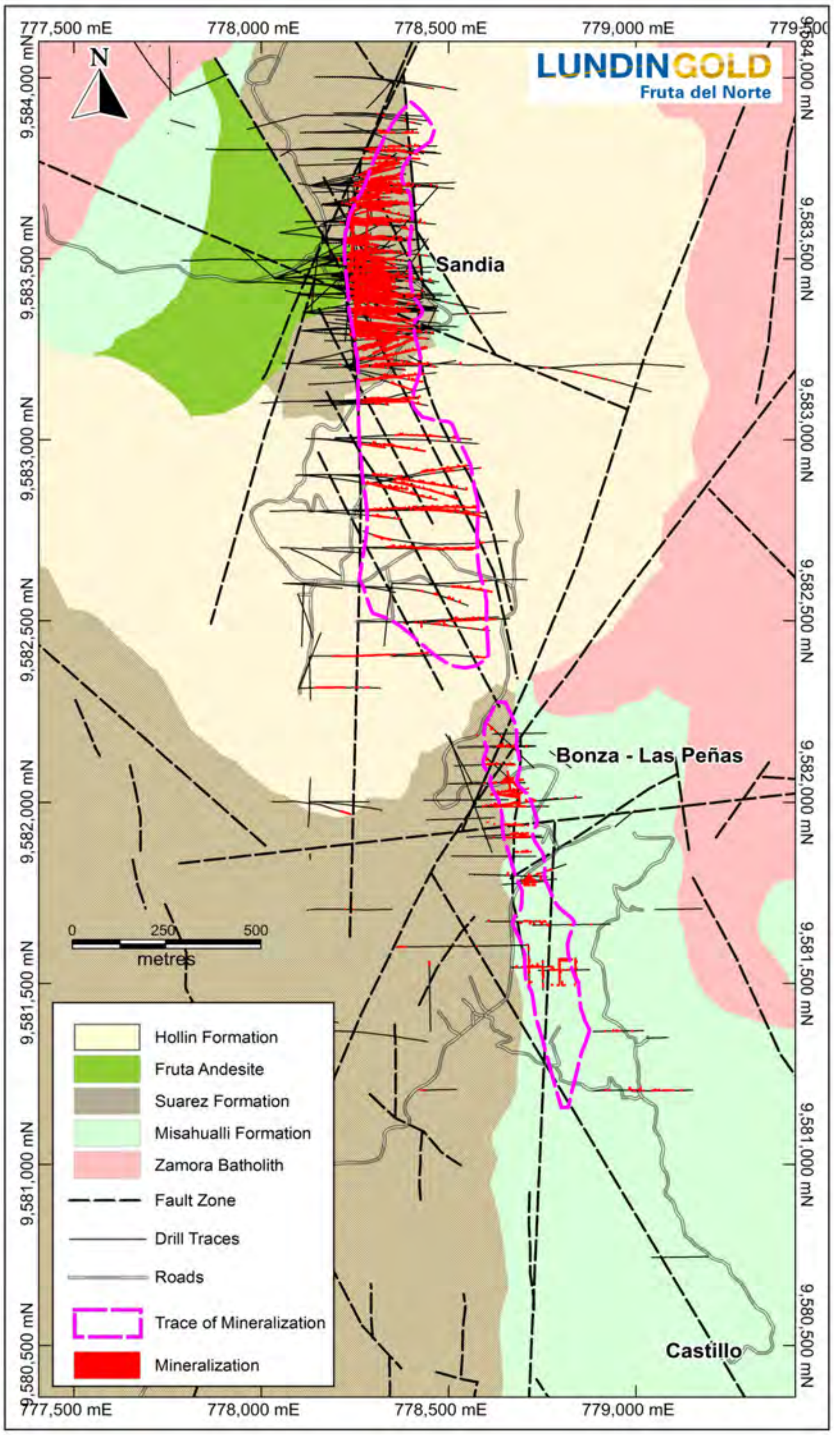
Suárez Formation

Spatially, the Suárez Formation is confined to the namesake pull-apart basin which extends over a surface area of approximately 26.4 km², is 2.2 km wide (east to west), and is at least 12 km in length (north to south). At FDN the Suárez Formation may reach as much as 400 m thick where it lies west of the West Fault, thinning to the east and disappearing at the East Fault Zone. The formation thins and wedges out to the north at the northern termination of the FDN deposit.

Table 7-1: Summary Stratigraphy of the FDN Deposit

Age		Formation	Member	Thickness (m)	Description	Logging Code
Early	Cretaceous	Hollín	Upper Sandstone	>60	Quartz sandstone; white, variable yellow brown and banded red-brown-purple iron staining	H
			Middle	≈20	Grey to black mudstone and siltstone, minor sandstone beds	
			Lower Sandstone	≈25	Quartz sandstone; white, variable yellow brown and red brown iron staining	
		~~ Regional Unconformity ~~~~~~~~~~~~~~~				
Late	Jurassic	Suárez	Fruta Andesite	≈250	Massive, light green to green-grey, fine grained to feldspar-hornblende porphyritic lava with exposed columnar joints	F
			Upper Mixed	≈250	Rhythmically bedded mudstone, siltstone, sandstone and conglomerate; lower contact is defined where polymict basal conglomerate becomes subordinate	Sm
			Machinaza Tuff	≈20	Brown to greyish to whitish massive beds, very fine grained with feldspar (minor hornblende and quartz) phenocrysts (<5mm); distinctive texture and colour differs from associated sedimentary beds. Strongly magnetic	Sv
			Lower Conglomerate	≈220	Massively bedded, immature (rounding, size and composition of clasts) polylithic conglomerate; matrix to clast-supported clasts (up to >1m core lengths) of andesite, andesite porphyry, medium grained granitoid, black mudstone and rare epithermal quartz vein and sinter fragments; minor interbeds of sandstone and Machinaza Tuff	Sc
		~~ Local Unconformity ~~~~~~~~~~~~~~~~				
		Misahuallí	Sinter-Mud Pool Facies	<20	Laminated to disaggregated pearl white to grey opal-A sinter, locally enriched in deep green celadonite. Includes dark grey sandy relict mud pool, geyserite deposits and surficial hydrothermal breccias	N, Nl, Nm, and Ns
			~~ Local Unconformity ~~~~~~~~~~~~			
Mid			Andesite	?	Dark green, massive, aphanitic to feldspar-hornblende porphyritic andesite; includes volcanic breccia; typically strongly altered at FDN; grades to feldspar porphyry	M

Figure 7-2: Surface Geology of the FDN and Bonza–Las Peñas Areas



Note: Figure courtesy Lundin Gold, 2016.

The fault-disrupted facies architecture of the Suárez Formation is characterized by four distinct stratigraphic sub-units listed in stratigraphic sequence as follows:

- Fruta Andesite (logging code F)
- Mixed Sequence (upper member; logging code Sm)
- Machinaza Tuff Member (logging code Sv)
- Polymict Basal Conglomerate (lower member; logging code Sc).

Basin-wide, the Suárez Formation shows a consistently mappable change upward from a lower unit consisting of a deep green, massive, polymict, coarse pebble conglomerate to a thinly bedded upper member which consists of >50% sandstone, siltstone and mudstone (with small coal seams) and subordinate conglomerate horizons. Conglomerates display a range of provenance, including clasts of diorite/monzonite and granodiorite derived from the Zamora Batholith, and black mudstone/siltstone clasts that are believed to be derived from the Triassic Pucará Formation in Peru. The lower conglomerate underlies and forms the eastern boundary of the basin over the entire length of FDN.

Within the formation are a number of intercalated syn-basinal ignimbrites and other tuffaceous horizons, together with later lavas. The ignimbrite-like Machinaza Tuff Member forms a basin-wide marker in the lower unit. The most important lava unit has been named the Fruta Andesite, and is a massive hornblende, plagioclase-phyric lava flow, and locally contains irregular enclaves of dioritic/monzonitic rock similar to the Zamora Batholith. The Fruta Andesite directly overlies the upper member but generally does not occur in contact with lower conglomerate or east of the West fault zone.

Sinter and Mud Pool Facies

The fault-disrupted sinter facies (logging codes N, Nl, Nm, and Ns) is located at the unconformable contact between the Suárez Formation and the underlying Misahuallí Formation. The 2 m to 5 m thick laminated silica sinter is typically white to pearly, composed of chalcedonic to opaline silica, with nodular, algal growths (stromatolite-like) and other biogenic or sedimentary features that are well preserved.

Although the sinter is typified by laminated facies, a disaggregated facies is equally common. The sinter is locally stained with bands and discordant vein-like bodies of deep green celadonite (iron-rich smectite), and veinlets or stockworks of chalcedony locally penetrate the carapace.

Misahuallí Formation

The Misahuallí Formation (logging code M) is dominated by a thick sequence of light greyish-green to dark green coloured hornblende–plagioclase-phyric andesitic and basaltic andesitic volcanic rocks, feldspar porphyritic andesitic intrusive rocks, locally voluminous phreatic breccia zones, and lesser planar intrusions. Subordinate amounts of intra-formational volcanogenic sandstones and other breccias are also present. To the southeast end of the Project area, rhyolites have also been described.

At FDN the Misahuallí Formation locally crops out as heavily-fractured wall rocks between parallel strands of the Las Peñas fault zone. Chalcedonic and manganese carbonate veins and stockworks in the Misahuallí Formation, together with chalcedonic breccias as float, are the main mineralized indications at surface in the Bonza–Las Peñas area of the epithermal system below. Chalcedonic veins with surface widths of as much as 0.5 m are locally exposed in the Machinaza River just south of FDN. The base of the Misahuallí Formation has not been intersected by drilling at FDN.

Footwall Feldspar Porphyry

A distinct, medium-grained feldspar porphyry body (logging code Ip) lies northwards from section 3200 N. This and other distinctive medium- to coarse-grained dikes and large intrusive bodies flanking the Misahuallí Formation are presumed to be phases of the composite Early to Late Jurassic Zamora Batholith. The feldspar porphyry crops out east of the East Fault Zone and underlies the Suárez Formation in the downthrown block to the west. The contact with the Misahuallí Formation andesites is locally sharp and commonly chilled. The intrusive contact dips between 65º and 70º to the west where it is not heavily fault-disrupted.

Drill hole data suggest that the intrusion is lensoid in shape, elongated north–south, and forms the footwall of the andesitic volcanic sequence. In places, multiple planar intrusions cut the volcanic rocks at the contact, which is almost entirely masked by intense veining and mineralization between sections 3200 N through 3800 N. This can cause difficulties in distinguishing the porphyry from Misahuallí Formation volcanic rocks of otherwise similar composition. High-grade, crustiform–colloform-textured veining is best developed at and above the intrusive contact in this segment of the deposit. The footwall porphyry narrows north of section 3900 N, where drilling indicates that it is also fault-disrupted and truncated. It is difficult to trace the porphyry intrusion south of line 3200 N where a complex mixture of volcanic rocks and intrusions prevail, rather than one coherent body. It appears that the unit trends eastward away from the mineralization. The intrusion appears to be at its widest through the central section of FDN, where the eastern margin of the porphyry has not yet been defined.

The feldspar porphyry intrusion may have originated as a crypto-dome emplaced through an actively accumulating volcanic pile, or alternatively, may be a contemporaneous sub-volcanic intrusion. The rheological contrast between intrusive and finer grained volcanic units to the west appears to have resulted in enhanced dilation and has allowed hydrothermal fluid flow along and adjacent to the contact during tectonism in the Las Peñas fault zone.

Phreatic Breccia

The phreatomagmatic breccia (logging code Xp) is the most prevalent breccia type at FDN. It is characterized by pale grey to white, sub-rounded to sub-angular fragments of both feldspar porphyry and hornblende-phyric andesite, supported in a fine grain silica–illite–pyrite ± carbonate altered rock-flour matrix. The fragments are often heavily illite-altered.

The dominant clast type reflects the host rock in which the breccia developed. Where the breccia occurs wholly within the feldspar porphyry, clasts are exclusively of that material. Conversely, where the breccia was emplaced along the feldspar porphyry/andesite contact, the breccia is polymictic. Where the breccia cross-cuts both rock units, it becomes progressively richer in host rock clasts with increasing distance from the lithological contact.

Epithermal veins are best developed along or adjacent to the breccia/wall rock contacts and can be very poorly developed within the rock flour matrix dominant breccia itself. Breccia zones are best developed on the east side of the deposit, near the intrusive/volcanic contact where it attains a stratigraphic height of some hundreds of metres and continues beyond the current depth of drilling.

Hydrothermal Breccia

Hydrothermal breccias are characterized by intense silica and silica-marcasite matrix breccias. The hydrothermal breccias are generally matrix supported, with polymictic clasts including vein fragments, and altered volcanic rock fragments. These show numerous episodes of cross-cutting veins and stockwork associated with multiple mineralizing events as a product of the hydrothermal process.

The hydrothermal breccias are directly associated with the mineralization process and contribute the majority of the gold reported as Mineral Resources. The breccias vary in colour from near black (central part of the deposit) to light grey, to white, depending in large part on their marcasite content within the silica component.

The hydrothermal breccias (logging code Xh) can be subdivided into two types:

- Hydrothermal explosion breccias that were formed during the main phase of epithermal mineralization. These have complex mutual overprinting features including clasts which are cut by new veins, stockwork, and other alteration.
- Vein breccias that are believed to be related to later-stage mineralization and cut all other epithermal features. Vein breccias are typically associated with crystalline quartz–calcite–barite and base metal sulphides and are commonly in the deeper parts of the deposit.

7.2.2 Weathering

Evidence for paleo-erosion of the FDN mineralized system is limited to the existence of sinter clasts (up to 1 m wide) and related chalcedonic vein fragments intercepted in drill holes through the lower member of the Suárez Formation. In the cover sequences, particularly the Suárez Formation, the upper 20–30 m of drill holes locally encounter very strongly weathered saprolite, continuing through zones of fracture oxidation to depth of about 50 m.

7.2.3 Alteration

Hydrothermal alteration consists primarily of a silica (quartz, chalcedony), sericite (illite, muscovite), clay–pyrite (± marcasite), ± carbonate mineral assemblage formed by relatively low-acidity fluids. The intensity of alteration is such that it is often difficult to conclusively discern the protolith given the levels of textural destruction. Overall, the deposit exhibits an alteration zonation downwards from the barren hot spring lithofacies (sinter and mud pool).

At or near the contact between the Suárez and the Misahuallí Formations, sericite alteration is zoned and forms a narrow, flat-lying to moderately-dipping paragonitic zone above the gold mineralization. This gives way to a phengitic zone that forms a broad envelope surrounding the gold mineralization. The gold zone itself consists of a second, steeper-dipping paragonite-dominated alteration zone with subordinate phengite. High-crystallinity sericite forms a broad zone immediately beneath and lateral to the gold-mineralized zone.

Clays include kaolinite, chlorite, and lesser smectite. Kaolinite forms a narrow (less than 100 m), flat-lying halo above the gold mineralization in the northern part of the deposit, and a thicker (as much as 250 m) halo above the gold mineralization to the south. The kaolinite halo occurs approximately 100 m to 200 m above the gold mineralization. Chlorite forms a broad (approximately 200 m thick) flat-lying halo above the gold mineralization, and also a steeply-dipping zone to the west of the gold mineralization.

Although the age relationships are complex due to repeated hydrothermal pulses, silica-pyrite alteration generally grades downward and outward into silica-sericite-pyrite alteration and finally to pyrite-carbonate alteration assemblages. Illite gives way to smectite in the upper parts, most notably within the hot spring lithofacies.

7.2.4 Structure

At the deposit scale, fault zones with a range of inclinations, orientations, offsets, fabrics and mineral associations were defined through trenching and road cuts prior to the discovery of the FDN deposit.

Four orientations of faults are present in the immediate area of FDN. The oldest identified faults are north-trending, and both bound, and cut, the gold domains at FDN. Later fault sets that trend northwest to north–northwest, northeast to north–northeast, and west also occur at FDN. Observed faults ranged from 0.1 m to 3.5 m in true width. Some faults consist of multiple, closely-spaced faults in fault zones ranging up to approximately 14.5 m true width. The features of the key faults are summarized in Table 7-2.

Sets of northeast- to north–northeast trending, northwest- to north–northwest-trending, and generally east-trending faults all post-date and offset mineralization as well as offsetting the older north-trending faults.

7.2.5 Up-Flow Zones

The location of FDN between faults related to basin extension is one of the prime considerations for the distribution of the gold mineralization in the deposit. The principal western up-flow zone is oriented north–south and is moderately to steeply west-dipping. The eastern up-flow zone has a north–northwest strike, a vertical to steep easterly dip, and extends southward to become the Bonza–Las Peñas–Castillo epithermal system, where it exits the basin. Although the eastern up-flow zone has an extensive strike extent (more than 3 km) along the Las Peñas fault zone, it does not appear to have been as dilatant as its western counterpart and contains much more disperse, lower-grade mineralization.

At the centre of the FDN deposit (line 3400N) the two up-flow zones converge, and it is here that the deposit contains the highest gold grades. The intersection also marks a mineralogical change, with the system transitioning from manganese–carbonate-rich to manganese–carbonate-barren and quartz-dominated styles. At the southern end where the two systems diverge, it becomes apparent that there were slight differences in fluid geochemistry, with the western up-flow zone generally higher in arsenic and antimony and the eastern up-flow zone containing higher silver (or silver:gold ratios), manganese, lead, and zinc.

Table 7-2: Summary, Major Faults and Fault Directions

	Fault Name	Features
Pre- to Syn-Mineralization Faults	West Fault	1 m to 10 m wide; cuts the Misahuallí Formation as a domain of foliated gouge and cataclasite, flanked by non-coherent breccias and fractured rocks. Is generally sub-vertical to steeply east-dipping and north-striking. West-side-down displacement of the Suárez/Misahuallí Formation contact, Machinaza tuff and the upper mixed member, and abrupt truncation of gold mineralization and epithermal alteration occur across this fault, which is consistent with the sedimentation pattern created during normal faulting in the extensional Suárez pull-apart basin. The West Fault and to some extent the Central Fault abruptly truncate and displace the auriferous zones in the FDN deposit. Also appears to represent a 'growth fault', which was active during basin sedimentation. The West Fault appears to have been one of the principal structural controls on main stage hydrothermal flow (the western up-flow zone). Weak silicification and barren, mainly black chalcedony within this structure also indicate pulses of late-stage fluids.
	West West Fault	Moderately west-dipping fault, located west of the West Fault. Movement on the fault is responsible for the apparent structural repetition of the Suárez–Mishualli package located west of the West Fault.
	Central Fault	Located between the West Fault and East Fault Zone and also displaces the FDN deposit. Varies in dip from steeply east to steeply west along its strike length, and is truncated by, or merges with, the West Fault at line 3600 N. Appears to have focused hydrothermal activity in the up-flow region at the south end of the feldspar porphyry unit over an extended time period. Defined by post-mineralization-aged brecciation and displacement of gold zones and epithermal veins, including the high-grade core of the deposit. Gold grades tend to be higher near the Central Fault on most sections. This relationship and the local north–northwest trend indicate this may be an extensional structure formed and mineralized during the main epithermal stage. Appears to have focused hydrothermal activity in the up-flow region at the south end of the feldspar porphyry unit over an extended period. Flexure of the Central Fault and its mergence with the West Fault contributed to the repeated flow of hydrothermal fluids in this region. Post-hydrothermal compression reactivated the Central Fault and modified the overall geometry of the deposit.
	East Fault Zone	A 50 m to 100 m wide zone of parallel faults each up to 2.5 m wide and separated by somewhat more competent rock. Characterized by fractured andesite and feldspar porphyry where it cuts Misahuallí Formation, and cataclasite where it cuts the Suárez Formation. Projections of individual faults upward into the Suárez Formation are poorly constrained due to a lack of drill holes, but west-side-down displacements are indicated.
Post-Mineralization Faults	East–northeast- to northeast-trending faults	North-northwest dipping normal faults, mapped in multiple locations. These normal faults often contain steep down-dip slickensides or relative dip-separation of stratigraphy consistent with normal fault movement.
	Northwest to north–northwest-trending faults	Consist of up to 3 m of fault breccia, cataclasite, gouge, silicified zones and narrow faults. Measured slickenlines as well as dip-slip separations of the Fruta Andesite, Suárez Formation, and modelled gold grade shells all suggest that these largely represent normal faults. Faults terminate against the Machinaza Fault, suggesting that they may pre-date the latest movement on the Machinaza Fault or in some cases may be splays off this fault.

Fault Name	Features
West-trending faults	On a regional scale, displacement of the base of the Hollín Formation by up to 400 m has been identified along west-trending faults identified in the area between the FDN and Bonza deposits. Displacement in the area of the Mineral Resource estimate is substantially less.

7.2.6 Mineralization

The mineralization is characterized by intense, multi-phase quartz–sulphide ± carbonate stockwork veining and brecciation over broad widths, typically between 100–150 m wide in the coherent central and northern parts of the system where the gold and silver grades are highest. Mineralized shoots are typically present within dilatant zones developed along inflections of vein strike or dip where the geometry permits maximum opening at the time of mineralization.

Zones of high-grade mineralization appear to be strongest and most consistent in the zone of boiling, brecciation, and fracturing localized along faults and the feldspar porphyry contact. Multi-phase, colloform and banded quartz–carbonate–(adularia)–(rhodochrosite)–(base metal) veins are observed in the central and lower portions of the zone where gold and silver grades are enhanced, and visible gold can be seen in many of these. At the base of the deposit, most high-grade mineralization appears to be associated with these discrete veins. To the south, the epithermal system broadens and the vein intensity disperses, reaching an overall width of 330 m but with a corresponding drop in gold and silver grades and an increase in the Au:Ag ratio. The mineralized envelope extends up to 700 m vertically and has a strike length of 1,670 m north to south. However, the cumulative strike length increases significantly to over 3 km further south when taking into account the Bonza–Las Peñas deposit and its dispersed continuation towards the Castillo prospect.

The mineralogy of FDN consists of chalcedonic to crystalline quartz, manganese-carbonates, calcite, adularia, barite, marcasite, and pyrite, as well as subordinate sphalerite, galena, and chalcopyrite, as well as traces of tetrahedrite and silver sulphosalts. Rare accessory minerals that have been identified (with varying degrees of confidence) include cinnabar, meta-cinnabar (both restricted to sinter), alabandite (MnS; only at depth), stibnite and arsenopyrite (both restricted to the basal Suárez Formation), pyrrhotite, hematite, proustite/pyrargyrite (Ag_3AsS_3), acanthite (Ag_2S), native silver, freibergite ($(Ag,Cu,Fe)_{12}(Sb,As)_4S_{13}$), boulangerite ($Pb_5Sb_4S_{11}$) and jamesonite ($Pb_4FeSb_6S_{14}$) and their oxidized products, valentinite or senarmontite (Sb_2O).

The bulk of the gold is microscopic and associated with quartz, carbonates and sulphides. Much of the gold is free milling, but the mineralization is moderately refractory, with approximately 40% of the gold locked in sulphides. However, coarse visible gold is commonly observed. Individual gold grains range from discrete specks less than 0.1 mm in size to broccoli-like, arborescent crystals >10 mm across. Visible gold occurs in all mineralized zones, in quartz or carbonate, as well as within pyrite or silver sulphosalt clusters.

A preliminary microprobe investigation of only a few samples shows that gold fineness is typically lower in the northern segment, roughly 750, whereas grains in the central segment have fineness values in excess of 900 (pure gold is 1,000). Silver sulphosalts are therefore interpreted to contain a percentage of the silver, enhancing the silver:gold ratios to approximately 1:1 in the upper part of the system. At depth and to the south, the system becomes increasingly silver rich relative to gold, with silver:gold ratios reaching 10:1. The increasing silver is also associated with increasing zinc and lead assays.

A cross-section showing the mineral domains within the FDN deposit is included as Figure 7-3.

7.3 Prospects/Exploration Targets

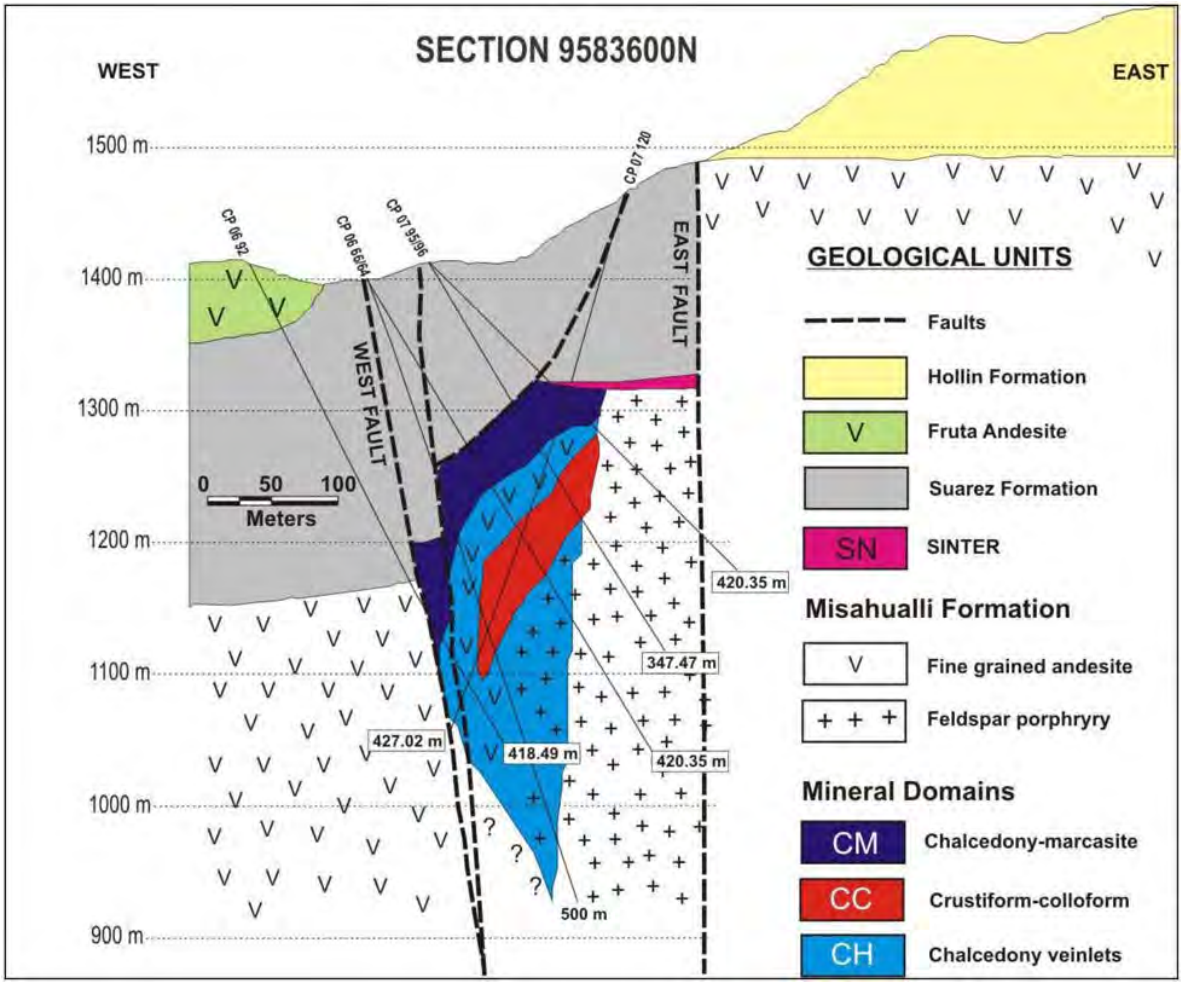
Descriptions of the major prospects and exploration targets in this section are derived from information from Henderson (2010) and from Lundin Gold. A location plan showing the prospects is included as Figure 7-4.

7.3.1 Bonza–Las Peñas

The Bonza–Las Peñas exploration prospect is located immediately south of FDN, and comprises the low-grade strike continuation of FDN south along the Las Peñas fault zone. The prospect consists of epithermal stockwork veining and breccias hosted within the Las Peñas fault zone by silica–sericite–pyrite-altered andesitic volcanic rocks of the Misahuallí Formation.

Mineralization comprises discrete quartz veins, quartz–rhodochrosite–manganese carbonate veins, cataclastic breccias, pyritic gouge, hydrothermal breccias, silicified pyritic zones, shatter breccias cemented by sulphides and possible magmatic-rooted intrusive breccia pipes. There is abundant textural evidence of multiple hydrothermal events, and any of the above mineralization types can mutually cross-cut each other. In places quartz veins can be followed cross-cutting the zones, but more often the veins have been tectonically milled and pulled apart into individual fragments. The gross pattern of mineralization is a network of anastomosing or basket-weave shear planes and slickensides surrounding otherwise intact pieces of country rock.

Figure 7-3: Simplified Cross-Section through the FDN Deposit



Note: Figure courtesy Lundin Gold, 2016.

Figure 7-4: Location Plan, Prospects

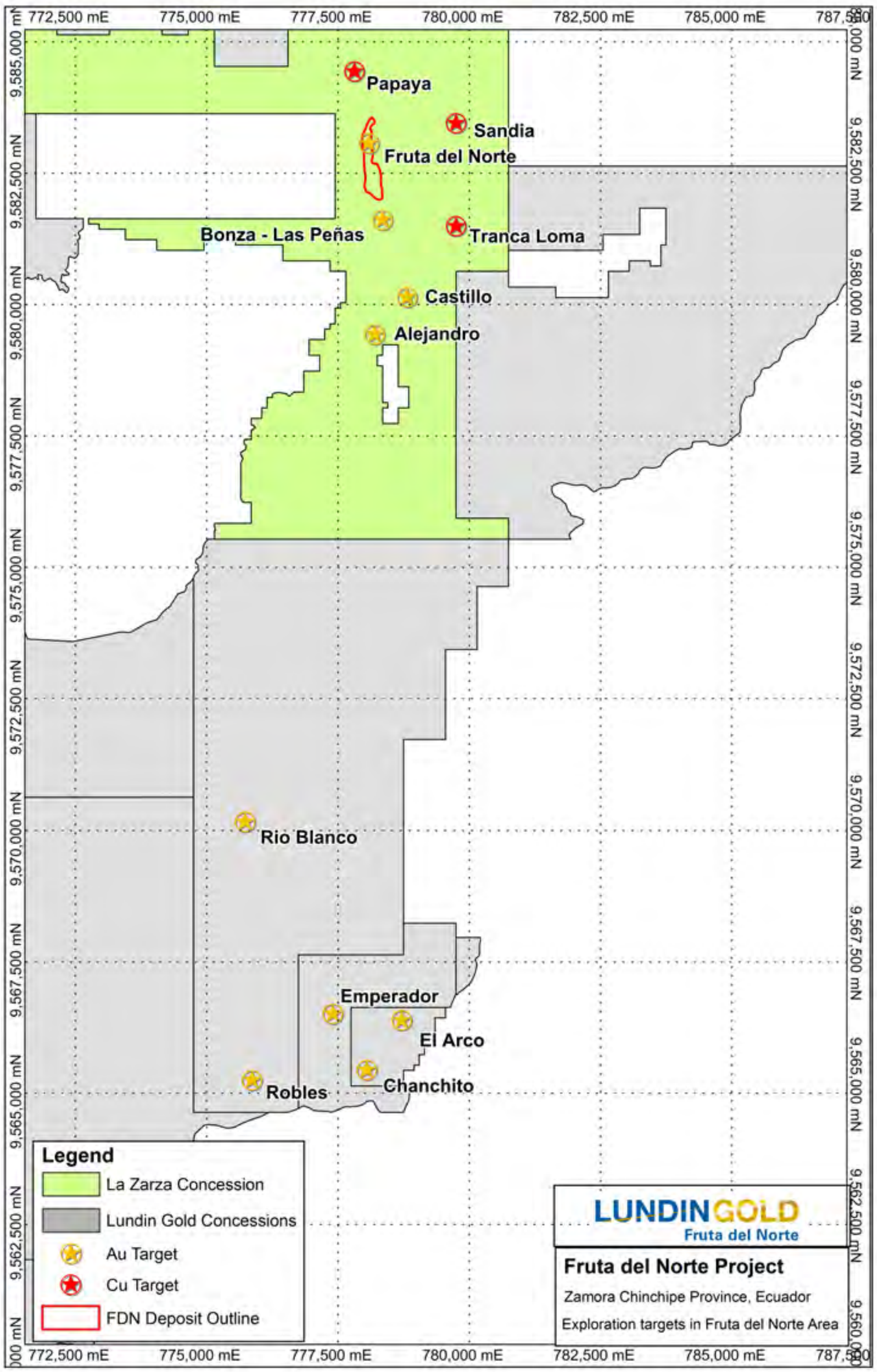


Figure courtesy Lundin Gold, 2016.

Quartz veins are variable in size but can reach as much as 5 m thick. There are various vein types: massive white quartz, white comb textured quartz, banded chalcedonic quartz, black cherty quartz, rhythmically-banded crustiform- and colloform-textured chalcedony, and rhodochrosite. In places, silica replacement of carbonate minerals is evident.

Within Bonza–Las Peñas, there are anomalous to significant concentrations of arsenic, antimony, manganese, zinc, mercury, lead and copper in addition to the gold and silver mineralization. Sphalerite and galena are locally abundant and the former can be yellow–brown or a dark red–brown in colour. Both are typically cross-cutting and are interpreted to have formed late in the paragenetic history.

7.3.2 Castillo

Exploration in 2005 defined a corridor of epithermal mineralization at the Castillo prospect (previously Ubewdy) that continues southward from Bonza–Las Peñas. The mineralization is hosted in andesitic volcanic rocks of the Misahuallí Formation, and developed as quartz–rhodochrosite veins. The prospect consists of a strong, laterally-extensive gold-in-soil anomaly.

The near-surface veins in the Misahuallí Formation volcanic rocks are interpreted to represent mineralization from deeper parts of an epithermal system with shallower parts possibly having been eroded long ago (this could also explain the gold occurring in streams, recovered by artisanal miners with dredges).

Directly to the west of the principal Castillo gold soil anomaly (Castillo West) and within the Suárez Formation conglomerate, soil geochemistry shows coincident arsenic, antimony, and mercury anomalies. Following the FDN structural interpretation, it is possible that higher-level mineralization is preserved in this area below the conglomerate within the same extensional pull-apart basin responsible for formation of the FDN deposit. No drilling exists to date to test this hypothesis.

7.3.3 FDN East Extension

The FDN East prospect consists of a broad zone of low-grade epithermal mineralization that starts approximately 300 m east of the FDN deposit. It is characterized by weak epithermal quartz–carbonate–sulphide stockwork veining and brecciation in andesite (often porphyritic) and within panels of the feldspar porphyry. Very limited drill testing in this area returned anomalous to low grade gold and silver values.

7.3.4 Alejandro

The Alejandro target represents a geochemical anomaly occurring in Suárez Formation conglomerate to the south of FDN and Castillo, along the southern extension of the Las Peñas fault zone. Although small in surface extent, the geochemical anomaly has strong coincidence of epithermal-style mineralization indicators in a favourable geological context, much like FDN itself. Values for arsenic, antimony and mercury are high and comparable to values that were returned over FDN.

The structural context is analogous to the previously described Castillo West area and more importantly to FDN, and could be the surface expression of similar underlying mineralization preserved below the Suárez Formation conglomerate. It is limited to the east by exposed volcanic rocks of the Misahuallí Formation, which are locally mineralized with pyrite and quartz veins.

7.3.5 Sandia

The Sandia prospect is located about 2 km to the east–northeast of FDN. It consists of anomalous copper values that were encountered in outcrop. Although primary textures are mostly destroyed by tropical weathering, silicification and quartz stockwork textures can still be distinguished. Oxidation has removed some of the primary sulphides, which are replaced by secondary sulphides including chalcopyrite, covellite, and chalcocite, together with limonite. There may well be a zone of supergene enrichment at the base of the saprolite.

Little work and no drilling has been done to date on the prospect, as porphyry copper targets were of secondary priority in past and current exploration efforts.

7.3.6 Papaya

The Papaya prospect is a copper–gold anomaly located approximately 0.9 km north–northeast of FDN, on the Las Peñas fault zone. Boulders of quartz–chalcopyrite–bornite veining locally contain coarse visible gold in a dull grey to blackened amorphous silica matrix. Sub-cropping quartz veins 5 cm to 25 cm wide trend approximately azimuth 340° and occur in close proximity to, or as survivor clasts or larger panels within, well developed cohesive gouge zones, which define strands of the Las Peñas fault zone. Analyses of float and sub-crop returned anomalous gold and copper values.

East-directed drill holes intercepted a number of broad zones (tens of metres true width) of crystalline quartz–sulphide veining hosted in Misahuallí Formation andesites that have been intensely propylitized. The andesite is heavily disrupted locally by 5 m to 15 m wide gouge and breccia zones that hampered drilling operations to the extent that two holes were lost close to or upon completion. Quartz monzonite porphyry dikes cut the andesites but are not as intensely propylitized as the andesitic host.

Zones of intermediate argillic alteration, along with widely spaced pyrite and anhydrite veinlets at depth, are suggestive of porphyry-style mineralization. The abundant chalcopyrite–bornite in surface samples is indicative of hypogene copper mineralization while the presence of chalcocite–covellite is most likely a supergene enrichment product.

The Papaya prospect is thought to contain two types of quartz veins, one mesothermal and intrusion-related, and the other of epithermal origin.

7.3.7 Tranca-Loma Porphyry

The Tranca–Loma porphyry occurs on the eastern margin of the Bonza-Las Peñas deposit as a northwest trending copper porphyry system that has been traced for over 2 km in length and across 600 m in width. The extent of the mineralization remains open along both strike directions. Shallow drilling has intersected disperse, often low-grade, porphyry copper mineralization.

7.3.8 Rio Blanco

The Rio Blanco target is located along the interpreted southern extension of the Las Peñas fault zone, approximately 12 km to the south of FDN. The target is best defined by a broad arsenic and antimony geochemical anomaly in soil samples, overlying dominantly andesitic volcanoclastic rocks of the Misahuallí Formation. The exploration target is for low sulphidation epithermal gold–silver mineralization. A cross-cutting resistivity anomaly based on induced polarization (IP) pole–dipole geophysics could be related to silicification and is planned to be tested by drilling during 2016.

7.3.9 Emperador

The Emperador target is located along the interpreted southern extension of the Las Peñas fault zone, approximately 16 km to the south of FDN. The target is best defined by coincident gold and arsenic anomalies in soil samples, overlying dominantly andesitic volcanoclastic rocks of the Misahuallí Formation and associated undifferentiated sediments. Although boulders of silica sinter had previously been found by Aurelian and Kinross within the Emperador target area, their source in outcrop was recently found in 2015. Gradient array IP data suggest subtle northwest and north–northwest structures in the target area (possibly associated with silicification) and coincident chargeability (possibly associated with sulphides) anomalies. During May 2016, four core holes were completed (1,551 m), and as of 24 May, 2016, assay results are pending. Additional work will depend upon the results of this first-phase evaluation.

7.3.10 Robles

The Robles target is the strongest soil sample gold anomaly outside of La Zarza and also shows a coincident arsenic anomaly. The target is located approximately 18 km to the south of FDN, slightly to the west of the interpreted southern extension of the

Las Peñas fault zone. The target area is dominated by intermediate composition volcanic rocks of the Misahuallí Formation with important north-northeast cross-cutting felsic dykes. Anomalous gold values are also found in the target area in boulders and outcrop in association with silica-marcasite veins and alteration.

A well-defined resistivity target at moderate depth (starting at about 200 m depth) based on IP pole–dipole may be associated with silicification. Drilling is currently under way. Two core holes had been completed as of 24 May, 2016, and an additional two holes are in progress, with 600.44 m drilled to date. Assay results are pending. Additional work will depend upon the results of this first-phase evaluation.

7.3.11 Chanchito and El Arco

The Chanchito target is characterized by high mercury values in soil samples defining the strongest-known soil mercury anomaly outside of FDN, with coincident anomalous arsenic values also in soils. The target is located approximately 18 km to the south of FDN and is associated with intermediate to felsic volcanic rocks of the Misahuallí Formation. Boulder samples indicated anomalous gold values were associated with silica–marcasite mineralization. The occurrence of the boulders is spatially coincident with resistivity targets (IP gradient array) which may indicate the occurrence of underlying silicification.

The geochemical anomalies appear to be associated with a north–northeast structure extending northward to the El Arco target (parallel to the Las Peñas fault zone). The north extension of this structure is highly anomalous in mercury and defines the El Arco target. The north–northeast structure generally separates dominantly felsic volcanic rocks to the east, from dominantly intermediate rocks to the west. After completing drilling on the Robles prospect, the 2016 exploration drilling program is planned to continue with evaluations of the Chanchito and El Arco targets.

7.4 Comments on Section 7

In the opinion of the QPs, the knowledge of the deposit settings, lithologies, mineralization style and setting, ore controls, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation.

8.0 DEPOSIT TYPES

8.1 Deposit Type

The setting, alteration mineralogy and mineralization characteristics of the FDN deposit are consistent with an intermediate sulphidation epithermal system as defined in Hedenquist et al., (2000). Some deposits with mostly low-sulphidation characteristics with respect to their alteration mineral assemblages have sulphide ore mineral assemblages that represent a sulphidation state between that of high-sulphidation and low-sulphidation deposits. Such deposits tend to be more closely spatially associated with intrusions, and Hedenquist et al., (2000) suggest the term ‘intermediate sulphidation’ for these deposits.

Intermediate-style epithermal systems are typically hosted in arc-related andesitic and dacitic rocks. Mineralization is silver- and base metal-rich, and associated with Mn-carbonates and barite. Sulphide assemblages in intermediate-style epithermal systems typically comprise tennantite, tetrahedrite, hematite–pyrite–magnetite, pyrite, chalcopyrite, and iron-poor sphalerite. Quartz can be massive or display comb textures. Sericite is common as an alteration mineral, but the adularia, more typical of low sulphidation systems, is rare to absent. Fluid inclusions range from 3–5% to 10–20% sodium chloride.

8.2 Comment on Section 8

The FDN deposit and many prospects that have been identified in close proximity to the deposit are classified as intermediate sulphidation-style epithermal systems on the basis of:

- The abundance of manganese-rich carbonate at FDN and the elevated base metal content (typically as iron-poor sphalerite and subsidiary tetrahedrite and chalcopyrite), are consistent with an intermediate sulphidation state
- The extensional tectonic setting of mineralizing fluid emplacement and the affiliation with intermediate magma types also complements the classification in terms of redox states
- Multiphase quartz–sulphide ± carbonate stockwork veining and brecciation over broad widths. Veins typically exhibit classic space-filling epithermal textures including intricate crustiform–colloform banding, and cockade and bladed calcite textures
- Mineralization comprises apparently free gold, refractory gold in sulphides, and is silver-rich

- Alteration comprises silica–pyrite alteration that grades outward and downward to silica–illite–pyrite alteration, and then to a silica (quartz, chalcedony)–illite–pyrite (±marcasite), carbonate mineral assemblage
- Sulphide assemblages include hematite–pyrite–magnetite and pyrite. Arsenopyrite, chalcopyrite, sphalerite, and galena have been noted.

Exploration programs that have used epithermal-style deposits as the geological model target have shown success in the FDN area, having discovered the FDN deposit and a number of prospects.

Two prospects have been identified (see Section 7.3) that may be indicative of porphyry–style mineralization, and a porphyry model is also applicable as an exploration geological model target in the wider Project area.

9.0 EXPLORATION

9.1 FDN Grids and Surveys

9.1.1 Kinross Exploration Grid

The Kinross exploration grid consisted of a north–south cut baseline with 100 m spaced east–west cut lines. The grid is based on UTM coordinates. The datum used in the survey network was originally the PSAD56 (Provisional South American) system applied to Zone 17S. Most data have been subsequently projected to UTM Zone 17S WGS84 using the using the EGM96 geoid to reference elevation.

As part of the 2009 Kinross PFS Kinross 159 of the then-total of 165 drill hole collars were re-surveyed. In addition, Leiva Engineering of Quito (Leiva) duplicated the northings and eastings of 25 road monuments and some of the old drill hole collars that had been surveyed by Kinross. It was found that the Kinross surveys corrected to an ellipsoid surface as opposed to EGM96 mean sea level; this resulted in the Leiva surveys having a 20 m difference from those of Kinross. As the 2009 Kinross PFS modelling efforts had begun using the ellipsoidal-corrected elevations, new infill-hole Z-coordinates used a 20 m constant addition to stay consistent with the original database.

All initial collar coordinates have been recalculated in the EGM96 system.

Leiva also established additional regional geodetic points in the Colibrí and Emperador concessions.

9.1.2 Ground Control Points

A ground control point at Las Peñas camp was established, guaranteeing a fixed “zero point” designated as GCP-01 (Ground Control Point-01). An Instituto Geográfico Militar (IGM) tie-in was set up on IGM point Los Encuentros-1 located 17.59 km west–northwest of Las Peñas, established (by the IGM) at Escuela Gabriela Mistral, in the village of Los Encuentros, Zamora. The Los Encuentros-1 data were purchased from the IGM in Quito. A tie-in to the International GPS System (IGS) was performed by the AUSPOS processing engine of the University of New South Wales, Australia.

9.1.3 LiDAR Surveys

In February 2008 Aurelian contracted Network Mapping UK to conduct a light detection and ranging (LiDAR)/orthophotographic survey of a priority area in the Project covering 402 km². An integral part of the LiDAR survey was the establishment of an independent survey network using long (>1 hour) static observation sessions using a dual frequency differential (DGPS) receiver.

A digital terrain model (DTM) survey set was acquired from IGM in 2005 that covers an area of 79.8 km². LiDAR data were acquired in February 2008 from a helicopter-mounted scanner. In 2010 Kinross commissioned Walsh Consultants (Walsh) to reprocess the LiDAR data with the purpose of reconstituting contours with corrected elevations. The LiDAR topography, orthophotos, Kinross survey and Leiva surveys have good agreement in northings and eastings; however, Walsh used the ellipsoidal-corrected elevations as a base reference.

9.1.4 Database Re-projection

The 2010 exploration grid was based on UTM Zone 17S coordinates using the PSAD56 datum. All data has since been re-projected to UTM Zone 17S using the EGM96 geoid to reference elevation. In 2010, Kinross retained Tetra Tech Wardrop (Wardrop) to assess the impact of implementing a new datum on collar coordinates, and on the subsequent translation of the geological interpretation (wireframes) to the new datum. Wardrop's assessment included various comparisons of re-surveyed holes and a visual verification of the corrected database with the LiDAR produced surface. No significant offsets were noted between the corrected data set and the LiDAR surface. Geological wireframe translation was based on the average offset from the drill hole coordinates.

References to section lines are abbreviated to the last four digits of the UTM northing. For example, the vertical section at 9,583,325 m north becomes vertical section 3325N.

Several holes drilled in 2010 and 2011 were assigned drill hole identifiers (IDs) based on the vertical section northings that used the previous projection. Therefore, those drill hole IDs do not match the current collar location.

9.2 Geological Mapping

Geological and structural mapping have been completed from regional (1:25,000 scale) to prospect scale (1:2,000). Mapping results were used to identify areas of quartz veining, silicification and sulphide outcrop that warranted additional work.

Data from remote sensing, geophysics, geological mapping and drilling were integrated to prepare an interpretation of the regional fault configurations. Analysis of Radarsat data showed that major topographic lineaments and regional geological contacts commonly trend north to south and northeast to southwest. The gaps in Cretaceous cover depicted from Radarsat are interpreted to coincide with pre- and/or post-Cretaceous fault zones. Geophysical data also defined a north–south-oriented fabric in proximity to FDN. A more complex picture of lineament configurations was revealed from high-resolution Ikonos images where drainage patterns, in particular, showed systematically corrugated traces that may reflect localized offsets of the regional fault/lineament fabric.

9.3 Geochemical Sampling

Approximately 27,489 surface samples had been taken over the entire Project area to the end of April, 2016. Surface sampling was used as a first-pass exploration tool to identify areas of geochemical anomalies; some of these anomalies remain to be followed up.

Soil (6,252 samples), stream sediment (3,266 samples) and channel, adit, panel, pit, grab and rock sampling (3,015 samples) were collected between 1997 and 2007 by Aurelian and its predecessor companies to evaluate mineralization potential and generate targets for core drilling.

Additional sampling was completed by Kinross from 2011–2013, and Lundin Gold in 2015–2016, and is summarized in Table 9-1.

The soil geochemical surveys are very effective in outlining new areas of interest, while rock samples (boulders and outcrop) help to evaluate the potential of these areas, and define targets for future drilling.

Five key targets, previously identified by their geochemical anomalies are planned to be drilled in 2016 in the Princesa, Emperador 1, and Emperadora concessions, based on additional work done by Lundin Gold. These areas of interest were anomalous to various extents in arsenic, antimony, gold, and/or mercury amongst other elements, all of which were key indicators of blind mineralization at FDN (refer to prospect discussions in Section 7.3). The 2016 drill program is underway.

9.4 Geophysics

Ground geophysical programs completed to date within the Project area include gradient array, pole–dipole array IP resistivity and chargeability surveys. These have been effective in identifying intrusive rocks, faults, basin fill materials, zones of silicification, and pyrite-rich zones at depth.

These methods are particularly effective at the regional level to help define geological and structural context in areas of interest. Because of the thick tropical vegetation and the very limited outcrop exposure in the Project area, IP has been very useful in defining the local geological context in order to help to better understand target areas. In addition, IP surveys are mainly used to identify zones of resistivity which can be related to hydrothermal alteration (silicification), and zones of chargeability which can be related to the presence of sulphides.

Table 9-1: Surface Sampling, 2011–2016

	Duque	Duquesa	Emperador 1	Emperadora	Guacamayo	La Zarza	Marquesa	Princesa	Reina	Sachavaca	Others
2011											
Rock	—	—	9	—	—	638	—	360	—	290	5
Soil	—	—	47	—	—	951	5	1328	1	457	—
Stream Sediment	—	—		—	—	—	—	11	—	—	—
2012											
Rock	68	—	309	189	—	203	69	340	21	16	15
Soil	15	—	824	444	—	1100	1	1228	1	—	—
2013											
Rock	—	—	15	38	14	579	69	—	12	—	232
Soil	—	—	—	—	—	237	—	—	—	—	—
2015											
Rock	8	7	90	69		6	1	35	139	5	2
Soil	301	172	—	—	—	—	—	31	1022	—	—
2016											
Rock	46	122	71	20	25	—	22	135	169	—	38
Soil	—	—	—	—	121	—	300	22	1262	—	—
Totals	***438***	***301***	***1,365***	***760***	***160***	***3,714***	***467***	***3,490***	***2,627***	***581***	***292***

The airborne geophysical program completed to date has included high-sensitivity airborne aeromagnetic and radiometric surveys. Both magnetic and radiometric data are useful at the regional scale to identify areas of interest, major boundaries which can be related to faults, or define geological domains, and large scale targets.

Zones of anomalously low magnetic signature can be associated with hydrothermal alteration when hydrothermal fluids destroy magnetic minerals in the rocks. Zones of anomalously high radiometric values (gamma-rays) may be related to potassic alteration (clays).

A summary of the completed geophysical surveys is provided in Table 9-2.

9.5 Pits and Trenches

Trenching was performed by Climax in 1996–1997 to evaluate areas of artisanal mine workings in the Castillo and Bonza–Las Peñas areas. These trenches were later re-opened by Aurelian. All trenches were geologically mapped and channel sampled. The number of samples taken are included in the overall geochemical sample totals in Section 9.3.

In 2016, new trenching was performed by Lundin Gold on the Emperador and Robles targets. In the case of the Emperador target, the intent was to better expose the sinter discovered in outcrop in 2015 in the principal target area. In the case of the Robles target, trenching was performed to follow up on highly gold-anomalous surface outcrop samples.

9.6 Petrology, Mineralogy, and Research Studies

9.6.1 Kinross and Aurelian Studies

Preliminary microprobe studies were completed to support gold fineness assessments.

Mineralogical studies were commissioned during 2007 to verify minerals associated with veining, in particular to determine the presence of adularia.

Samples of hydrothermal minerals (molybdenite, marcasite and adularia) and igneous units were selected and submitted to the Colourado State University for radiometric isotope dating by rhenium/osmium ratios (Re/Os) and to the University of British Columbia for dating by argon-argon and uranium/lead methods (Ar^{40}/Ar^{39}, U/Pb).

Extensive mineralogical and mineral deportment studies were also completed as part of the 2009 Kinross PFS and the 2011 Kinross FS.

Table 9-2: Completed Geophysical Surveys

Contractor	Survey Type	Date	Company	Comment
Val d´Or Geophysics (VDG del Peru SAC)	IP	1998	Climax	51 line km survey over the Castillo and Bonza targets in the La Zarza concession. A gradient array configuration used a 50 m electrode separation on line separations of 100 m. Instrumentation included a pair of BRGM IP2 receivers coupled with an IPT-1 Phoenix Geophysics transmitter operating in the time domain.
New-Sense Geophysics Limited	High-sensitivity airborne aeromagnetics and radiometrics	2012	Kinross	3,270 line km over the Emperadora, Emperador 1, Princesa, La Zarza, Sachavaca, Colibrí, part of Duque, Duquesa, Reina, Baronesa, Marques, Marquesa, Barón and Colibrí 1 concessions. The survey collected magnetic and radiometric data at a mean flight height of 30 m and a mean line spacing of 100 m using an Astar 350BA helicopter with a fixed mount stinger assembly with a Cesium magnetometer mounted on it (Ellis, 2012). The magnetic data were collected using a KMAG4 magnetometer and the radiometric data were collected using an RS-500 Airborne Spectrometer with an RSX-5 detector pack.
KTTM Geophysics	IP	2015	Lundin Gold	83.7 line km over parts of the Emperadora, Emperador 1, Princesa, Duque and La Zarza concessions. The IP survey was completed in the time domain using a pair of Elrec 6 Pro receivers coupled with a GDD Model II 5,000 W transmitter. Survey configurations for the gradient array and the pole-dipole arrays utilized 100 m electrode separations and a 1,000 m long receiver electrode array. The IP line separations were generally on 100 m or 200 m centres.

Note: The Sachavaca, Colibrí 1 and Duquesa concessions have been relinquished. The Sachavaca 2, Colibrí and Duquesa 2 concessions were amalgamated into the La Zarza concession.

9.6.2 SRK Alteration Study

SRK conducted an alteration study and associated modelling exercise during 2015 to:

- Characterize the extent of the degradation zones within the Suárez Formation conglomerate; postulate causes
- Characterize zones of hydrothermal sericite and clay minerals within and surrounding the gold mineralization; extrapolate for exploration vectoring in other concession areas
- Quantify total clay contents within the gold mineralization using a suite of X-ray mineral liberation (MLA) clay quantification analyses

SRK conducted a degradation survey involving graphic logging of the Suárez Formation conglomerate intercepts and collection of associated infra-red spectra. Data was collected at a spacing of one spectrum per box for the entire length of 83 historical drill holes. Graphic logging data were compiled into a digital database, and a comparison of current and historical core box photographs was completed in PowerPoint format for 58 of the 83 drill holes examined during the degradation survey.

An inspection of drill core indicates that degradation within the Suárez Formation conglomerate is inhomogeneous and does not involve significant volume increase through the production of swelling clays. Degradation is strongest in intervals that are observed or interpreted as containing disseminated pyrite. It is interpreted that the pyrite breakdown upon exposure to air or water leads to the generation of sulphuric acid, which promotes acid attack and further breakdown of pyrite and clay minerals.

The products of core degradation include the residual (i.e. pre-existing) clay minerals paragonite, illite, and minor smectite, and an enhanced concentration of fine-grained silica. The fine-grained silica is interpreted to be amorphous silica that becomes concentrated upon destruction of the smectite. Silicification in the lower parts of the Suárez conglomerate prevents degradation.

The speed of reactions that lead to degradation of the Suárez Formation conglomerate is uncertain. Systematic monitoring and collection of photographs and infrared spectra from the Suárez Formation conglomerate sections of 2015 MET1 holes is necessary in order to determine the speed of degradation.

9.7 Exploration Potential

Exploration along the Las Peñas fault zone remains the first priority in the region, since the discovery of the FDN gold–silver deposit.

Exploration in 2011 continued to focus on the Las Peñas fault zone, more specifically in the La Zarza, Princesa, Sachavaca and Colibrí concessions where epithermal (and possibly mesothermal) systems were targeted.

Since acquiring Aurelian from Kinross in 2014, Lundin Gold exploration work has been mainly focused on concessions outside of La Zarza and includes prospecting, geological mapping, trenching, rock sampling and associated geochemistry, as well as a geophysical survey of key exploration targets (refer to Table 9-1 and Table 9-2).

Among epithermal targets selected for further exploration within the La Zarza concession are the Castillo, Alejandro and FDN East targets, all of which have received some previous work.

Some of the previous geochemical targets outside of La Zarza also show excellent justification for additional work including drilling. These targets include Rio Blanco, Emperador, Robles, Chanchito, and El Arco. All are considered low to intermediate sulphidation epithermal gold–silver targets with evidence of alteration characteristic of the upper levels of hydrothermal systems.

Additionally, although historically not a principal commodity focus for Aurelian, stand-alone, porphyry-hosted deposits, both associated with and proximal to the Las Peñas fault zone, provided secondary tier objectives for future exploration programs. Porphyry-related targets include the Tranca-Loma, Sandia and Papaya targets.

9.8 Comments on Section 9

Geochemical sampling, geological mapping and geophysical surveys have identified a number of anomalies, a portion of which have been drill tested.

Exploration programs conducted are appropriate to the work phase conducted at the time.

The methods used were adequate for the models used of epithermal- and porphyry-style deposits, and the results were instrumental in properly outlining the extent of the mineralization and defining the FDN deposit and other prospects.

There is considerable remaining exploration potential within the Project area.

10.0 DRILLING

10.1 Introduction

Drilling completed within the Project area to 1 December 2015 totals 479 core holes (171,831.03 m), see Table 10-1. Within these programs, the drill campaigns completed in and around the La Zarza concession between 1997 and 1 December 2015 consisted of 438 holes (162,200 m) completed at the FDN deposit, on areas with potential to host infrastructure, and on a number of exploration prospects within the La Zarza concession. A total of 284 holes (126,708 m) were completed over the FDN deposit.

Drill collar location plans are provided in Figure 10-1 for the regional drilling, and in Figure 10-2 for the drilling in the vicinity of the FDN deposit.

No drilling has occurred on the Project between 1 December 2015 and 26 April 2016. A new exploration drill program commenced on 26 April 2016, and is provisionally envisaged as consisting of 20–30 drill holes (7,500–10,500 m), with the number of drill holes completed depending on results. These drill holes will test five to six key low- to medium-sulphidation epithermal gold targets including Rio Blanco, Emperador, Robles, Chanchito, and El Arco. As of 26 April 2016, six holes had been completed on the Emperador and Robles targets, and two were in progress, for a total of about 2,217 m. The drill metres total includes one core hole at Emperador which was abandoned at 36.24 m due to platform stability issues. Drill results were pending as of 24 May, 2016. A drill collar location plan is provided in Figure 10-3 for drill holes that were completed or in progress as at 24 May, 2016.

10.2 Drill Methods

10.2.1 Climax

Four phases of core drilling on the La Zarza concession conducted by Climax were contracted to Connors Perforaciones S.A. A man-portable 20HH drill was used that was capable of drilling up to 150 m of HQ-sized core (63.5 mm core diameter) or up to 300 m depth with NQ-sized core (47.6 mm).

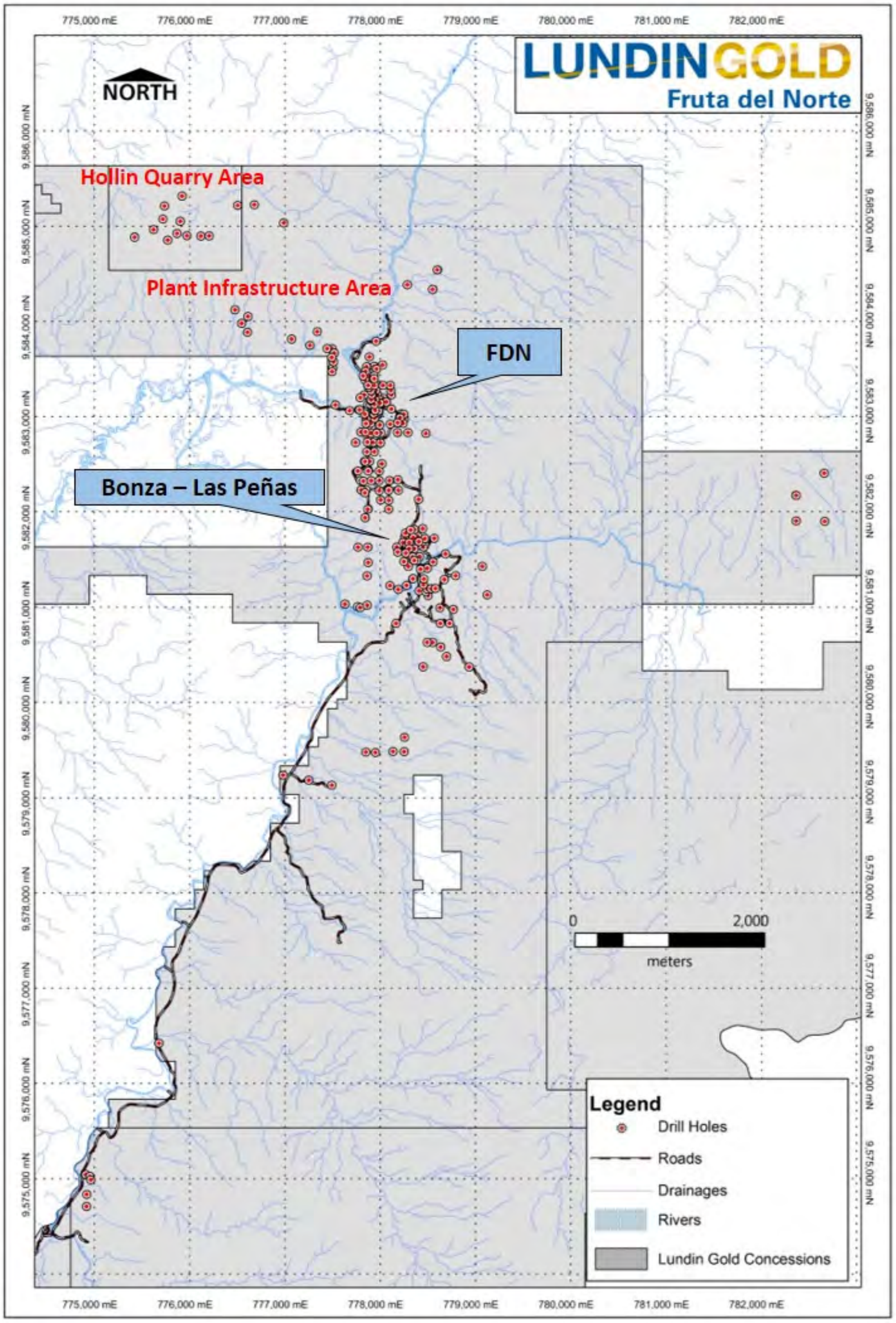
Core holes were collared with HQ-size casings, and usually reduced to NQ before terminating at depths ranging from 50.9–323.7 m. The majority of drill holes were drilled at an azimuth of 090°; however some holes were drilled with 270° azimuths and one drill hole had an azimuth of 075°. Drill hole collar inclinations varied from -45° to -70°.

Table 10-1: Drill Summary Table

Operator	Year	Number of Core Holes Drilled	Metres	Location/Area	Concession
Climax	1997	17	2,566.08	Bonza, Peñas, Castillo	La Zarza
Climax	1998	5	977.8	Bonza, Castillo	La Zarza
Aurelian	2003	14	1,160.85	Aguas Mesas Sur and Norte*	La Zarza
Aurelian	2004	34	7,676.44	Bonza, Peñas, Aguas Mesas Norte	La Zarza
		9	1,266.45	Puente	Princesa
Aurelian	2005	17	3,255.5	Peñas, Tranca Loma, Castillo	La Zarza
Aurelian	2006	48	23,579.13	FDN, Bonza, Las Arenas	La Zarza
Aurelian	2007	101	52,019.78	FDN, Las Arenas, Papaya	La Zarza
		12	3,730.05	El Tigre	Duque
Aurelian	2008	47	23,608.66	FDN, Bonza, La Negra	La Zarza
Kinross	2009	4	2,055.6	FDN	La Zarza
		5	1,739.55	FDN	La Zarza
Kinross	2010	45	18,738.15	FDN	La Zarza
		11	3,636.93	FDN	La Zarza
		4	1,681.05	FDN	La Zarza
		8	505	Plant site - tailings dam	La Zarza
Kinross	2011	4	2,457.25	FDN	La Zarza
		3	354.6	FDN	La Zarza
		16	807.4	Plant site - tailings dam	La Zarza
Kinross	2012	4	2,652.2	Bonza	La Zarza
		7	3,460.3	Naranjilla	Sachavaca
Lundin Gold	2015	11	5,344.07	FDN	La Zarza
		5	511.29	FDN	La Zarza
		7	2,681.64	FDN	La Zarza
		10	1,215.6	FDN	La Zarza
		4	1,673.67	FDN	La Zarza
		5	723.67	FDN	La Zarza
		3	130.5	FDN	La Zarza
		4	248.32	Plant site	La Zarza
		15	1,373.5	Hollín Borrow Pit	La Zarza
Totals		***479***	***171,831.03***		

Note: * The Aguas Mesas Norte and Sur prospects are no longer within the current mineral tenure boundaries as at the Report effective date.

Figure 10-1: Project Drill Hole Collar Plan



Note: Figure courtesy Lundin Gold, 2016. Hollín Borrow Pit (termed a quarry on the figure) area and plant infrastructure areas indicated are the locations for these infrastructure items in the 2016 FS, and are not existing structures/operations.

Figure 10-2: FDN Drill Hole Collar Plan

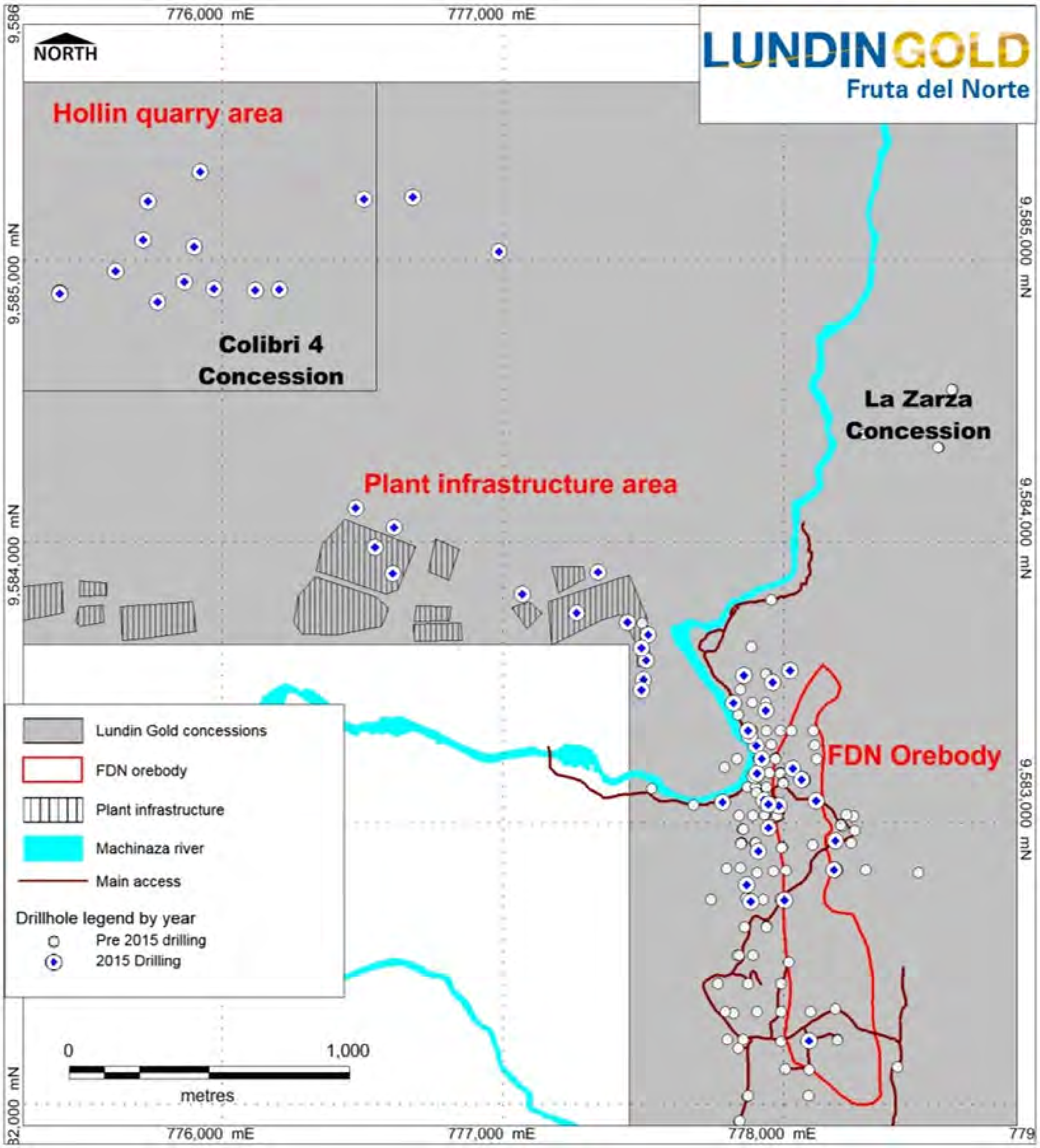
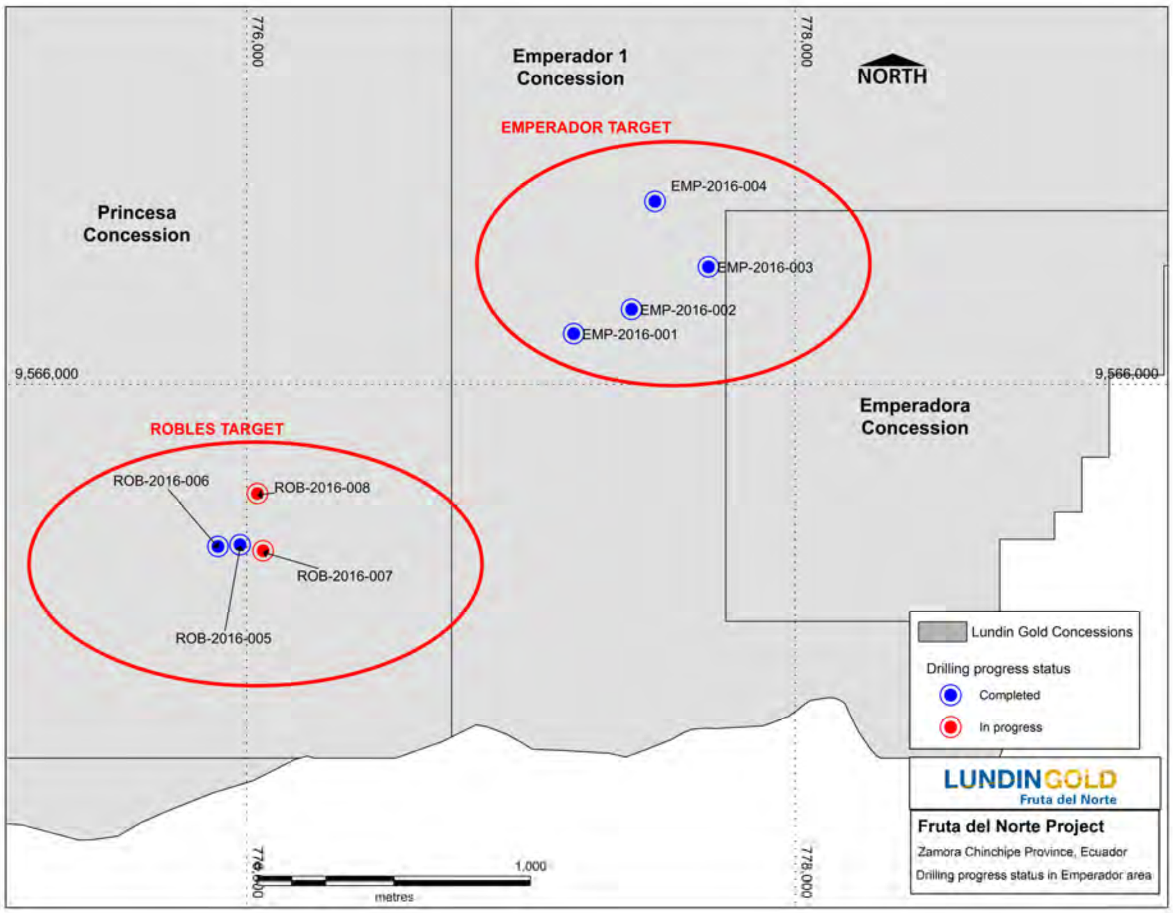


Figure courtesy Lundin Gold, 2016. The outline of the FDN deposit is shown projected to surface. The Hollín Borrow Pit area (termed a quarry on the figure) and plant infrastructure areas indicated are the locations for these infrastructure items in the 2016 FS, and are not existing structures/operations.

Figure 10-3: 2016 Exploration Program



Note: Figure courtesy Lundin Gold, 2016.

10.2.2 Aurelian–Kinross

Drill contractors used on the Project by Aurelian included:

- Paragon del Ecuador S. A. (Cuenca, Ecuador); Hydrocore rig
- Kluane Drilling (Vancouver, Canada); Hydrocore rig
- SFP-Drilling (Lima, Peru); skid-mounted Longyear-70; Christiansen CS-1000
- Major Drilling (Val d'Or, Canada); two Boyles 37 drill rigs; ATV5000 tractor-mounted machine
- Choque Drilling, (Cuzco, Peru); Longyear 38
- Roman Drill (Ecuador); Hydrocore 2000.

Initially, rigs were manually transported along the trails to individual drill platforms following delivery by truck to San Antonio. From 2007, all remote, portable rigs deployed on the Project were lifted/air-supported by ICARO helicopters when needed.

The core diameters varied according to the drilling contractor and drilling program; however, the majority of core ranged from HQ to NQ for exploration purposes, with lesser diameter HQ3 and NQ3 (61.1 and45 mm respectively) for geotechnical purposes using a triple tube system to maximize core recovery and to allow for oriented core, and occasionally HTW (70.9 mm) and NTW (56.0 mm).

Drilling operations at FDN involved rig set up using collar inclinations ranging between -45° and -84°, the majority of which were drilled from west to east (towards azimuth 090°). Most drill holes were collared west of the West Fault. The drill holes were collared with tri-cone or HQ/HTW tools and reduced as necessary to NQ/NTW or even to BTW (42.0 mm) depending on the drilling contractor. This generally occurred at a depth range of between 280 m and 350 m, depending on the ground conditions, drill hole inclinations and operator skill. Many of the drill pads were used consecutively to fan-drill on section either up or down dip along the mineralized system before stepping out to infill on an adjacent section.

Core was placed into plastic-lined wooden boxes to prevent loss of fines. Initially, in the early years, the filled, lid-covered, core boxes were hand-carried by field workers to the covered core logging facility at the Las Peñas camp. Once the road was built between the camp and FDN, core was carried back by workers to the nearest road, then transported to the logging facility in pick-up trucks.

Care was taken to keep all core boxes level and upright during transport.

10.2.3 Lundin Gold

Lundin Gold used two drill contractors in 2015:

- RDMandrill (Cuenca, Ecuador); one Hydrocore 2000 rig and one Hydrocore 4000 rig
- Kluane Drilling Ecuador S.A.; three KD-1000 rigs for the FDN drilling program, and one KD-600 for the civil geotechnical drilling.

Rigs were transported by truck to the road-accessible platforms, and for the remote platforms the drills were brought in via helicopter.

Drilling operations at FDN and at the planned Hollín Borrow Pit area involved rig set up using collar inclinations ranging between -45° to -90°, with azimuths ranging from 0° to 300°. Most drill holes were collared west of the northern half of the FDN deposit. The drill holes were collared in HQ3 or HTW, depending on the contractor, and reduced to NQ3 or NTW at a depth range of 214.9 m to 450 m, depending on ground conditions, drill hole inclination, and operator skills. At the proposed Hollín Borrow Pit area, all holes were drilled in HQ3 diameter to the mostly 100 m target depths.

Core was placed into plastic-lined wooden boxes to prevent loss of fines. Depending on collar location, filled core boxes were covered with lids, then hand-carried to the nearest adjacent road and transported to the core shed via pick-up truck. For remote drill pads, core boxes were piled several boxes high on a wooden pallet, secured, and lifted inside a net using a line from the helicopter, and transported to the Las Peñas heliport; from there the boxes were transported by pick-up truck to the core shed. Care was taken to keep all core boxes level and upright during transport.

For the civil geotechnical drilling all but one of the holes were vertical. The holes were drilled in the proposed North Portal and plant site areas. Standard penetration tests (SPT), vane shear tests (VST) and Shelby sampling was conducted. Holes were collared with a hydraulic hammer for SPT tests, then switched to HQ3 size.

Current 2016 Drill Program

For the 2016 exploration drilling (in progress at the time this report was written) Hubbard Perforaciones Cia. Ltda. (Cuenca, Ecuador) was used as a contractor who utilized three Hydrocore 2500 rigs for the drilling. Rigs were transported by truck to the nearest road, from which the drills were mobilized with helicopter support.

Two drilling teams operated two rigs, while the third rig was mobilized and prepared on the next platform, to reduce standby times during moves.

Drilling operations for the exploration work contemplated rig set-up using collar inclinations ranging between -45° to -90°, with azimuths ranging from 55° to 270°. The drill holes were collared in HTW, with the ability to reduce if required to NTW, depending on ground conditions, drill hole inclination, and operator skills. As of 24 May 2016, the first six holes had been completed using only HTW core to a depth of 401.42 m (depth of the deepest hole).

Core was placed into plastic-lined wooden boxes to prevent loss of fines. Core boxes were piled several boxes high on a wooden pallet, secured, and lifted inside a net using a line from the helicopter, and transported to the nearest road. From there the boxes were transported by pick-up truck back to the Las Peñas camp core shed. Care was taken to keep all core boxes level and upright during transport.

10.3 Geological Logging

10.3.1 Climax

There is no information on the Climax logging procedures. Micon International (Hennessy and Puritch, 2005) noted that geotechnical and geological features were logged. Some of the core was photographed (holes LZD-18 to LZD-22, Phase 4).

10.3.2 Aurelian–Kinross

During the Aurelian drill programs, the following was undertaken.

Initial logging comprised evaluation of geotechnical parameters, comprising core recovery (REC), rock quality designation (RQD), degree of breakage (BRKG), rock hardness (HARD), degree of weathering (WTHR), surface characteristic of joints (SHAPE) and roughness (RGS).

Geological logging was performed using paper logging sheets that were later transcribed to digital files. Logging recorded lithology, alteration, presence of visible gold, mineralization, weathering, veining, textures, and structure, using pre-set codes. Samples for assay were selected by the geologist in charge during the logging process. A summary drill hole trace at 1:1,000 scale was also plotted from GEMCOM giving the geologist the opportunity to summarize the hole and sketch in structural orientations in a form easily transferred to sections.

Kinross programs built on the original Aurelian work. Following arrival at camp, core was photographed, recovery measured and the core was geotechnically logged. This included calculating rock quality designation (RQD), estimating hardness on a 1 to 5 scale and taking structural measurements (i.e. the angle of structures to the core axis).

Lithological logging followed with the geologist recording a detailed description of the lithology, texture, alteration, mineral assemblage and intensity and level of oxidation/weathering. A graphic log column with a sketch of the geology was also included. In addition to conventional logging sheets, the additional information was recorded: percent veining and the percentage of different minerals represented in either vein, breccia or disseminated form (i.e. quartz, carbonates, pyrite, etc.); presence of visible gold; sample intervals; sample numbers and duplicate, blank and analytical standard numbers; magnetic susceptibility measurements; and bulk density measurements.

10.3.3 Lundin Gold

For the 2015 Lundin Gold drill program, given that oriented core was obtained, the first step was to do geotechnical logging at the drill site, which had not been done in any of the previous drill campaigns. Geological logging was then carried out at the core shed in the Las Peñas camp as in the earlier drill programs. For each 1.5 m drill run, the core was marked at the bottom, photographed, the orientation line was drawn along the core axis using a wax crayon, labelled, and logged before placing in the core box. The geotechnical data were recorded on log sheets designed by SRK geologists; the later in-camp logging of lithology, alteration, mineralization and structural was recorded on the same log sheets used in previous drill programs. Data from the log sheets were then entered into a digital database by administrative assistants, under supervision of database personnel. For the geotechnical drill logs (consisting of numbers) a double data-entry into the database was done, by two different people, with the databases then cross-referenced to detect and correct errors.

For the drill program in October and November 2015 that was completed at the planned Hollín Borrow Pit area, double data-entry at the drill site, both paper and digital, was used. Data were either recorded on the SRK geotechnical log sheets, checked by consultant Peter Voulgaris, or on tablet computers using CORE digital logging software designed by independent consultant, Stephane Peloquin. The paper logs were entered into the database and cross-checked with the electronic logs; some errors were noted and the electronic database was corrected where required.

Drill core was delivered to the camp where it was labelled, photographed, logged and sampled under the supervision of FDN staff geologists. Data recorded on log sheets included RQD, hardness estimate, structure, lithology, texture, alteration, mineral assemblage, visual estimate of visible gold abundance and intensity, and level of oxidation/weathering. Log sheets also recorded basic drill hole data including collar coordinates, down hole survey data, core size and depth, drilling dates and sample number series. Occurrences of visible gold were marked on the core using wax crayons.

The 2016 drill program core holes are being logged on tablet computers using CORE digital logging software. The logging protocols are summarized in Figure 10-4.

10.4 Recovery

10.4.1 Climax

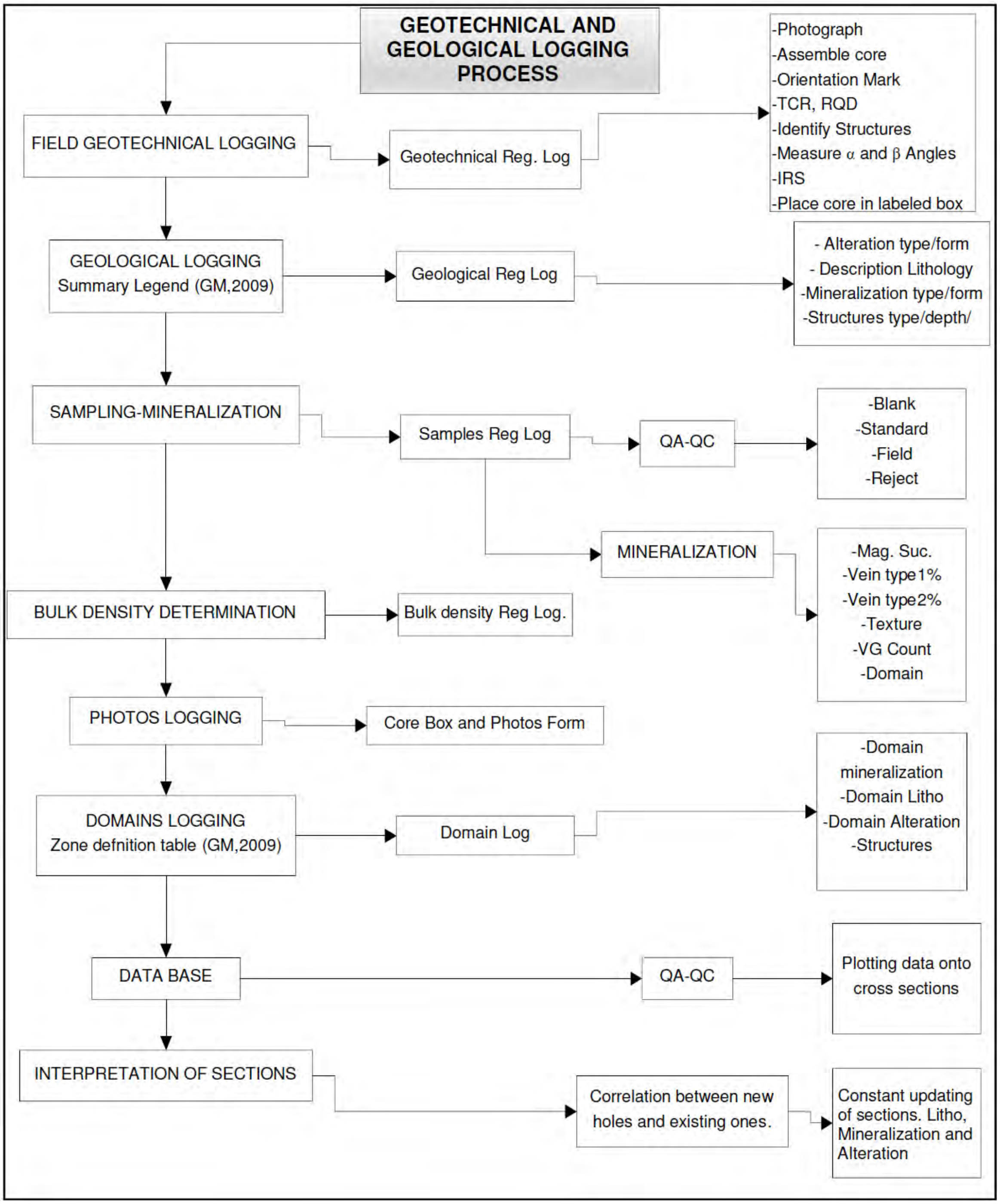
No recovery data are available for the Climax drilling.

10.4.2 Aurelian–Kinross

For the majority of the Aurelian drilling, recovery was typically in the 95% to 100% range and commonly exceeded 98%. Occasionally, recovery appeared to exceed 100% but this is probably due to difficulty in measurement of gouge intervals, rather than down-hole caving, or from core from previous drilling run having been left at the bottom of the hole and recovered in the following drilling run (core recovered exceeding length of core drilled).

As part of the Kinross feasibility study, the relationship between drill core recovery and lithology was measured and tables plotted. Percentages are generally excellent with most lithologies approaching 100% of core recovered. Overburden is the exception, with widely fluctuating percentage recoveries depending on the degree of consolidation and composition of material. Average recoveries are representative of rock type and show acceptable dispersion around the standard deviation.

Figure 10-4: Geotechnical and Geological Logging, Lundin Gold Programs



Note: Figure courtesy Lundin Gold, 2016.

10.4.1 Lundin Gold

For the 2015 and 2016 Lundin Gold drill programs, core recoveries have averaged 91%. Lower recoveries during the 2015 drilling (with respect to previous programs) may in part be due to the number of the 2015 drill holes drilled to the west of the FDN deposit, and others drilled outside the FDN deposit to better define known fault zones where lower core recovery and drill hole problems could be expected.

10.5 Collar Surveys

10.5.1 Climax

Collar locations were not surveyed for the Climax drill holes during the drill programs.

10.5.2 Aurelian–Kinross

During the 2005 to 2007 drill programs, drill hole collars were located by professional Ecuadorian surveyors using a Total Station survey instrument. During the same programs, the existing Climax drill collars were surveyed, where they could be located.

Drill holes completed since the moratorium was lifted were surveyed by Aurelian–Kinross personnel using Total Station survey instruments.

In June 2010, Kinross contracted Leiva to set up a high-precision seven-point FDN geodesic grid, tied to the Ecuador National GPS Grid (RENAGE). It was followed by a differential GPS survey of 73 selected drill holes in the period 2010–2011 (eight from the Aurelian drill programs and 65 from the Kinross drill programs). For these holes, as well as the 2015 survey (see Section 10.5.3), the WGS84 (SIRGAS95) datum was used; and the WGS84 ellipsoidal elevations and the geopotential EGM96 model to obtain the elevations above mean sea level (orthometric elevations). In the 2010–2011 surveys, the double frequency geodesic equipment used consisted of two Trimble 5800 and two Trimble R4 models. Two base stations and two mobile stations were used.

10.5.3 Lundin Gold

For the 2015 and 2016 Lundin Gold drill programs, the orientation of the inclined drill holes was done at the start of each drill hole using a Total Station, with two points surveyed on the drill rods to give the precise 3D drill hole orientation at the collar. For the eight inclined drill holes at the Hollín Borrow Pit area, the drill hole orientation at the collar was measured using a Brunton compass. Subsequent to the completion of each drill hole, the collar was marked by either a) a segment of PVC protruding from the ground where a piezometer had been installed, with a cement box around it; or b) a drill rod where hydrological testing would later be done, with a cement box around it;

or c) a flat base concrete monument with a nail over the drill collar, and the drill hole number imprinted into the cement.

As part of the quality assurance and quality control (QA/QC) process, at the end of the drill program the local engineering firm Leiva was contracted to survey the drill collars using differential double frequency GPS equipment. Leiva used three double frequency Trimble 4 GNSS (Global Navigation Satellite System) and one double frequency Aztech Zxtreme GPS equipment, indistinctly as base and mobile stations. The survey was tied to two base stations on two geodesic grid points previously surveyed by Leiva and tied to the national geodesic grid. The latitude and longitude and the WGS84 UTM coordinates and orthometric elevations for each drill hole collar were thus determined. There was a 30 minute minimum reading time at each collar. The resulting absolute horizontal and vertical precision for this survey is <10 cm.

10.6 Downhole Surveys

10.6.1 Climax

Core holes from the Climax programs were surveyed by either acid tests or Tropari tests.

10.6.2 Aurelian–Kinross

Early Aurelian drill holes were surveyed using acid tests, or a Sperry Sun down-hole camera or a Tropari instrument. In general, holes were surveyed approximately every 50 m down hole and at end of hole.

Down hole surveys during 2006 to 2007 were conducted with either a Sperry Sun or Tropari single shot survey instrument taking a measurement every 50 m, or a Flexit digital multi-shot survey instrument with a reading every 30 m down the drill hole. The instruments were regularly checked in a down-hole survey instrument check station that was set up at the Las Peñas camp to ensure the correct calibration was maintained.

Once downloaded, down-hole deviation data were reviewed before importation into the database. In an audit by Scott Wilson RPA (Evans et al., 2010) the average deviation of azimuth and inclination in a population of multi-shot and single shot data were calculated at 1.7° and 0.9°, respectively, per 100 m.

With the arrival of skid-mounted drill rigs, Flexit and Reflex digital multi-shot survey instruments were used to provide more accurate drill hole survey measurements with a reading on azimuth, dip, rotation angle with respect to gravity and magnetic north, intensity and inclination of the magnetic field, and drill hole temperature. These parameters were measured every 30 m. The digital drill hole survey instrumentation was enclosed in a non-magnetic brass tube that projected 3 m beyond the end of the drill string.

10.6.3 Lundin Gold

For the 2015 Lundin Gold drilling program, a Reflex EZ-TRAC digital down hole survey instrument took readings every 30 m, except in the few cases where drill rods were left in the drill hole, in which case readings were only taken below that level due to magnetic interference. The data were downloaded and reviewed, and if the difference between two contiguous readings was <3° for the azimuth and <1.5° for the dip, they were validated and imported to the drill hole database.

10.7 Twin Holes

When technical difficulties were encountered in the Aurelian–Kinross drill hole CP-07-132, the drill rig was moved 2 m north and the hole was re-drilled as CP-07-137. This resulted in a “twin” intercept of 135 m of mineralization. The two holes had different core sizes for most of the interval, with a reduction from HQ to NQ occurring at 153 m in hole CP-07-132 and at 253 m in hole CP-07-137 (Sims, 2012).

The grade correlation was considered to be reasonably good, given the nature of the mineralized system, until CP-07-137 drilled into a high-grade zone at 245 m with 14 out of 16 samples >10 g/t Au, five samples >50 g/t Au, and one sample assaying 1,135 g/t Au. At the same depth, CP-07-132 also drilled into a high-grade zone, with 11 out of 16 samples >10 g/t Au, but with a maximum of only 34.8 g/t Au.

10.8 Scissor Holes

To help better define the deposit geometry, Aurelian drilled 10 scissor holes. For example, CP-06-63 on section 3400N was designed to drill through the FDN system with an azimuth of 270° and a dip of -63° at the collar. The hole flattened significantly, with a final dip of -52.5° at 590 m.

The geology and grades seen in the drill hole correlate well with mineralization intercepted in the three easterly orientated holes on that section (CP-06-57, CP-06-58 and CP-06-59). Within the upper section of the scissor hole, vein orientations were typically mixed, indicating the zone is a typical stockwork/breccia zone. At depth, however, the scissor hole had a greater number of veins sub-parallel to the core axis.

The current interpretation of the lower part of the system is that it has more sheeted veins that dip towards the west and feed the upper zone that is more brecciated. The evidence indicates that the scissor hole was drilled down dip. It was therefore concluded that in order to optimally intercept veins at a high angle to the core axis (the preferred orientation for sampling), the drilling of 090° oriented holes is preferred over those oriented at 270° (Sims, 2012).

10.9 Geotechnical and Hydrological Drilling

For the purposes of identifying potentially suitable locations for mine infrastructure (soil and rock mass characteristics, and ground water conditions) several geotechnical drilling campaigns were conducted within the proposed FDN mine area and surrounding infrastructure areas. Rock geotechnical and hydrogeological investigation programs for the planned mine area were completed by Golder Associates in 2007 and 2008, Itasca Consulting in 2010, and SRK Consulting in 2015. Soil and hydrogeological investigations for the process plant and tailings dam areas were completed by KCB during 2010, 2011 and 2015. Amec Foster Wheeler completed rock geotechnical investigations for the planned Hollín Borrow Pit in 2015.

Diamond coring techniques were common to most of the programs, using HQ3/HTW diameters in the upper portions of drill holes, reducing to NQ3/NTW diameter at depth. Reflex ACT tools were used for core orientation purposes, and rock core was systematically point load tested at the FDN core facility. Logistics, geological logging and other technical support for each program was provided by FDN staff.

Geotechnical logging was completed by Project staff and/or consultants, with training and supervision provided to the staff where required by each relevant consulting group. In each program, procedures were provided by the consultant, thus, geotechnical data were collected using Bieniawski RMR-76, Barton Q, or Laubscher RMR-90 classifications schemes. Hydrogeological testing was generally completed by the consultant.

In all the programs, rock and soil strength testing was performed off site at one or more of the following facilities: Queen's University, Kingston, Ontario, Canada; the Polytechnic University in Quito, Ecuador (Escuela Politecnica Nacional); and Golder Associates in Vancouver, British Columbia, Canada.

10.10 Metallurgical Drilling

Between January and March 2010, a total of six HQ core holes were completed to obtain sample material for metallurgical testing (drill holes CP-09-241 to 245 and FN3650m01). From June until August 2010, three PQ holes were drilled from west to east to provide intact large diameter core in mineralized domains in a range of grades and elevations (drill holes FN3600p01 to 03). The mineralized intervals were analyzed for gold and other elements using similar methods. The samples had the same system of QA/QC samples inserted and the assays were used in the Kinross resource database.

At the start of 2015, an Early MET program was initiated using existing samples from previous drill campaigns, in addition to fresh samples from the 2015 drill holes. Between March and August 2015, 11 holes using HQ3–HTW and NQ3-NTW size drill core, totalling 5,344.1 m were drilled for metallurgical purposes as part of the Lundin Gold drill program. Most were designed to be used for gravity concentration, flotation, leach and SMC testwork; with the last two, at the end of the program, to make a concentrate for a marketing study. In order to maximize sample quantity for use and to maintain sample integrity for comminution testwork, all HQ3–HTW diameter drill core was split in half and the NQ3–NTW diameter drill core was sent for analysis un-split. Samples from the MET1 program were shipped to SGS Antofagasta, Chile for analysis; and samples from the MET2 and MET4 programs (the MET3 program was cancelled) were shipped to ALS Lima, Peru. For the 2015 Lundin Gold drill program, only the 11 metallurgical drill holes were assayed.

The objectives of the other drill holes were geotechnical, structural, hydrological and mine ventilation evaluations, and the holes were mostly collared outside the FDN deposit.

10.11 Sample Length/True Thickness

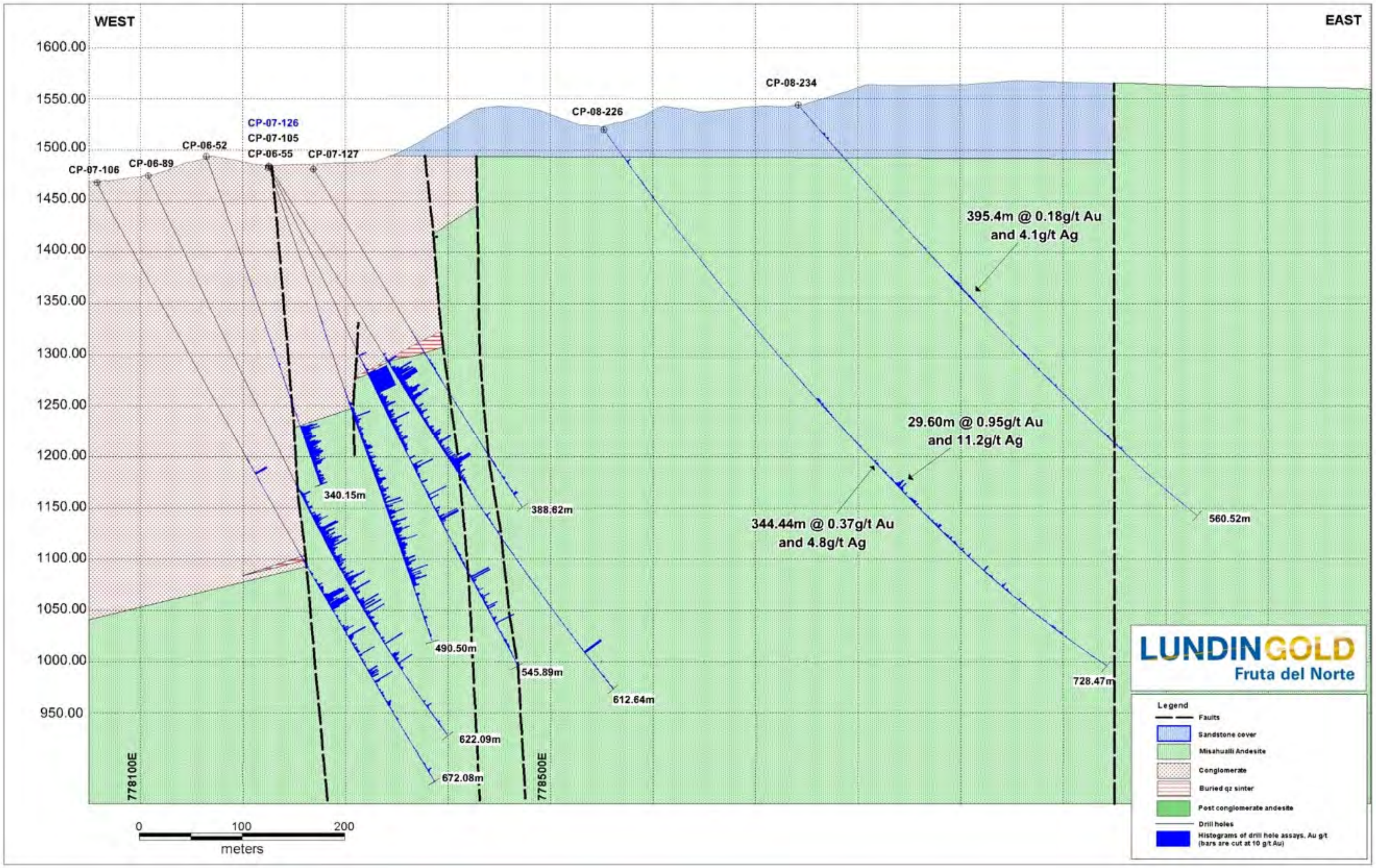
The deposit was systematically drilled out on 50 m to 100 m sections between lines 2500N and 3900N. The grade and mineralization intensity characteristics clearly delineated zones of high grade and high tonnage mineralization in the north versus more disperse, albeit locally high grade mineralization, in the south.

Infill drilling on 50 m centres was focused over 300 m of strike between 3300N and 3600N. The drilling tactic typically involved fan drilling from the pad collar to facilitate between 50 m and 25 m infill drilling before stepping out across strike to define the up or down dip geometry.

Even though the majority of Aurelian core holes were drilled with an easterly (approximately 090°) azimuth and the dominant dip of the mineralized system is west, no single method or percentage adequately describes the complex relationship between down hole (core) length and the true width of the intersected mineralized zones. Drill hole inclinations vary significantly (from -45° to -84°) and the mineralized zones have variable dips from moderate to steep westerly to steep easterly dips. Therefore, most holes intersect the zones at an angle, and the drill hole intercept widths reported for the Project are greater than true widths.

An example of the relationship between true widths, drill intercepts, lithologies and gold grades for drill hole intervals in drill holes is shown on the cross section in Figure 10-5.

Figure 10-5: FDN Cross Section 9,583,200N



Note: Figure courtesy Lundin Gold, 2016.

10.12 Summary of Drill Intercepts

Sample drill intercepts for a selected cross section at 3200N along the strike of the deposit that are illustrative of the nature of the mineralization within the Mineral Resource estimate area are summarized in Table 10-2. The sample drill holes illustrate the grade and thickness variations that can be encountered in the drilling.

Figure 10-5 included labelled examples of a lower-grade interval, and an area of higher-grade within a lower-grade interval.

10.13 Comments on Section 10

In the opinion of the QPs, the quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs during the 1997 and later campaigns are sufficient to support Mineral Resource and Mineral Reserve estimation as follows:

- Core logging meets industry standards for this type of deposit
- Collar surveys have been performed using industry-standard instrumentation
- Down hole surveys were performed using industry-standard instrumentation
- Recovery data from core drill programs are acceptable
- Geotechnical logging of drill core meets industry standards for planned underground operations
- Drill orientations are appropriate for the mineralization style, and have been drilled at orientations that are acceptable for the orientation of mineralization for the bulk of the deposit area
- Most core holes intersect the mineralized zones at an angle, and the drill hole intercept widths reported for the Project are greater than true widths
- Drill orientations are shown in the example cross-section, and can be seen to appropriately test the mineralization.

Table 10-2: Sample Drill Results for Drill Holes on Drill Section 3200

Drill Hole ID	Easting (X)	Northing (Y)	Elevation (Z)	Azimuth (°)	Dip (°)	Total Hole Depth (m)	Depth From (m)	Depth To (m)	Drilled Intercept Thickness (m)	Gold Grade (g/t Au)	Silver Grade (g/t Ag)
CP-06-064	777923.117	9583227.072	1386.929	86.2	-58.3	420.4	125.3	420.4	295.1	2.61	5.3
CP-06-066	777923.02	9583227.027	1386.915	88.1	-69.5	500.0	149.0	500.0	351.0	2.54	9.5
CP-06-092	777832.464	9583228.272	1401.073	87.8	-62.9	418.5	260.0	418.5	158.5	3.32	7.0
CP-07-095	777970.478	9583229.707	1400.525	90.8	-59.4	347.5	94.3	347.5	253.2	5.12	6.5
CP-07-096	777970.675	9583229.714	1400.488	89.8	-45.7	329.2	121.1	329.2	208.1	1.3	4.4
CP-07-120	778117.267	9583228.459	1448.654	270.1	-75.0	427.0	140.7	427.0	286.3	5.66	7.8
CP-07-178	777789.464	9583198.509	1411.631	95.9	-61.9	424.2	350.1	424.2	74.1	4.1	9.6
CP-07-178A	777789.464	9583198.509	1411.631	95.9	-62.1	538.0	383.1	538.0	154.8	5.51	14.3
CP-07-181	777789.296	9583198.603	1411.644	80.9	-62.6	447.4	195.7	447.4	251.8	4.25	7.2
CP-07-181A	777789.296	9583198.603	1411.644	80.9	-62.9	627.0	338.3	627.0	288.7	2.02	9.1
CP-08-209	777922.43	9583228.597	1387.046	89.1	-44.5	280.2	142.6	280.2	137.6	2.91	5.4
CP-08-214	777922.284	9583228.665	1386.983	88.2	-53.0	416.9	133.8	416.9	283.1	2.05	4.3
CP-08-218	777922.167	9583228.631	1387.013	87.2	-65.6	483.9	143.5	483.9	340.4	3.64	7.1
CP-09-244	777970.752	9583228.887	1400.591	86.0	-59.3	236.8	117.9	226.6	108.7	5.51	10.3
FN3600d01	777911.648	9583228.835	1384.49	76.9	-52.4	270.0	143.4	270.0	126.7	4.51	6.4
FN3600d02	777911.235	9583229.249	1384.504	70.6	-65.0	390.0	156.6	390.0	233.5	3.79	5.7

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

11.1.1 Climax

Sampling methods for the Climax programs are summarized in Table 11-1.

11.1.2 Aurelian–Kinross

Sampling methods for the Aurelian–Kinross programs are also included in Table 11-1.

During Aurelian drilling, sacks containing samples were generally stored briefly (less than 1 week) at the field camp or at the Las Peñas camp before transport by canoe or men to vehicles at San Antonio. During the Kinross programs, once sealed, core boxes were transported to Las Peñas exploration camp from the drill platform.

For both programs, at the camp, core was checked by geologists and stored in the core shed during the logging and sampling process. As soon as samples were taken, they were sealed in plastic bags and rice sacks and stored in a locked shed overnight. The Las Peñas camp retained 24-hour security, which monitored any activity in the core shed area. Samples were then transported overland by a company driver directly to Quito where the custody of the samples was transferred to laboratory personnel. Signatures for responsible parties were required at every step of the process and records were archived at the Las Peñas camp.

11.1.3 Lundin Gold

After the geologist had marked out the sample intervals, drill core was split along the long axis using an electrically-powered bench saw. Areas of very soft rock were cut using a machete and sections of very broken core were sampled using spoons. As with the Kinross programs, the right hand side of the core was always sampled. Sample interval selection criteria remained the same as for the Aurelian–Kinross programs.

11.2 Metallurgical Sampling

Selection of samples for the metallurgical testwork programs that support the 2016 FS is discussed in Section 13.

Table 11-1: Sampling Methods, Climax and Aurelian–Kinross

Operator	Sampling Method
Climax	Core was sawn in half and sampled at 2 m intervals, regardless of geology. Each sample consisted of 2 m composites of half core, with the exception of the first and last intervals in each hole.
Aurelian	All strongly altered or epithermal-mineralized intervals of core were sampled, with the exception of some intervals within the Suárez Formation once it was established that this material did not contain potentially economic levels of gold. Sampling always began at least five samples above the start of mineralization typically encompassing the basal 10–20 m of Suárez Formation. Sample intervals were selected using the following criteria. - Maximum sample length of 2 m in un-mineralized lithologies; - Maximum sample length of 1 m in mineralized lithologies; - Smaller samples may be selected around high grade, visible gold-bearing veins; - Minimum sample length of 20 cm; - Geological changes in the core such as major mineralization/alteration intensity and lithology changes were used as sample breaks; - Core size changes and any zones of core loss were used as sample breaks; - Large discrete veins that might possibly be modelled or mined as separate structures were sampled separately; - The begin/end marks were placed so that the entire vein ended up in the sample(s) and the vein is not smeared into samples on either side. - One half of the core has been retained in the field for future examination and verification.
Kinross	Similar methodology to Aurelian. Areas of very soft rock (e.g. fault gouge) were cut with a machete and very broken core (pieces less than 1 cm) were sampled using spoons. The right hand side of the core was always sampled. After cutting, half the core was placed in a new plastic sample bag and half was placed back in the core box.

11.3 Density Determinations

After the core had been sampled, intervals of solid core (20 cm to 10 cm in length) were selected for bulk density determinations. Measurements were made from every hole at an interval rate of approximately 50 m in unmineralized rock and every 20 m in the mineralized system. Bulk density was estimated according to Archimedes' principle, where the sample is dried, weighed, waxed and then weighed in water.

Rock density is relatively constant within specific lithologies and shows only minimal variation between different lithological groups with the relatively small difference of 0.5 t/m^3 between the lowest density of 2.4 g/cm^3 and highest density of 2.9 t/m^3.

11.4 Analytical and Test Laboratories

Climax used Bondar Clegg Laboratories in Canada and Bolivia as the assay laboratories for its drill programs. Bondar Clegg has subsequently been purchased by ALS Chemex Laboratories (ALS Chemex). At the time of analysis, Bondar Clegg was independent of Climax. Accreditation status of the laboratory is not known for that time.

Preparation and analysis of FDN samples for the period 2006–2015 was completed at laboratories as detailed in Table 11-2.

11.5 Sample Preparation and Analysis

Laboratory information on sample preparation procedures is provided in Table 11-3, and laboratory information on sample analytical procedures is provided in Table 11-4. Laboratories in the tables were/are independent of Aurelian, Kinross, and Lundin Gold.

The following abbreviations are used for analytical methods:

- ICP: Inductively-coupled plasma
- ICP-AES: Inductively-coupled plasma - atomic emission spectroscopy
- AAS: Atomic absorption spectroscopy.

11.6 Quality Assurance and Quality Control

Aurelian implemented a thorough QA/QC program that included the regular insertion of blank samples, certified reference materials (CRMs), field and reject duplicates and pulp check assaying. The program was continued and modified by Kinross and more recently by Lundin Gold. Ongoing monitoring of the program was performed by the operators, with spurious results being investigated and changes implemented when required. Insertion rates and procedures employed by Aurelian, Kinross, and Lundin Gold are shown in Table 11-5.

11.6.1 Certified Reference Materials

Certified reference materials were sourced from Rocklabs in New Zealand. CRM material was included in the sample stream at a rate of one-in-20 from 2006 to 2008, and reduced when Kinross became the operator in 2009 to a rate of one-in-25. In any single year, a minimum of five, and a maximum of 12 separate CRMs were included in the sample stream. The actual insertion rate of gold CRMs ranged annually from 2% to 7% and averages 5% for the Project. The insertion rate of silver CRMs is significantly lower than for gold, averaging 2% for the Project. Almost no silver CRMs were included in the sample stream for the Project after 2009.

For FDN, the gold grades of interest are approximately 3 g/t (cut-off grade), 9 g/t (average grade) and over 20 g/t (high grade). Silver grades of interest, although supplemental to gold, are from 10–20 g/t Ag. The ranges of expected values of the submitted CRMs for gold is from 0.819–30.14 g/t Au and for silver is from 11.02–58.38 g/t. Although still of some value, a total of 10 of the 23 CRMs used were below the grade range of interest for the Project, representing almost 50% of total submittals.

Table 11-2: Laboratories Used, 2006–2015

	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015
Preparation Facility										
ALS Quito										
Inspectorate Quito										
SGS Santiago										
Analytical Facility										
ALS Vancouver										
ALS Lima										
Inspectorate Lima										
SGS Toronto										
SGS Antofagasta										
Samples in drill hole database	9,759	22,418	10,451	1,705	10,296	654	682			2,388

Notes: ALS: ALS Global, formerly ALS Chemex; Inspectorate: Bureau Veritas Commodities, formerly Inspectorate Services Inc.; diagonal stripes indicate the laboratory has been used as a check assay laboratory.

Table 11-3: Sample Preparation Laboratory Summary

Laboratory	Accreditation	Comment	Sample Preparation Methodology
ALS Quito	ISO 9001:2008 for quality management systems	Principal preparation laboratory for drill holes CP-06-49 to CP-06-53 and CP-06-57 to FN3750d01 and MET2-2720, MET2-2780, MET2-3400, MET4-2920, and MET4-3070 Check assay laboratory for selected samples from drill holes CP-06-53 to CP-06-56 (coarse rejects re-pulverized)	- Oven dry the sample on steel trays - Crush entire sample to better than 70% passing -2 mm (10 mesh) - From mid-2006, the crusher was cleaned with quartz flush and air gun between each sample - Riffle split 250 g (1,000 g); 2015 MET2 and MET4 holes were riffle split to obtain 300 g or 1,000 g sample splits, respectively * - Pulverize split to better than 85% (90%) passing -75 microns or 200 mesh (-100 microns, 150 mesh); 2015 MET holes were pulverized to better than 85% passing -75 µm (200 mesh) * - Clean pulverizers with an air gun between samples - 110 g, 150 g or 200 g pulps sent in Kraft bags to Vancouver (Lima) for analysis
Inspectorate Quito	ISO 9001:2008 for quality management systems	Principal preparation laboratory for drill holes: CP-06-53 to CP-06-56	- Oven dry the sample on steel trays - Crush entire sample to better than 90% passing -2 mm (10 mesh) - Riffle split 1,000 g - Pulverize 1,000 g split to better than 90% passing -100 µm (150 mesh) - Clean sand flushes between each pulverization - 100 g pulps sent (via TNT courier) in Kraft bags to Peru for analysis.

Laboratory	Accreditation	Comment	Sample Preparation Methodology
SGS Santiago	ISO 9001:2008 for quality management systems	Principal preparation laboratory for metallurgical drill holes: MET1-2900, MET1-3070, MET1-3170, MET1-3257, MET1-3310	- Oven dry samples on steel trays - Crush entire sample to 100% passing 3.35 mm (6 mesh) - Split of 5% of the sample using rotary splitter of 20 divisions - Pulverise the split to 100% passing 106 µm (150 mesh) - 180 g pulps sent by surface transport (via Chilexpress) in Kraft bags to SGS Antofagasta for analysis - All remaining coarse reject and pulps are stored in SGS installations - Compressed air guns used to clean the crushers and pulverisers between each sample in drill holes MET1-3170, MET1-3257 and MET1-3310

Note: * Bracketed values indicate change to procedure implemented on 01 January 2007 (from hole CP-07-95) following observations of visible gold in the drill core. This change corresponds to a change in the primary analytical laboratory from ALS Vancouver to ALS Lima.

Table 11-4: Analytical Laboratory Summary

Laboratory	Accreditation	Comment	Sample Preparation Methodology
ALS Lima and ALS Vancouver	ISO 9001:2008 for quality management systems; ISO/IEC 17025:2005 for competence of laboratory testing	Principal analytical laboratory for drill holes: CP-06-49 to CP-06-53, CP-06-57 to CP-06-094, CP-07-095 to FN3750d01, MET2-2720, MET2-2780, MET2-3400, MET4-2920, and MET4-3070. Check assay laboratory for selected samples from drill holes: CP-06-53 to CP-06-56.	- Gold was determined by 30 g (50 g) 1 fire assay with an ICP-AES2 finish, method code AU-ICP21 (AU-ICP22) [1]. Detection range for this procedure is 0.001 g/t Au to 10 g/t Au. - The principal Au determination method was changed to method code Au-AA24 from drill hole CP-07-98 to BLP2130e01 (end of 2012), which applies an AAS [3] finish following a 50 g fire assay. Detection range for this procedure is also 0.005 g/t Au to 10 g/t Au. Pulps from drill holes CP-06-57 to CP-06-64 originally assayed using method AU-ICP22 were re-assayed using method Au-AA24 for comparison. - If Au assays greater than 10 g/t were detected using either of the above techniques then over-limit re-assays were completed using a 50 g fire assay with a gravimetric finish, method code AU-GRA22. The detection range for this procedure is 0.05 g/t Au to 1,000 g/t Au. This technique was also applied as the initial gold assay (rather than over-limit) to the 2015 drill holes listed above (with prefix MET). - Multi-element analysis was performed on samples from 2006 to 2012 using method code ME-ICP41, a 34 element package, including silver, with a nitric aqua regia acid digestion, and ICP-AES2 finish. The silver detection range for this procedure is 0.2 ppm to 100 ppm.

Laboratory	Accreditation	Comment	Sample Preparation Methodology
			- Multi-element analysis was performed using method code ME-ICP61, a 33 element package, including silver, with four acid digestion and ICP-AES [2] finish. The silver detection range for this procedure is 0.5 ppm to 100 ppm. This technique was applied in 2015 to silver assays from drill holes listed above with prefix MET. - Over-limit re-assays were run on selected drill holes [4] for silver, zinc, lead and copper if Ag >100 ppm, Zn >10,000 ppm, Pb >10,000 ppm or Cu >10,000 ppm. Over-limit re-assays were completed using an aqua regia acid digestion and AAS [3] finish (method code AA46). The silver detection range for this procedure is 1 ppm to 1,500 ppm
Inspectorate Lima	ISO 9001:2008 for quality management systems; ISO/IEC 17025:2005 for competence of laboratory testing	Analytical laboratory for drill holes: CP-06-53 to CP-06-56. Check assay laboratory for selected samples from drill holes: CP-06-51 to CP-06-52, CP-06-57 to CP-06-64 (Au only), CP-06-65 to CP-08-236, and 2015 metallurgical holes MET2-2720, MET2-2780, MET2-3400, MET4-2920, and MET4-3070 (Au only).	- Au was determined by 50 g fire assay with an AAS finish, method code Au-FA/AAS, which has a detection range from 0.005 g/t Au to 5 g/t Au. - If Au assays greater than 5 g/t were detected using the above technique then over-limit re-assays were completed using a 50 g fire assay with a gravimetric finish. The detection range for this procedure is 0.01 g/t Au to 1,000 g/t Au. - Multi-element analysis was completed using a 32 element package (including silver) with an aqua regia acid digestion and ICP-AES finish (method ICP-AES 32). The detection limits for this procedure range from 0.2 ppm to 200 ppm Ag.
SGS Toronto	ISO 9001:2008 for quality management systems; ISO/IEC 17025:2005 for competence of laboratory testing	Check assay laboratory for selected samples from drill holes: CP-06-51 to CP-06-52, CP-06-57 to CP-06-64 (gold only) and CP-06-65 to CP-08-236	- Au was determined by a 50 g fire assay with an AAS finish, using method code FAI505. The Au detection range for this method is 0.01 g/t to 10 g/t. - If Au assays greater than 10 g/t were detected using the above technique then over-limit re-assays were completed using a 30 g or 50 g fire assay with a gravimetric finish (FAG333 or FAG505, respectively). The detection range for this procedure is 0.3 g/t Au, or 0.5 g/t Au, to 3,000 g/t Au. - Ag was assayed using method code AAS12E, which involved a two-acid digestion of a 2 g sample and AAS finish. The detection limits for this procedure range from 0.3 ppm to 300 ppm Ag
SGS Antofagasta	ISO 9001:2008 for quality management systems; ISO/IEC 17025:2005 for competence of laboratory testing	Principal analytical laboratory for metallurgical drill holes: MET1-2900, MET1-3070, MET1-3170, MET1-3257 and MET1-3310	- Au was determined by a 50 g fire assay with a gravimetric finish, using method code FAG505. The Au detection range for this method is 0.05 g/t to 3,000 g/t. - Ag was assayed using method code ICP040B, which involved a four-acid digestion followed by

Laboratory	Accreditation	Comment	Sample Preparation Methodology
			ICP-AES finish on a multi-element analysis (35 elements). The silver detection limits for this procedure are 0.5 g/t to 100 g/t. - Ag was also assayed using method code AAS042D, which involved four acid digestion and an AAS finish. The silver detection limits for this procedure are 1 g/t to 500 g/t Ag.

Note: **1**. Bracketed values indicate change to procedure implemented at ALS Vancouver from hole CP-06-57, following a return to ALS Vancouver as the primary laboratory after a brief respite when three holes were assayed at Inspectorate Services in Lima, Peru. This procedure was continued by ALS Lima from drill hole CP-07-095 when it took over from ALS Vancouver as the primary laboratory on 01 January 2007. The gold assay procedure changed to use AAS finish in place of ICP-AES from drill hole CP-07-98. **2** ICP-AES: Inductively-coupled plasma - atomic emission spectroscopy; **3** AAS: Atomic absorption spectroscopy; **4** Over-limit re-assays for silver, zinc, lead and copper were completed for drill holes CP-09-237 CP-09-238, CP-09-239 and CP-09-240 only. All other silver assays that returned values above the detection limit were entered in the database as equal to the upper detection limit (100 ppm).

Table 11-5: Summary of QA/QC Sample Submission Rates

From	To	QAQC Type	Insertion Rate/Procedure
Aurelian			
2006	2008	CRM	1 of 12 individual Rocklabs CRM were inserted every 20th sample, selected at a grade roughly equivalent to surrounding samples
2006	2006	Blanks - sand	1 in 20 or 1 in 25, and after visible gold
2006	2008	Blanks - coarse rock	1 in 20, and after visible gold
2006	2008	Field duplicate	1 in 50
2006	2008	Coarse reject duplicate	1 in 50
2006	2008	Check assay	Four separate batches sent to at least one umpire laboratory.
Kinross			
2009	2012	CRM	1 of 14 individual Rocklabs CRMs inserted every 25th sample
2009	2012	Blanks - coarse rock	1 in 25
2009	2012	Field duplicate	1 in 50 (2, 1/4 core samples)
2009	2012	Coarse reject duplicate	1 in 50
Lundin Gold			
2015	2015	CRM	1 of 10 individual Rocklabs CRMs inserted every 25th sample
2015	2015	Blanks - coarse rock	1 in 20
2015	2015	Field duplicate	1 in 50 (2, 1/4 core samples) samples submitted to ALS Lima (MET2 and MET 4)
2015	2015	Coarse reject duplicate	1 in 50 samples submitted to ALS Lima (MET2 and MET 4)
2015	2015	Check assay	1 in 10 samples submitted to ALS Lima (MET2 and MET 4) were also assayed at Inspectorate Lima

Despite this high number of low-grade CRMs, good temporal coverage of the gold grade range of interest exists for the Project.

Failure rates, defined as a gold or silver value reporting more than three standard deviations from the expected value, or two consecutive gold or silver values reporting more than two standard deviations from the expected values were tabulated. Overall, ALS Lima, ALS Vancouver and Inspectorate Lima demonstrated good performance with regard to both silver and gold. The performance of SGS Antofagasta was notably poor, with an overall failure rate of 29%.

Some of the CRMs showed poor reproducibility in expected grade values, and were submitted over several years during which other CRMs consistently performed well. These findings indicate these particular CRMs may not be well homogenized, and the performance should be reviewed in this context.

11.6.2 Blank Material

During the first part of 2006 blank material was sourced from Hollín Formation sands located near the Emperador concession. Following poor performance, the source of blank material was changed to Hollín Formation quartz sandstone, sourced from an outcrop north of FDN.

RPA reviewed the results of the gold blank samples and tabulated the number of failures each year:

- A high number of failures in 2006 was reported from blank material prepared in ALS Quito and assayed at ALS Vancouver. Performance of blank material prepared and assayed in Inspectorate's facilities in Quito and in Lima, respectively (four holes, 9% of 2006 blank submittals) was excellent, with no reported failures
- Performance of blank samples improved and are considered to be acceptable from 2007 to 2015, with an overall failure rate of 1.6%. Anomalous results are interpreted as contamination or a sample switch. Site operators have consistently monitored the results of blank samples and followed up spurious results with investigations throughout the Project life. Blanks were analyzed using procedures with very high lower detection limits in 2015 (0.05 g/t Au), and thus the failure criteria were relatively high compared to earlier campaigns.

11.6.3 Duplicates

A total of 1,632 field duplicate and 1,739 coarse reject duplicate samples were collected from 2006–2015:

- Field duplicate samples were collected as two quarter core samples, with the remaining half core sample retained for reference

- Coarse reject samples were collected as an additional split from the crushed reject material (better than 70% passing -2 mm or 10 mesh)
- Pulp reject duplicate samples were submitted as part of the 2015 QA/QC program at FDN only (152 sample pairs; gold only).

The results of the field, coarse reject and pulp duplicate samples prepared and assayed at the ALS facilities were reviewed through the preparation of summary statistics, as well as scatter, quantile-quantile, and precision plots. Gold duplicates tended to compare better than the silver duplicate results; however, both gold and silver showed good correlation.

Field duplicate samples for both gold and silver show a fairly high level of scatter, which is to be expected in a project with visible gold. No bias was observed, indicating that the samples with more visible gold have not been preferentially sampled.

Coarse reject duplicate pairs reported a correlation coefficient greater than 0.995 for both gold and silver, and each sample set reported similar statistical parameters, indicating that the procedure for crushing the sample and splitting to obtain a sample for grinding is sufficiently homogenizing the sample and the split is representative of the crushed sample. Pulp duplicate pairs had a correlation coefficient of 0.999 and graphically showed excellent precision.

11.6.4 Check Assays

Pulp reject samples were submitted to Inspectorate in Lima and SGS in Toronto from 2006 to 2008, and to Inspectorate in Lima during 2015. Check assays prior to 2015 were not supported through the inclusion of blank and CRM samples with sample submissions. As of 2015, CRM samples were included in the check assay sample batches.

The results of the secondary and tertiary laboratory testing were analyzed using basic statistics, scatter, quantile-quantile (QQ), and percent relative difference plots, separately for each primary laboratory, and considering the method type employed, for both gold and silver.

The results of the check assay review demonstrate overall good correlation of the ALS Vancouver laboratory with results from both Inspectorate Lima and SGS Toronto. A slight high bias is observed between the primary laboratory and SGS Toronto at grades above approximately 5 g/t Au and Inspectorate Lima above approximately 18 g/t Au. The Inspectorate Lima data set is less scattered than SGS Toronto.

The original ALS Lima gold results were compared with the results from the secondary and tertiary laboratories, considering the analytical method employed at the primary laboratory. The results indicate an improvement in correlation with the adoption of method code AU-AA24 (fire assay with AAS finish) from method code ICP22 (fire assay with ICP-AES finish) by ALS Lima; however, both methods compare well, particularly below 10 g/t Au. The slight positive bias observed in the ALS Vancouver laboratory remains present in the ALS Lima laboratory, where assays were finished using ICP-AES. Following the ALS Lima method code switch to AU-AA24, the bias is no longer present.

Comparative statistics of the silver assay results demonstrated mixed results, depending on the assay method employed. During 2006, a small number of pulp reject samples were submitted to Inspectorate Lima for four-acid digestion and to SGS Toronto using method code FA-ICP-OES, in addition to the standard method codes. The SGS Toronto FA-ICP-OES results are particularly poor; however, the laboratory utilizes a separate analytical technique that differs from the standard technique. Good correlation exists between ALS Vancouver with both Inspectorate Lima and SGS Toronto, although ALS Vancouver results assay slightly higher than Inspectorate Lima. This bias was reduced to a negligible amount following the 2007 switch to ALS Lima as the primary assaying facility.

11.7 Databases

Most data were originally recorded as hard copy. Technicians later entered the following into the database: sample number, sequence, interval, QA/QC data and other geological information such as collar information, depth of drill size reduction, date, and drill company details. Once the data had been entered, they were validated against the original hard copy. After validating input data, geological assistants were required to sign a statement confirming the data had been checked and were correct. Basic database checks were also carried out by the database administrator.

11.8 Sample Security

During the Lundin Gold programs, drill core was delivered to the camp where it was labelled, photographed, logged and sampled under the supervision of staff geologists.

After the geologist marked out the sample intervals, drill core was split. The following standard sampling procedures were employed:

- After cutting, half the core was placed in a new plastic sample bag and half was returned to the core box
- Samples were clearly and securely bagged and tagged and QC samples inserted into the sequence
- Batches of approximately 10 samples were bagged into labelled poly-weave sacks for shipment.

Once ready for shipment, a list of sample batches and included samples was sent via electronic mail to camp administration and logistics, to the sample preparation laboratory, and to camp security, before the sample batches left camp. The Las Peñas camp has 24-hour security, which includes monitoring of the core shed area. Drilling samples were then transported from camp overland by a transport company truck directly to Quito where the custody of the samples was transferred to laboratory personnel. During transport camp security maintained communication with the transport company driver in order to track the progress and safety of the transport truck.

No Aurelian, Kinross or Lundin Gold personnel conducted any sample preparation. Preparation and analysis of FDN samples were completed at independent laboratories as detailed earlier in Table 11-3.

11.9 Sample Storage

Mineralized half and quarter core retained after analysis and sampling for all holes is presently stored in permanent core storage facilities at the Project site.

11.10 Comments on Section 11

Sample collection, sample preparation, analytical methods and security for all Aurelian–Kinross and Lundin Gold drill programs are in line with industry-standard methods for epithermal gold–silver deposits and can support Mineral Resource and Mineral Reserve estimates.

The QPs note:

- The number, grade range and temporal range of submission of gold CRM samples at the Project are sufficient to monitor accuracy of the laboratory. Insufficient silver CRM samples were included at the Project from 2009 to present
- With few exceptions, the gold CRMs assayed at ALS Vancouver, Inspectorate Lima and ALS Lima demonstrate a high level of laboratory accuracy and a good level of precision, with fewer than 3% of submissions failing to meet expected criteria
- CRM and blank samples assayed at SGS Antofagasta demonstrated poor performance and were excluded from the Mineral Resource database. These data are from the 2015 drilling and were not used for the current Mineral Resource estimate
- Several CRMs assayed at ALS Vancouver or ALS Lima with expected values ranging from approximately 1 g/t Au to 6 g/t Au showed a slight positive bias ranging from approximately 0.01 g/t Au to 0.1 g/t Au. This bias was not observed in CRMs ranging in expected grade from approximately 6 g/t Au to 30 g/t Au

- Results from all silver CRMs assayed at ALS Vancouver and ALS Lima show a slight high bias. The silver CRM failure rate was sufficiently low to demonstrate laboratory precision and accuracy
- The positive bias observed in the silver CRM samples is also present in the check assay data for silver, indicating the silver results from both ALS Vancouver and ALS Lima are likely slightly biased high
- The positive bias at a gold grade range from 1 g/t Au to 6 g/t Au observed in CRM data is not seen in the check assay data for gold assayed at ALS Lima using method code Au-AA24. It is present in results obtained at ALS Vancouver and ALS Lima using method codes Au-ICP21 and Au-ICP22. These results indicate that gold results within this grade range obtained during 2006 and the first part of 2007 may be slightly biased high. This slight bias is unlikely to meaningfully impact the program results
- Samples collected during 2006 likely were subject to a moderate degree of contamination during preparation at the ALS preparation laboratory in Quito, Ecuador, as evidenced by the results of the blank sampling program. Remediation actions taken by the Project operator and preparation laboratory successfully reduced the instances of sample contamination on the Project from 2007 to 2012
- The results of the field duplicate sampling program have confirmed the presence of natural grade variability within the samples at FDN. The number of field duplicates taken to date is sufficient to understand the natural variability and there is no further need for field duplicate samples for FDN
- The results of the coarse reject duplicate sampling program have confirmed that the coarse reject material is homogenous and representative of the sample. The monitoring of coarse reject material should continue at the Project at a reduced insertion rate.

12.0 DATA VERIFICATION

12.1 Scott Wilson Roscoe Postle Associates Inc. (2009)

A significant portion of the database verification was performed by Scott Wilson Roscoe Postle Associates Inc. (Scott Wilson RPA), a predecessor company to RPA, during an audit of the December 31, 2009 Mineral Resource and Mineral Reserve estimates. Scott Wilson RPA's audit was completed by Luke Evans, M.Sc., P.Eng., Dennis Bergen, P.Eng., Associate Principal Mining Engineer, and Holger Krutzelmann, P.Eng., Principal Metallurgist. Mr. Evans and Mr. Bergen visited the FDN site from 6 to 9 April 2010 (Evans et al., 2010).

Data verification activities carried out by Mr. Evans included a detailed review of the standard operating protocols, the drill hole spacing, the core diameter used, how the final collar coordinates were determined, the down hole surveying procedures, the drill core logging protocols, the core recovery, collection of the bulk density data, the sample layout, sample preparation and sample security procedures, and the QA/QC protocols.

12.2 RPA (2014)

In June 2014, Kinross provided to RPA a Dassault Systèmes GEOVIA GEMS (GEMS) Project containing updated drill hole database, core recovery and density measurement files in digital format. To ensure the benefit of the previous data verification work related to the 2010 audit, RPA compared the updated database provided in June 2014 with the database used for the December 31, 2009 Mineral Resource and Mineral Reserve estimate. No significant discrepancies were identified.

12.3 RPA (2016)

Mr. David Ross visited the FDN site from 7 to 9 April, 2016.

12.4 RPA Site Visits and Core Review

During the site visits, RPA reviewed drill core from numerous core holes and compared observations with assay results and descriptive log records made by Aurelian and Kinross geologists. In addition to reviewing core, RPA examined outcrops, drill rigs, sampling procedures and other general exploration protocols.

12.5 Checks on Assay Data by Other Consultants (2007)

Hennessey and Stewart (2007) used a two-phase verification process to check 100% of the assay data compiled up to and including 2007 drilling. The first phase was to check all database assays on an ongoing basis as the certificates arrived from the laboratory. The second phase was to re-check 10% of the database against the laboratory certificates.

12.6 Checks on Assay Data by Kinross–Aurelian (2009–2010)

Between late 2007 and early 2009 no drilling activities were undertaken. At the end of the 2009 and 2010 infill programs, site personnel compiled and checked all certificates against the database for all elements. The comparison showed no errors. Kinross also did a manual 5% check of the 2010 drill assay data on site in June 2010. No errors were identified.

12.7 Comments on Section 12

RPA is of the opinion that database verification procedures for the Project comply with industry standards and are adequate for the purposes of Mineral Resource and Mineral Reserve estimation.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Metallurgical testwork commenced in 2006. Initial testwork and Project design by Aurelian and Kinross focused on a pressure oxidation (POX) flowsheet. Prior to the 2015–2016 metallurgical programs, Kinross conducted a metallurgical program to assess the potential of a flowsheet to produce a saleable concentrate in conjunction with the production of doré from cyanidation of a gravity concentrate and flotation tailings. This work assessed the differences between a gravity, flotation, leach (GFL) versus a gravity, leach, flotation (GLF) flowsheet. The outcome of the testwork indicated that the GFL flowsheet was the preferred flowsheet option due to improved metal recoveries and lower capital and operating costs.

Amec Foster Wheeler reviewed the Kinross data and, due to the capital costs associated with a POX plant, concurred with the GFL flowsheet approach.

Much of the initial testwork summarized in Section 13.2 is not relevant to the current design, but is included for completeness.

13.2 Historical Metallurgical Testwork

13.2.1 Testwork Completed Prior to the 2011 Kinross FS

At the time of the 2011 Kinross FS, the following work had been undertaken.

2006–2007 Testwork

The first two phases of metallurgical testwork on the Project was completed by SGS Lakefield (SGS), Ontario, Canada, under the supervision of Micon during 2006–2007.

The first phase of metallurgical testing included a series of preliminary tests on composite samples representing five zones of mineralization identified. The metallurgical program comprised Bond ball work index determinations, leach tests, gravity separation tests, flotation tests and a mineralogical study of the five samples.

The second phase of testing was to investigate the metallurgical response of a bulk concentrate from the FDN mineralization to leaching following oxidation pre-treatment including included ultra-fine grinding, POX, bacterial oxidation and roasting.

Key findings from the first two phases of testwork were:

- Conventional cyanide leach testwork indicated that mineralization is moderately refractory with tied up with pyrite and minor marcasite

- Fine grinding of flotation concentrate gave only minimal improvement in gold and silver recoveries; the Bond work indices suggest relatively high unit power consumption for grinding
- Testing of the mineralization indicates that it is not preg-robbing
- Pre-treatment of sulphide refractory ores can be applied to the ground feed or to a flotation concentrate
- Expected gold recoveries would be around 10% to 20% by gravity concentration, 5% to 15% by flotation tailings leaching and 70% to 80% by the pre-oxidation and leaching of the flotation concentrate
- Expected silver recoveries would be 0% to 8% by gravity concentration, 5% to 15% by flotation tailings leaching and 30% to 75% by the pre-oxidation and leaching of the flotation concentrate.

2009 Kinross PFS Testwork

During the pre-feasibility study, metallurgical test programs were carried out by SGS, G&T Metallurgical Services (G&T), Knelson Research & Technology (Knelson) and FLSmidth & Co. A/S (FLSmidth). Tests were performed on four composites, FDN-1 to FDN-4.

SGS conducted sample characterization, grindability testing and simulation studies, gravity recovery, flotation optimization, cyanide destruction, tailings characterization and paste strength testing. G&T performed gravity recovery and whole ore leach testing. Knelson conducted gravity recovery and modelling studies, and FLSmidth performed thickening and filtration testing.

The chemical composition of the seven primary test samples, comprising a bulk sulphide flotation concentrate and six whole ore samples, including one oxide sample, was determined using a combination of head assay, ICP scan composition and whole rock analysis. Results indicated:

- The gold and silver content of the bulk sulphide flotation concentrate were relatively low at 23.8 g/t Au and 33.0 g/t Ag, respectively. There was a significant amount of mercury present, but only small concentrations of arsenic and graphitic carbon in the concentrate. The primary diluent in the concentrate was silicate gangue which calculated to give 64.7% SiO_2. Low concentrations of CaO and MgO constituents were also present
- Precious metals head grades for all whole ore samples were in a relatively close range of 8.73–10.9 g/t Au and 10.3–15.9 g/t Ag, with metallurgical composite FDN-2 having the highest gold grade and metallurgical composite FDN-1 with the highest silver grade. Total sulphur assays varied from 0.78% to 2.88% and

sulphide sulphur assays from 0.65% to 2.65%. Other notable assays included the relatively high carbonate content in FDN-1 of 1.95% followed by the carbonate content in metallurgical composite FDN-4 of 0.64%. FDN-1 had 0.15% total organic content, which could be a preg-robbing constituent. Metallurgical composite FDN-3 had an elevated mercury assay of 4.9 g/t, while FDN-2 had the highest arsenic assay at 170 g/t. ICP data indicated the presence of minor base metal elements in the form of elevated zinc, antimony and copper values. All zones were high in silica content, with FDN-4 the highest at 83.1%.

Mineralogy studies were performed on the bulk sulphide flotation concentrate and the four zone composites using a combination of optical microscopy, X-ray diffraction (XRD), automated scanning electron microscopy (QEMScan), scanning electron microscopy (SEM) and electron microprobe analysis. Gold deportment was checked using heavy liquid separation and super-panning. Results included:

- The composition of the flotation concentrate, using XRD and QEMScan, indicated the primary mineral was quartz, followed by pyrite, mica and K-feldspar, with lesser plagioclase, and traces of galena, sphalerite, marcasite, chalcopyrite, pyrrhotite, arsenopyrite, chalcocite, covellite, magnetite, hematite, and rutile. Gold grain sizes measured ranged from 1 µm to 106 µm in size with an average size of 8 µm. Gold occurs in the form of electrum and native gold
- The major constituent in all whole-ore composites is quartz which comprises more than 30% of the mineral content. The next most common minerals are K-feldspar and mica, followed by pyrite, chlorite, calcite, and manganite. Trace amounts of galena, sphalerite, chalcopyrite, arsenopyrite, barite, anatase, calcite, native gold, electrum, petzite, hessite, acanthite, Ag-tetrahedrite, tetrahedrite, pyrargyrite, mangolite, freibergite and tellurium were noted in the XRD analysis. The major gold minerals in all four composites were electrum and native gold, with the silver content varying from 22% to 33%.

The four FDN zone composites were evaluated using the SMC mill comminution and Bond ball mill grindability test procedures, together with Bond abrasion tests. Results were:

- The SMC A x b value, which is a measurement of impact breakage resistance, was lowest (hardest) for FDN-1 at 44.5. The FDN-2 sample returned a value of 56.7 while FDN-3 and 4 were 61.5 and 68.5 respectively. The FDN-1 composite is considered as medium in impact breakage resistance, while the other three composites are classified as soft
- Bond ball mill work indices ranged from a low of 16.7 kWh/t for FDN-1 to a high of 22.2 kWh/t for FDN-3. FDN-1 is considered as moderately hard, while FDN-2, 3 and 4 are considered as very hard

- The Bond abrasion index values for the four composites varied from 0.347 to 0.517, and are all considered to be abrasive
- Composite relative densities were in a tight range from 2.60 to 2.70 with FDN-1 having the highest value and FDN-3 the lowest.

Three separate gravity recovery test programs were carried out. Work by Knelson indicated that gravity-recoverable gold (GRG) was 32.7%, while the gravity-recoverable silver was 19.9% for the composite sample tested. The highest stage gravity recoveries for both gold (15.9%) and silver (8.7%) were with a feed size P80 of 98 µm. G&T performed tests on a sample created to test the whole ore POX-leach route. Gold and silver recoveries into the gravity concentrate were 25.6% and 11.6%, respectively. Concentrate from the gravity test on the whole ore POX-leach was subsequently processed by intensive leaching providing gold and silver extractions of 92.9% and 81.6%, respectively. This indicates an overall recovery contribution of 23.8% and 9.5% for gold and silver respectively. SGS also conducted gravity tests on the same composite and achieved of 22.7% and 10.3%, respectively. Two additional gravity tests were performed to generate additional gravity tailings products for POX optimization testing. The results indicate a higher mass pull of 0.88% produced the highest gravity recoveries after intensive leaching of 25.6% for gold and 8.3% for silver.

Gravity circuit modelling studies were conducted to evaluate gravity recovery alternatives at different ball mill circulating loads. A conservative circulating load of 500% was selected and the expected gravity gold and silver recoveries with this circuit are 20% and 5.8%, respectively.

Following the gravity work, testwork was completed to support trade-off studies including whole ore leaching, flotation concentrate bio-oxidation, pressure oxidation and roasting, as well as whole ore pressure oxidation. As a result of this work, Kinross proceeded with a conventional semi-autogenous grind (SAG) comminution circuit in combination with a whole ore pressure oxidation followed by CIL.

Testing at SGS examined processing options to reduce the residue weakly acid-dissociable cyanide (CN_{wad}) concentration to a target value of <10 mg/L. Alternatives evaluated included SO_2/air, Caro's acid, and aging (to produce ionic hydrolysis). Inco SO_2/air was the optimal method of choice for the CIL tailings.

Thickening and filtration testwork was completed by FLSmidth on a variety of FDN samples and processing options. The thickening testwork included flocculant screening tests, flux testing, and finally tests to determine thickener unit area requirements and design underflow densities. Rheology testing was also completed on the thickener product to determine rake torque requirements and underflow manageability. Pressure and vacuum filtration testing were completed by FLSmidth to determine filter area requirements and performance for the flotation concentrate and the paste plant.

SGS performed additional tests in support of the tailings dam designs.

13.2.2 Kinross 2011 FS Testwork

The 2011 Kinross FS was supported by a variety of testwork (including deportment studies) looking at different process alternatives at different tonnages. The study assumed that the preferred flowsheet for extraction would be whole-ore pressure oxidation (WOPOX) followed by CIL.

Deportment Studies

Three pilot plant composites were submitted for gold deportment analysis at SGS. Both microscopic and sub-microscopic gold were studied.

Liberated gold, gold locked in silica and gold attached to silica represented the main microscopic gold phases of the three pilot plant composites. Locked gold ranged from 23–45%, attached gold was relatively constant at about 13% and liberated gold ranged from 25–45%. Pyrite, other sulphides, and non-opaque minerals were found to be the main mineral phases associated with the gold grains.

Microscopic gold and silver analysis revealed that gold–silver alloys, including native gold and electrum, were the main gold minerals in all three composites, containing an average of 74–78% Au. Marcasite had the highest concentration of sub-microscopic gold, with an average concentration of 239 g/t Au. Sub-microscopic gold evaluation found that all three composites had 13 to 19% of the gold content in solid solution.

The main microscopic silver carriers were gold–silver alloys, including native gold and electrum. An average silver concentration of 21–24% Ag was observed in these grains. Other major silver minerals included hessite (59.5–61.7% Ag) and Ag-tetrahedrite–tennantite (5.3–8.6% Ag).

Grinding Studies

The primary objectives of the grinding evaluations were to determine grinding power requirements and preliminary SAG mill and ball mill sizes. Testwork was undertaken by the MacPherson and JKSimMet methodologies.

Comminution tests run on the PQ core samples included JK drop weight and Bond ball mill grindability tests on all 24 composites and Bond low energy impact (crushing work index) tests, rod mill grindability tests, and SAG mill comminution (SMC) tests on six of the 24 composites. In addition, MacPherson autogenous work index tests were performed on three of the six composites that were used for the crushing and rod mill work index tests. Bond abrasion tests were also run on these six special PQ composites. Relative density measurements for each composite were made as part of the JK drop weight and SMC test procedures. SMC and Bond ball mill grindability tests were performed on all 24 HQ grind variability composites, while Bond abrasion tests were run on 17 of the 24 composites. Results are summarized in Table 13-1.

A comparison of the SGS grinding study results for the JKSimMet and MacPherson methods indicated that the two methodologies produced similar SAG/ball mill grinding power requirements and mill sizes. A study completed by Metcom Technologies Inc. (Metcom) was used to further refine the 2011 Kinross FS comminution circuit design. Total grinding circuit power consumption was 8.8% higher for the Metcom study when compared with the SGS recommendations.

Whole-Ore Carbon-in-Leach and Carbon-in-Leach Stope Composite Testing

CIL geometallurgical testing was applied on ore selected to fill in the gaps in the prefeasibility study geometallurgical model, and to further evaluate the regression equation for CIL gold recovery with the cyanide soluble to fire assay (CN/FA) gold ratio to again attempt to use this relationship in geometallurgical modelling from the large 5 m composite database. This work was performed at G&T and SGS.

Six special CIL stope composites were also prepared in November of 2010 in order to test larger core samples. This testwork was carried out at both SGS and Canada Centre for Mineral and Energy Technology–Mining and Mineral Science Laboratories (CANMET–MMSL) to compare recoveries between the two laboratories. In addition, SGS also investigated the impact of gravity separation on overall gold recovery. CANMET–MMSL performed testing with pre-aeration and lead nitrate to try and improve gold recovery and reduce iron dissolution, as well as testwork to evaluate the effect of longer CIL residence time.

SGS and CANMET–MMSL work produced similar results for CIL recovery and reagent consumptions. Gravity separation gold recoveries for stope composites 3650-4 and 245-5 were 30.1% and 21.2%, respectively. Project-standard CIL tests were run on each of these stope composites at SGS. CANMET–MMSL also ran CIL tests to evaluate the effect of pre-aeration (eight hours), pre-aeration (eight hours) and lead nitrate (500 g/t) and increased CIL retention time from the standard 24 hr to 48 hr. Comparison of the CANMET–MMSL results for the stated conditions with the baseline SGS test results indicate there were only minor differences in CIL gold recovery or cyanide and lime consumptions for the two composites, including doubling of the CIL retention time to 48 hr.

Table 13-1: Parameter Statistics

Statistics [1]	JK Parameter		Work Indices (KWh/t)				AI 7 (g)
	A x b [2]	Ta [3]	CWI [4]	RWI [5]	BWI [6] (75 µm)	BWI [6] (53 µm)	
Minimum	77.8	0.82	6.0	14.5	15.1	14.2	0.196
Maximum	38.7	0.34	9.4	18.0	24.3	28.4	0.910
Average	53.1	0.54	8.0	16.2	19.6	19.8	0.447

Notes: 1. Only includes samples intersecting POX and/or CIL mine stopes; 2. A x b: JK Rock breakage parameter; 3. Ta: JK abrasion parameter; 4. CWI: Crusher work index; 5. RWI: Rod mill work index; 6. BWI: Bond work index; 7. AI: Abrasion index

Whole-ore CIL (WOCIL) tests with pre-aeration were also performed by CANMET–MMSL on stope composite 245-6 and a combined composite of all six individual stope composites to determine if iron dissolution could be reduced to a low level. The CIL test with pre-aeration on stope composite 245-6 was very successful in reducing iron dissolution from 81.2 ppm to 1.4 ppm. However, pre-aeration was only able to reduce the iron concentration from 67.8 ppm to 42.5 ppm on the combined CIL stope composite. Pre-aeration also increased lime consumption.

Tailings were sent for settling and characterization work together with cyanide destruction and material characterization.

Whole Ore Pressure Oxidation Testwork

Between December 2010 and February 2011, SGS conducted three autoclave pilot plant campaigns using whole ore composites. The testwork indicated good sulphide sulphur oxidation and gold recovery. The POX work was undertaken at a fine grind size of 60 µm.

During the three pilot plant campaigns, 12 pilot plant runs were performed and autoclave profile samples were collected. Gold recovery of each of the neutralized profile samples was also measured to investigate the relationship between CIL gold recovery and the sulphide sulphur oxidation extent. It was found that while gold recovery of the ore increases with the sulphide sulphur oxidation, the effect diminishes as the oxidation extent increases. No correlation was found between silver recovery and sulphide sulphur oxidation.

Variability Tests

A series of variability batch tests was performed on the three pilot plant composites, as well as six variability composites blended from the PQ core samples.

Results of the variability test showed that all samples had achieved a sulphide sulphur oxidation greater than 95%. Gold recovery was satisfactory for all the tests, except for Variability Composite #2, whose gold recovery was less than 89%. Re-leaching test of the sample showed a gold recovery of 97.4%, suggesting that high recovery could be achieved with additional cyanide.

Downstream Testwork

An extensive testwork program was conducted on the material from Campaign #3 of the pilot plant including cyclone testwork, tailings settling and characterization work together with cyanide destruction and material characterization.

Preg-Robbing

The 2011 Kinross FS testwork found that a chloride concentration of about 150 ppm was expected to accumulate in the process water, due to potassium chloride leaching from the ore. At these chloride levels, preg-robbing in the autoclave would cause gold recovery to drop if total carbonaceous matter (TCM) levels were high enough to adsorb the dissolved gold chloride. Despite the demonstrated potential for dissolved gold adsorption, it was concluded at that stage of the Project that the risk of preg-robbing was relatively low and chloride treatment of the autoclave feed was not necessary, based upon the high recoveries achieved in the pilot plant tests.

Cyanide Destruction Testwork

Cyanide destruction (CND) testwork on the POX tailings samples conducted by SGS showed that with proper addition of reagents it is possible to control the CN_{WAD} under 5 mg/L.

Thickening Testwork

Thickener testwork was performed by FLSmidth at the SGS facilities on samples representing various feed streams of the anticipated thickening stages in the flow sheet. These thickening stages included the grind thickener, tailings thickener and paste backfill thickener. Samples were tested at grinds from 40 µm to 60 µm, with grind thickener samples at natural pH and cyanided samples tested at a pH of 10. Samples generated by cycloning were also tested for suitability as paste backfill material.

13.2.3 Kinross 2012–2013 Testwork

Additional work was performed in 2012 and 2013. This testwork examined three flowsheets which included:

- Gravity–flotation–leach (GFL): where flotation is conducted after the initial gravity process in order to produce a saleable concentrate. The tailings from flotation are then leached by CIL to produce doré in conjunction with the intensive cyanidation of the gravity concentrate

- Gravity–deslimed–flotation–leach (GDFL): has size separation performed after gravity and before flotation. The coarser product undergoes flotation to provide a saleable concentrate while the slimes are combined with the flotation tailings and undergo CIL. Doré is produced from the leaching performed on the gravity concentrate and the CIL

- Gravity–leach–flotation (GLF): where leaching is performed on the gravity tailings prior to flotation. This approach puts more of the gold value into doré produced from the gravity pre-concentration and CIL. A concentrate is produced from the flotation of the leach tailings.

The process development testwork in this phase was on a composite sample. All the flowsheets targeted a low mass pull (<1.0%) in the gravity circuit, together with the same 6% mass pull into the flotation concentrate in all three flowsheets. While the cyanidation conditions were the same for all three flowsheets, there were some minor differences in the flotation conditions used.

This development work provided the following results:

- GFL had a recovery range of 84.8 to 90.0% for Au and 85.4 to 94.0% for Ag. With this flowsheet, a concentrate grade of 128 g/t Au and 201 g/t Ag was produced

- GDFL had a recovery range of 82.5 to 86.3% for Au and 75.5 to 81.9% for Ag. With this flowsheet, a concentrate grade of 110 g/t Au and 132 g/t Ag was produced

- GLF had a recovery range for 85.6 to 90.5% for Au and 83.0 to 93.3% for Ag. With this flowsheet, a concentrate grade of 36 g/t Au and 79 g/t Ag was produced.

Overall, the best results were indicated by the GFL approach which had acceptable recoveries and also produced a saleable concentrate. Attempts to perform intensive cyanidation on the concentrates indicated that good recovery was not achievable, and that it would be necessary to produce a saleable concentrate.

Further testwork concentrated on the GFL option to improve performance. This was performed on composites representing the first five years of mine life (5-Year) and a life of mine (LOM) sample on the mine plan that was current in 2012.

This optimization testwork confirmed that the optimum primary grind was in the 60 to 75 µm range and that a natural rougher pH of 7.0 to 7.7 produced satisfactory results. Reagent optimization in this testwork resulted in flotation locked cycle tests which produced 94.8% Au and 92.8% Ag recovery (at a concentrate grade of 105 g/t Au and 135 g/t Ag) for the 5-Year average composite and 93.2% Au and 92.6% Ag recovery for the LOM composite (at a concentrate grade of 141 g/t Au and 227 g/t Ag). Flash flotation was also attempted on the material but showed no significant advantage.

The final work in this phase, reported in 2013, was performed on six composite samples selected to represent low- medium- and high-grade feeds from two domain types. The testing of these six samples indicated a good relationship between head grade and gravity performance. Intensive leaching of the gravity concentrate was typical except when a lower grade gravity concentrate was produced from one of the two domains. In that case, leach recovery was lower at 70%. Flotation produced concentrates ranging from 65.0 g/t Au and 149 g/t Ag to 445 g/t Au and 490 g/t. Overall, the flowsheet produced a recovery range of 83.1 to 99.3% for Au and 80.5% to 96.9% for silver. Whereas one domain produced consistent saleable grades, issues with the other domain indicated the requirement for further testwork to optimize results. Given the range of concentrate grades and to a lesser extent impurities, the work also suggested that the feed blending that will be produced from underground mining will be important to produce a consistent saleable concentrate.

13.3 Testwork for 2016 Feasibility Study

13.3.1 Introduction

Prior to the 2015–2016 metallurgical programs, Kinross had conducted a metallurgical program to assess the potential of a flowsheet to produce a saleable concentrate in conjunction with the production of doré from cyanidation of a gravity concentrate and also that of the flotation tailings. This work assessed the differences between GFL versus a GLF flowsheet. The outcome of the testwork indicated that the GFL flowsheet was the preferred flowsheet option due to improved metal recoveries and lower capital and operating costs.

During the feasibility study the Early MET, FDN MET1 (MET1) and FDN MET4 (MET4) testwork programs were carried out under the supervision of Amec Foster Wheeler. Metallurgical testwork programs were carried out at SGS Minerals S.A in Santiago, Chile for MET1 and at SGS Lakefield in Ontario, Canada, for the Early Testwork and MET4 programs. The results of each testwork program were independently reported by each SGS laboratory. While the early MET program provided early confirmation of the GLF flowsheet, MET1 and MET4 provide the basis for the 2016 FS design. Work completed for MET1 and MET4 is summarized in Table 13-2.

13.3.2 Early MET Program

The purpose of the Early MET program was to assess the differences between a GFL and a GLF flowsheet. The outcome of the testwork indicated that the GFL flowsheet was the preferred flowsheet option due to improved metal recoveries and lower operating costs.

The Early MET program was also used to develop the preliminary flowsheet and conditions for the following MET1 program. These conditions were further optimized during execution of the MET1 program.

Table 13-2: Scope of MET1 and MET4 Testwork Programs

Testwork Component	MET1	MET4
Sample selection	X	X
Chemical characterization including mineralogical analysis	X	X
Physical characterization	X	X
Gravity-recoverable gold (GRG) investigations	X	X
Flotation of fine free gold and gold associated sulphides	X	X
Leaching of flotation tailings	X	X
Carbon absorption testing		X
Cyanide detoxification	X	
Solids settling testwork	X	
Preparation of concentrate for market studies		X

13.3.3 Sample Selection

Metallurgical samples were composited from 1 m drill core intervals for MET1 and MET4.

The MET1 program material was provided from five separate drill cores, which were split to create two types of samples:

- 25 variability samples: proportionally representing the deposit spatially, lithologically, and with respect to the precious metal grade of the deposit
- Composite samples: representing the 0–3, 4–7 and 8–12 Year intervals of the conceptual mine plan available at the time.

The MET4 program sample selection was originally based on the need to produce flotation concentrate for further market studies. As such, the material was selected from two new (MET4) drill cores and available MET1 core rejects (Table 13-3). Intervals were selected on the basis of grade, with remaining material composited to achieve an overall composite head grade that was similar to the conceptual life of mine plan.

13.3.4 Head Characterization

Representative splits from each of the 25 MET1 variability, three MET1 composites and MET4 composite were submitted for chemical analysis (Table 13-4, Table 13-5 and Table 13-6).

Results of the head grade analysis indicate some variability in gold and silver grades with low levels of impurities. The variability in the analysis also suggests some variation in the proportion of free gold throughout the deposit.

Table 13-3: MET1 Variability Sample Head Grades

Element	Au	Ag	Fe	Ni	Pb	Zn	Co	As	Hg	Cl	Si	S	Al
Units	g/t	ppm	%	%	%	%	%	%	ppm	%	%	%	%
Method	FAG001V	AAS042D	AAS042D	AAS042D	AAS042D	AAS042D	AAS042D	AAS036G	AAS012D	ISE108G	AAS091A	LEC010B	AAS091C
Detection Limit	0.1	1	0.01	0.001	0.001	0.001	0.001	0.005	0.1	0.005	0.01	0.01	0.01
VAR-1	12.6	12.3	1.03	<0.001	0.003	0.007	<0.001	0.028	0.3	0.010	44.92	0.84	0.94
VAR-2	6.7	5.7	2.05	<0.001	0.011	0.041	<0.001	0.025	0.9	0.007	40.56	2.16	4.63
VAR-3	27.5	24.3	1.59	<0.001	0.014	0.047	<0.001	0.023	1.2	<0.005	37.48	1.63	5.79
VAR-4	17.3	13.3	3.35	<0.001	0.067	0.109	0.001	0.047	1.3	<0.005	40.05	3.79	4.01
VAR-5	71.5	24.7	1.36	<0.001	0.017	0.084	<0.001	0.016	1.5	0.010	42.76	1.27	2.88
VAR-6	8.4	14.0	3.48	0.001	0.017	0.045	<0.001	0.053	<0.1	0.010	36.95	3.6	4.65
VAR-7	4.6	4.3	1.65	<0.001	0.002	0.001	<0.001	<0.005	0.4	0.007	41.11	1.68	4.02
VAR-8	5.9	10.3	2.02	0.007	0.001	<0.001	<0.001	0.008	0.4	0.018	35.14	1.66	4.47
VAR-9	4.1	5.7	0.72	<0.001	0.002	0.001	<0.001	<0.005	1.1	<0.005	41.19	0.54	2.23
VAR-10	7.7	7.0	0.92	0.009	<0.001	<0.001	<0.001	<0.005	0.1	0.025	39.73	0.34	0.84
VAR-11	13.9	13.3	0.56	<0.001	0.105	0.002	<0.001	<0.005	7.6	<0.005	45.6	0.18	1.67
VAR-12	6.3	12.3	2.78	0.006	0.005	0.002	0.001	0.020	0.4	0.009	32.67	2.74	6.6
VAR-13	21.6	31.3	2.11	<0.001	0.013	0.042	<0.001	0.017	1.3	0.018	37.9	2.15	3.24
VAR-14	6.3	13.7	3.01	0.006	0.017	0.107	<0.001	0.024	0.4	0.01	33.37	2.88	4.08
VAR-15	3.3	4.3	3.36	<0.001	0.013	0.038	0.001	0.019	0.6	0.007	33.9	3.64	5.76
VAR-16	5.5	21.3	3.06	<0.001	0.076	0.062	<0.001	0.018	0.6	0.05	33.32	3.36	6.28
VAR-17	10.0	8.3	4.69	0.005	0.010	0.049	0.002	0.051	0.7	0.009	29.27	4.86	4.89
VAR-18	3.4	7.0	2.53	<0.001	0.018	0.016	<0.001	0.015	0.1	0.007	33.38	2.77	6.73
VAR-19	4.1	9.0	2.97	0.005	0.034	0.08	<0.001	0.025	0.1	0.011	31.17	2.8	4.12
VAR-20	5.2	14.3	4.25	<0.001	0.091	0.172	0.001	0.041	0.2	0.014	31.67	4.43	5.69
VAR-21	4.1	6.7	1.49	<0.001	0.003	0.003	<0.001	<0.005	0.6	<0.005	39.07	1.37	2.85
VAR-22	7.0	7.7	1.18	<0.001	<0.001	0.003	<0.001	<0.005	0.2	0.014	38.85	1.09	2.97
VAR-23	11.8	12.3	0.91	<0.001	0.002	0.003	<0.001	0.023	0.7	0.011	37.66	0.72	1.17
VAR-24	11.6	12.0	0.51	<0.001	0.001	<0.001	<0.001	<0.005	0.7	0.018	37.62	0.28	0.61
VAR-25	22.3	8.7	5.23	<0.001	0.027	0.067	0.002	0.045	0.4	0.007	31.6	3.95	5.54

Notes: 1. Au and Ag results are reported as average of triplicate assays; 2. Cd, Sb, Bi were reported below the detection limits for the applied methodologies.

Table 13-4: MET1 Composite Sample Head Grades

Element	Au	Ag	S	Fe	As	Pb	Zn	Cu	Cl	Hg	Si
Units	**g/t**	**ppm**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**ppm**	**%**
Method	**FAG0001V**	**AAS042D**	**LEC010B**	**AAS042D**	**AAS036G**	**AAS042D**	**AAS042D**	**AAS042D**	**ISE108G**	**AAS012D**	**AAS091A**
Detection Limit	**0.1**	**1**	**0.01**	**0.01**	**0.005**	**0.001**	**0.001**	**0.001**	**0.005**	**0.1**	**0.01**
0-3 Year Composite	11.2	15.7	2.92	2.78	0.037	0.032	0.055	0.022	<0.005	1.5	38.1
4-7 Year Composite	9.1	10.0	1.47	1.64	0.023	0.017	0.015	0.006	0.025	1.1	40.8
8-12 Year Composite	17.4	12.5	2.56	2.83	0.033	0.026	0.058	0.015	0.017	0.7	37.5

Notes:

1. Au and Ag results are reported as average of triplicate assays
2. Ni, Cd, Co, Sb, Bi were reported below the detection limits for the applied methodologies.

Table 13-5: MET4 Composite Sample Head Grades

Element	Au	Ag	S	Fe	As	Pb	Zn	Cu	Cl	Hg	Al	Ni	Si
Units	**g/t**	**ppm**	**%**	**%**	**%**	**%**	**%**	**%**	**%**	**ppm**	**%**	**%**	**%**
Method	**FAA35V**	**AAS21C**	**CSA06V**	**ICP46C**	**ICP46C**	**ICP46C**	**ICP46C**	**ICP46C**	**ICP46C**	**FAA35V**	**AAS21C**	**CSA06V**	**ICP46C**
Detection Limit	**0.02 g/t**	**0.5 g/t**	**0.01%**	**4 g/t**	*******	**20 g/t**	**2 g/t**	**0.5 g/t**	**2 g/t**	**0.02 g/t**	**0.5 g/t**	**0.01%**	**4 g/t**
MET4 Composite	10.1	10.7	2.38	2.44	<.02	0.040	0.038	0.014	5.1	10.7	2.38	2.44	<.02

Table 13-6: Comparison of MET Composite Head Grades to Mine Plan

Mine Plan/ Sample	Au (g/t)	Ag (g/t)	Au/Ag	As (ppm)	Fe (%)	S (%)	Au/S
Feasibility Study Mine Plan	9.67	12.74	0.759	394	—	2.34	4.13
MET1 0-3 Year Composite	11.2	15.7	0.713	370	2.78	2.92	3.83
MET1 4-7 Year Composite	9.1	10.0	0.910	230	1.64	1.47	6.19
MET1 8-12 Year Composite	17.4	12.5	1.39	330	2.83	2.56	6.80
Calculated MET4 Head Grade[1]	10.1	10.7	0.943	102	2.44	2.38	4.24

Notes: 1. Head grade calculated from detailed analysis of feed size fractional analysis.

Table 13-6 also compares the average head grades of each sample to that of the updated feasibility study mine plan.

Specific gravity measurements were performed on various samples as part of each MET program. Overall, the average specific gravity was 2.65, with only minor variations observed.

13.3.5 Mineralogy

XRD analysis was performed on each of the 25 MET1 variability samples, with selected samples being further submitted for QEMscan analysis. Additional XRD and QEMscan analysis was carried out on samples of the MET4 composite. Results of the XRD analysis indicate that the samples contain mainly quartz, micas, feldspar, pyrite, and calcite; with traces of rutile, chalcopyrite, lazurite, and jarosite.

QEMscan results highlight the variability of free gold and gold middlings throughout the samples (Figure 13-1). Figure 13-2 illustrates that gold middlings are most commonly gold associated sulphides (pyrite) and gold associated silicates (quartz). Overall, the results between the MET1 and MET4 QEMscans were found to be consistent.

The high degree of variability in free gold, gold associated sulphides and gold associated quartz may play a significant role in recovery contributions made by the gravity, flotation and leaching circuits.

13.3.6 Physical Characterization

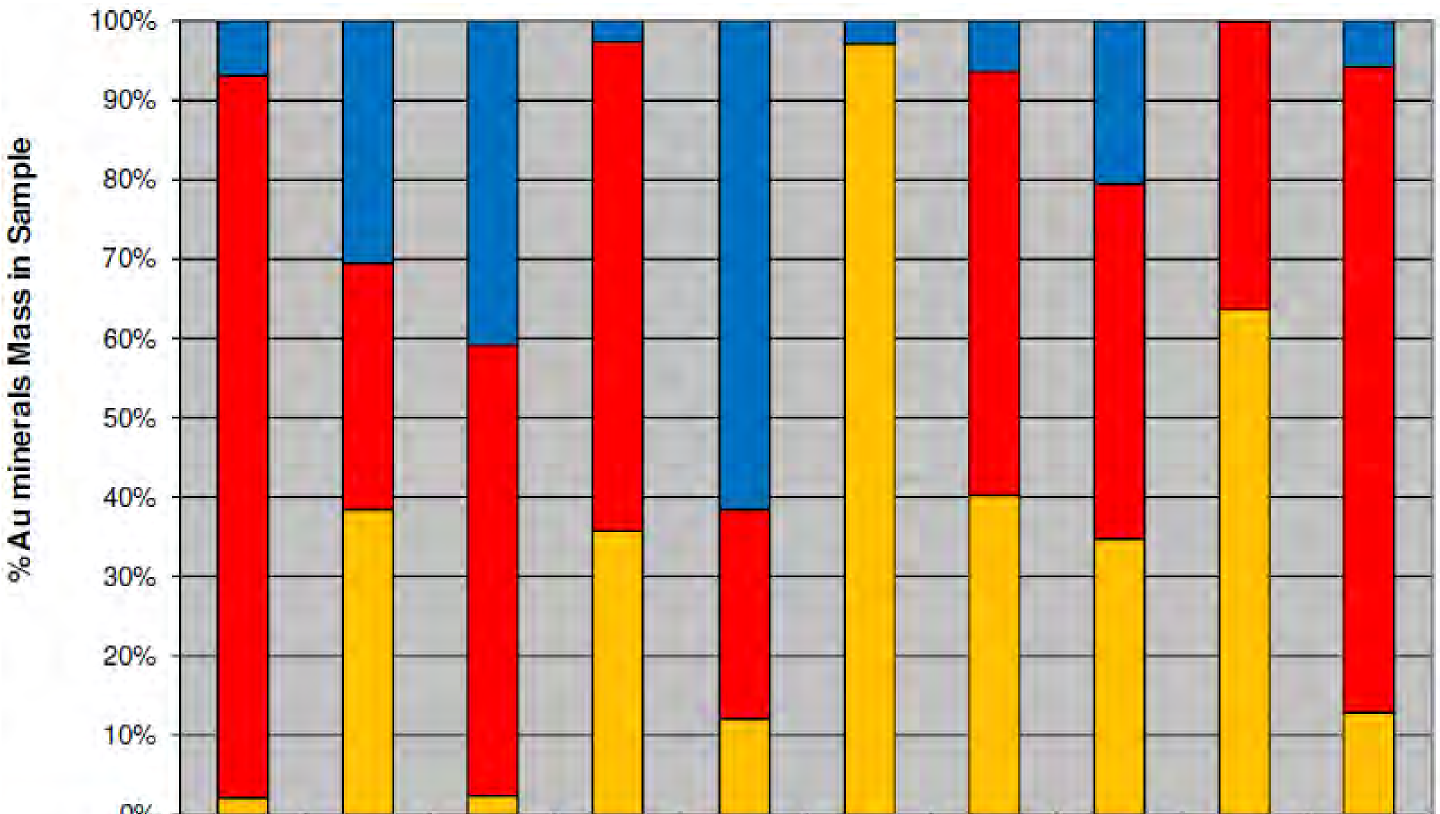
Physical characterization testwork was carried out on selected drill core intervals for both MET1 and MET4 programs. The characterization work included SMC testing and Bond ball mill work indices (BWi). In total, 24 MET1 and 14 MET4 samples were submitted for SMC testing and representative samples of each MET1 composite were submitted for Bond ball mill work indices. The average results of the SMC and Bond ball mill tests are presented in Table 13-7 and Table 13-8, respectively.

Based on the individual SMC results, the orebody can be classified as moderately hard in comparison to data within the Julius Kruttschnitt Mineral Research Centre (JKMRC) database. These results remain consistent with the previous testwork programs and historical data on the deposit. An overall global statistical summary is presented in Table 13-9 and Table 13-10 that summarizes the source of the samples comprising the overall physical characterization database for FDN.

13.3.7 Gravity-Recoverable Gold

Both MET1 (composite and variability) and MET4 samples were submitted to gravity concentration using laboratory scale Knelson concentrators. The Knelson concentrator feed size was approximately 150 µm for both MET1 and MET4 programs. Table 13-11 summarizes the average results obtained for each group of samples.

Figure 13-1: MET1 Variability Head Sample - Gold Deportment

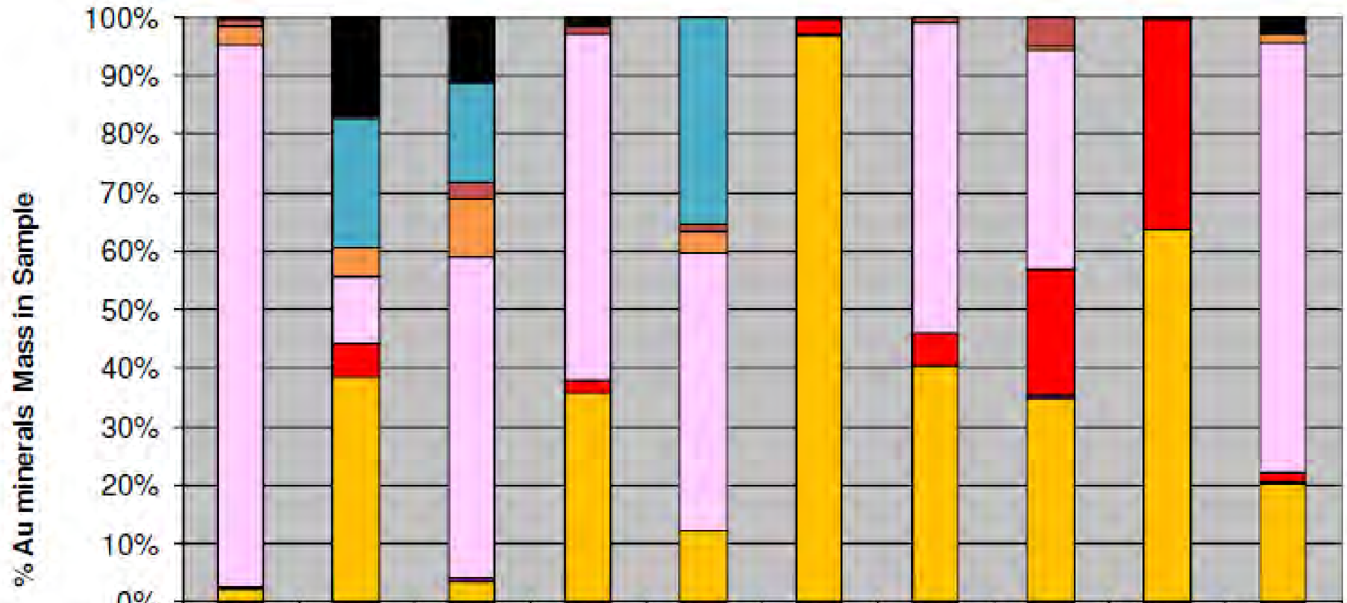


	M-Var 1	M-Var 2	M-Var 3	M-Var 4	M-Var 5	M-Var 6	M-Var 7	M-Var 13	M-Var 16	M-Var 23
Locked Au minerals	6.88	30.56	40.86	2.79	61.52	2.86	6.37	20.52	0.04	5.82
Au minerals Mid	91.04	31.13	56.77	61.59	26.47	0.00	53.35	44.83	36.37	81.33
Free Au minerals	2.08	38.31	2.37	35.62	12.01	97.14	40.28	34.65	63.59	12.85

Sample

Note: Figure from SGS, 2015a.

Figure 13-2: MET1 Variability Head Sample - Gold Associations



	M-Var 1	M-Var 2	M-Var 3	M-Var 4	M-Var 5	M-Var 6	M-Var 7	M-Var 13	M-Var 16	M-Var 23
Complex/Other	0.00	17.36	11.21	1.39	0.00	0.00	0.00	0.00	0.00	2.72
Au Minerals: Pyrite:K-Feldspar: Quartz: Muscovite	0.27	21.78	17.01	0.00	35.29	0.05	0.00	0.00	0.22	0.00
Au Minerals: Pyrite:Quartz	1.22	0.00	2.80	1.39	1.28	0.03	0.80	4.96	0.00	0.11
Au Minerals: K-Feldspar: Quartz	3.32	5.32	10.12	0.00	3.89	0.00	0.00	0.78	0.00	1.55
Au Minerals: Quartz	92.68	11.38	54.68	59.50	47.54	0.00	53.35	37.57	0.03	73.67
Au Minerals: Pyrite	0.44	5.85	0.00	2.09	0.00	2.78	5.57	21.52	36.17	1.50
Au Minerals: Ag Minerals	0.00	0.00	0.65	0.00	0.00	0.49	0.00	0.52	0.00	0.29
Free Au Minerals	2.08	38.31	3.52	35.62	12.01	96.66	40.28	34.65	63.59	20.17

Sample

Note: Figure from SGS, 2015a.

Table 13-7: Summary of MET1 and MET4 SMC Test Results

Statistic	Specific Gravity	Axb	DWI (kWh/m^3)	SCSE (kWh/t)
Average	2.65	45.2	6.05	9.35
85^{th} Percentile	2.70	39.87	6.03	9.68
Minimum[1]	2.54	69.7	3.73	7.72
Maximum[1]	2.77	35.4	7.87	10.62

Notes:

1. Minimum and maximum refer to the softest and hardest results, respectively.

Table 13-8: MET1 Composite Bond Ball Mill Work Indices

Composite	Bond Work Index (BWi)	
	kWh/st	kWh/t
0–3 Year Composite	16.9	18.6
4–7 Year Composite	19.0	20.9
8–12 Year Composite	18.6	20.5

Table 13-9: FDN Global Grinding Statistics

Statistic	Specific Gravity	Axb (JKDW)	Axb (SMC)	RWi (kWh/t)	BWi (kWh/t)	AI
Number of Samples	75	24	57	7	80	23
Average	2.67	50.64	49.27	16.36	19.96	0.44
85^{th} Percentile	2.74	41.78	41.10	17.01	23.51	0.70
Minimum[1]	2.54	70.9	77.8	14.5	14.2	0.20
Maximum[1]	2.82	38.7	35.392	18	28.4	0.91

Notes:

1. Minimum and maximum refer to the softest and hardest results, respectively.

Table 13-10: Summary of Numbers of Grinding Samples

Source	Specific Gravity	Axb (JKDW)	Axb (SMC)	RWi (kWh/t)	BWi (kWh/t)	AI
Historical (2004)	24		24		27	24
Historical (2006)	24	24	6	7	30	6
MET1	13		13		23	
MET4	14		14			
Total Number of Samples	***75***	***24***	***57***	***7***	***80***	***30***

Table 13-11:Summary of the MET1 and MET4 Gravity-Recoverable Gold Tests

Sample	Mass Pull (%)	Assay (g/t)		Global Recovery (%)	
		Au	Ag	Au	Ag
Average of MET1 Variability	0.68	493	333	21.1	17.5
MET1 0-3 Year Composite	0.65	547	368	31.7	14.9
MET1 4-7 Year Composite	0.59	102	10	6.6	0.6
MET1 8-12 Year Composite	0.70	654	254	26.4	13.7
MET4 Composite	0.07	3,141	963	20.6	5.6

The amount of gold that potentially can be recovered by gravity in this deposit is considered high, as supported by the global recovery results of the gravity testwork and QEMscans of the head feed. Of note is the additional recovery of silver, suggesting that a large proportion of free gold is in the form of electrum.

During MET1, excessively high (five times the expected industry norm) mass pulls to the gravity concentrate were observed. As a result, the reported concentrate grades are considerably lower than the MET4 concentrate grades, without a significant reduction in gold and silver recovery.

It was also noted that the MET1 4–7 Year Composite performed poorly during the GRG test. Amec Foster Wheeler compared this result to the individual variability sample results comprising the composite. The individual variability samples were shown to give substantially higher gravity recoveries. As such, it is believed that the MET1 4–7 Year Composite result is unrepresentative of the true sample behaviour.

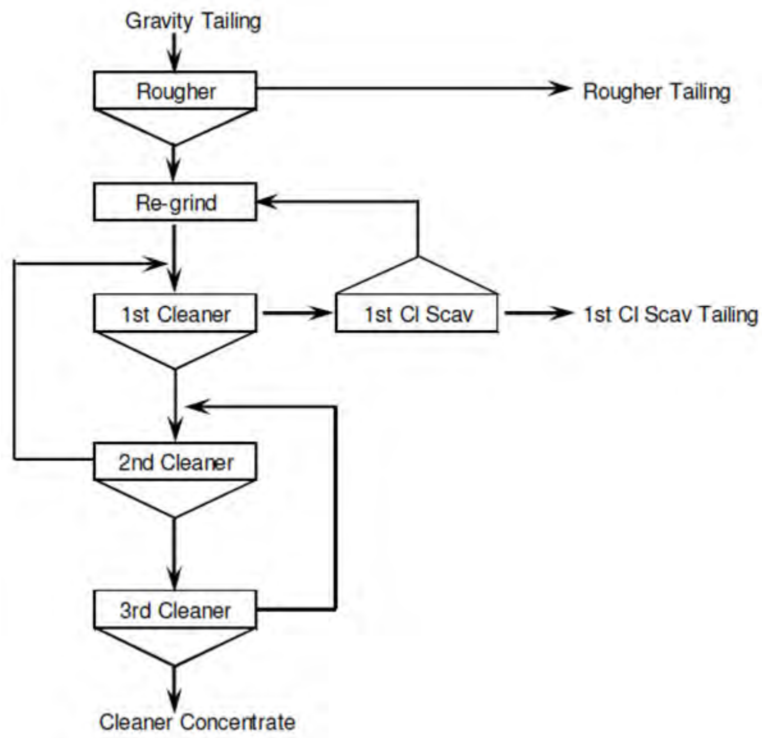
Leaching characteristics of the gravity concentrates were also investigated. Gold extraction rates were found to be consistent with industry norms. MET1 and MET4 composite samples tested achieved between 94% and 98% leached gold recovery from the gravity concentrates produced.

13.3.8 Conventional Flotation

A sulphide flotation test program was developed for the concentration of a gold and silver rich concentrate, knowing that flotation tailings would be subsequently cyanide leached. The objective of the flotation circuit was to recover fine free gold and gold associated sulphides to produce a saleable concentrate.

During the MET1 program each variability sample was subjected to an open circuit flotation test (OCT) to determine the optimal flotation conditions. Subsequently, the MET1 composite sample and MET4 sample were submitted for locked cycle tests (LCTs) at the optimal conditions, using the same flowsheet (Figure 13-3).

Figure 13-3: MET1 and MET4 Conventional 3 Stage Cleaning Flotation Circuit



Note: Figure from SGS, 2015b

All samples tested reported only moderate gold recoveries. The overall flotation process requires lengthy residence time and relatively high reagent dosage to a large degree considered to be the consequences of the middlings gold being a combination of sulphide and quartz associations.

Analysis of the flotation tailings indicates fine free gold, gold associated sulphide and gold associated quartz occlusions, which cannot be recovered by conventional sulphide flotation.

Final concentrates showed reasonable gold and silver grades, with mid-level impurities. Overall, the concentrates produced are considered suitable for sale to a refinery for further processing. Table 13-12 and Table 13-13 summarize the concentrate grades, recoveries and impurities levels obtained.

Table 13-12: Summary of Flotation Concentrate Grade and Recoveries

Sample	Mass Pull (%)	Concentrate Grade					Global Recovery (%)				
		Au (g/t)	Ag (g/t)	As (%)	S (%)	Fe (%)	Au	Ag	As	S	Fe
Average of MET1 Variability	2.53	208.6	202.3	0.593	43.3	39.0	82.7	83.0	83.0	88.4	72.0
MET1 0-3 Year Composite	3.00	131.9	225.0	0.540	42.5	38.2	85.5	90.5	84.8	89.6	78.8
MET1 4-7 Year Composite	1.00	252.4	214.0	0.620	42.1	38.6	78.5	80.5	76.4	84.9	63.8
MET1 8-12 Year Composite	2.60	241.4	168.0	0.620	45.3	40.1	84.0	78.0	87.9	90.8	73.4
MET4 Composite	4.40	126	171	—	44.6	38.9	67.2	67.7	—	79.9	67.0

Table 13-13: Average Element and Impurities Contained in Final Concentrate

Element	Units	MET1 Composite Average	MET4 Composite
Gold	g/t	197.1	126
Silver	g/t	203.5	171
Sulphur	%	42.8	44.6
Iron	%	38.8	38.9
Arsenic	ppm	6,099	7,200
Copper	%	0.28	.27
Zinc	ppm	4,729	8,000
Lead	ppm	3,213	6,400
Tellurium	ppm	50.7	<30
Molybdenum	ppm	61.0	<50

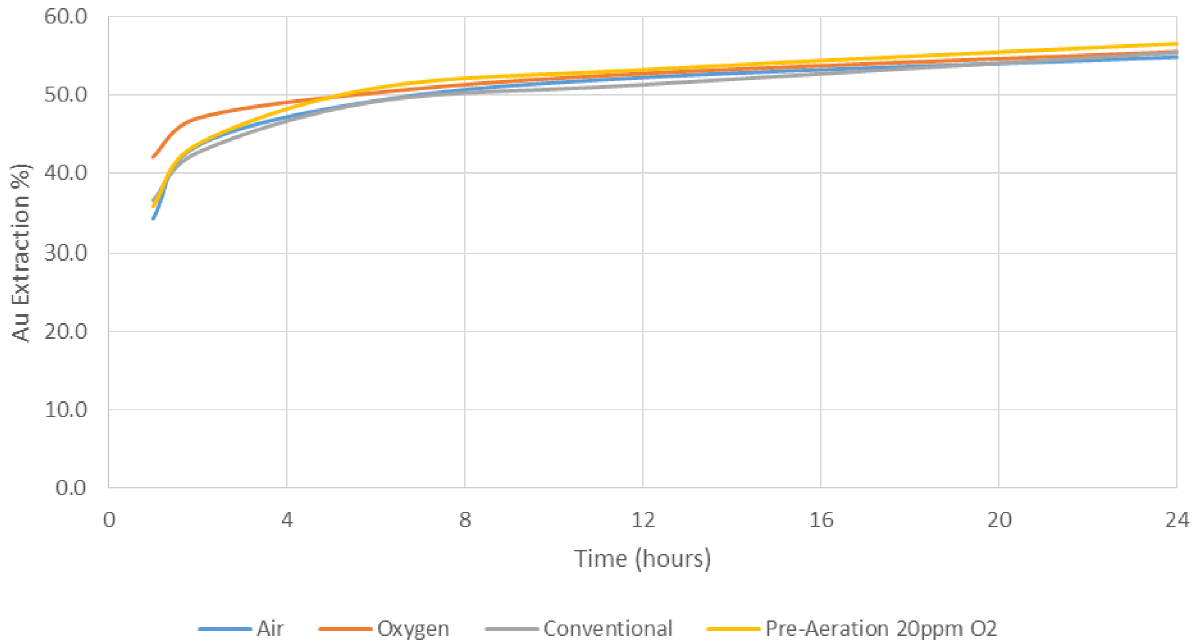
13.3.9 Leaching

Bottle roll leaching tests were performed on each variability and composite sample (including MET4). During the MET1 composite testing, kinetic studies were carried out using air and oxygen injection methods. In addition a pre-oxidation stage was tested to determine the optimal leaching conditions.

Kinetic testing of each composite showed negligible difference between using air, oxygen and pre-oxidation. Ultimate leach recoveries between 51.3% and 64.4% were obtained after 24 hours of leaching. Figure 13-4 presents the typical leach kinetics observed in all MET1 samples. The MET4 program leaching results confirmed the ultimate recoveries obtained in the MET1 program.

Cyanide consumption during the leach tests was low due to the recovery of sulphides to the concentrate during the flotation stage.

Figure 13-4: Average Gold Leaching Kinetic Curve (MET1 Composites)



Note: Figure prepared by Amec Foster Wheeler, 2016.

13.3.10 Supplementary Testwork

Cyanide Detoxification

Leached tailings of the MET1 composite samples were submitted for cyanide detoxification testing, using the Inco SO_2/air process. The results of the cyanide destruction were positive, yielding <0.1 mg/L CN_{WAD} after two hours (on average). Total cyanide at the completion of batch tests was measured at <0.4 mg/L.

Reagent consumptions are considered reasonable, ranging between 3.84 and 4.71 g SO_2/g CN_{Total}.

Sedimentation

Settling testwork was carried out on detoxified MET1 tailings composite samples to determine the optimal flocculant dosage and corresponding settling rate. Only Magnafloc 333 was used, based on the flocculant screening results of the Early MET program. Flocculant dosages between 30 and 60 g/t were tested, with the highest dosages obtaining the highest settling rates.

13.4 Recovery Estimates

The MET1 and MET4 programs generated recovery values which are summarized in Table 13-14.

Using the MET1 variability results, recovery relationships for gold and silver were developed using the Au/S and Au/Ag ratios of the feed. The relationships developed were confirmed by the results of the MET1 and MET4 composite samples.

All recovery relationships are bounded by the condition of the Au–S ratio of the flotation feed ≤10 g/t Au:%S. The boundary was checked against the monthly reported grades and resulting Au–S ratios of the feasibility study mine plan. All monthly values reported in the mine plan were found to fit within this boundary.

The MET1 results suggested some discrepancies in the MET1 testwork. As a result, Amec Foster Wheeler adjusted the MET1 composite recoveries to take account of the material losses in the LCTs and potential improvements that may be achieved in full-scale production:

- A reduction in the mass loss of tailings, concentrates and intermediate streams to approximately 100 g or 0.8% in the flotation locked cycle tests. The adjustment was made based on previous metallurgical testwork programs at SGS
- Recalculation of the final concentrate grades and associated mass to assume a mass of impurities of 10% in the final concentrate (10% is considered a normal operating level for impurities contained in concentrate).

Due to modifications in the gravity and flotation circuit, the feed grade to the CIL circuit was reduced. It was assumed that CIL circuit recoveries would be maintained and this resulted in an overall reduction in CIL recovery.

13.4.1 Gold Recovery

Gold recovery relationships were developed for the flotation circuit (grade and recovery curves) and for the total number of gold units reporting as doré (via gravity recovery and flotation tailings CIL).

By combining the doré recovery relationship and the flotation recovery relationship, the overall recovery for the entire flowsheet can be calculated as follows:

$$\mathrm{Recovery} = 1.7631\frac{Au}{Ag} + 90.3922$$

The metallurgical recovery relationships are considered valid predictors despite the low R^2 value. In selecting the reported metallurgical relationships consideration was given to the degree of scatter of the dataset and alignment to metallurgical theory.

Table 13-14:Summary of Metallurgical Recoveries

Sample	Au Global Recoveries				Ag Global Recoveries			
	Gravity[4]	Flotation	CIL	Total	Gravity[4]	Flotation	CIL	Total
Average of MET1 Variability Samples[1]	19.0	61.8	10.8	91.7	13.4	62.1	0.5	76.0
MET1 Composite 0–3 Years[2]	30.1	58.0	5.6	93.7	11.6	70.2	5.5	87.3
MET1 Composite 4–7 Years[2]	6.2	70.5	14.7	91.4	0.5	56.6	18.6	75.7
MET1 Composite 8–12 Years[2]	25.5	59.4	8.7	93.6	12.0	66.8	6.9	85.7
MET4 Composite[3]	20.6	53.2	13.3	87.1	5.6	63.9	17.2	86.7

Notes:

1. Flotation trials of MET1 variability samples performed in open circuit. All other flotation results are reported from locked-cycle tests.
2. MET1 composite recoveries have been adjusted based on Amec Foster Wheeler knowledge of the interpretation of the testwork results and known testwork errors.
3. MET4 composite results are reported for disclosure purposes only. The program was undertaken to generate concentrate for marketing purposes and the flotation result obtained is considered to be sub-optimal.
4. Gravity recoveries include intensive leaching of Knelson gravity concentrates.

Further confidence is gained when considering that all relationships were developed solely on the MET1 variability samples and subsequent MET1 and MET4 composite tests showed results close to the relationship (Figure 13-5).

Final Flotation Concentrate Grade

The analysis of the testwork showed that it was possible to predict the quality of the concentrate produced as a function of gold to sulphur ratio:

$$C_{Au} = 34.604 \frac{Au}{S} + 6.1098$$

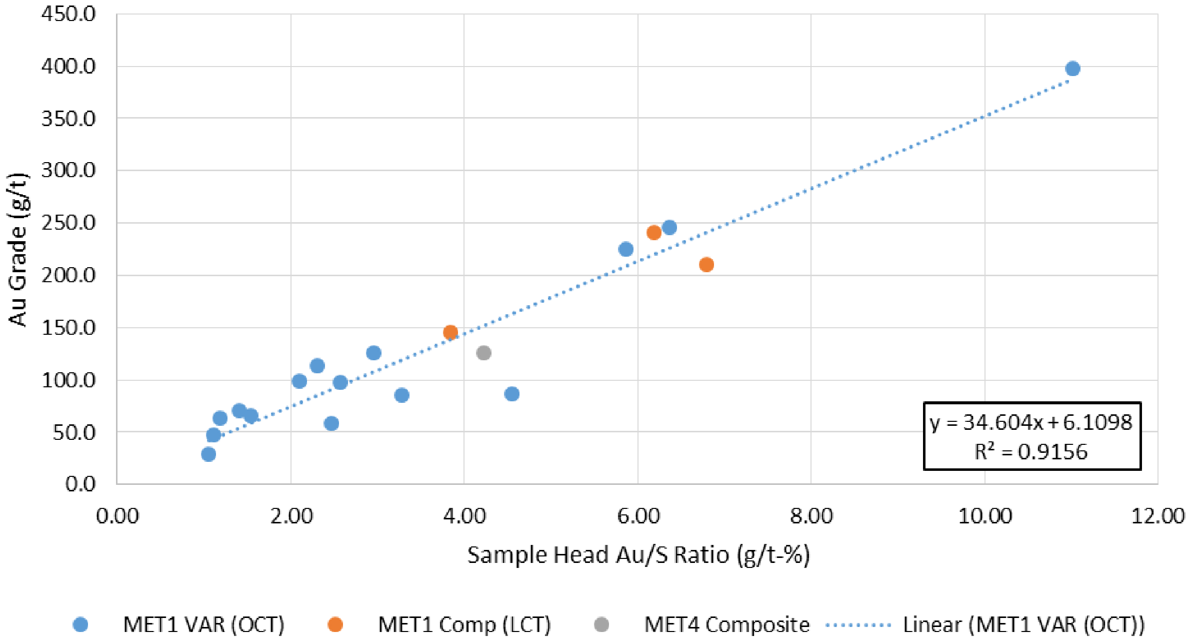
Where:

C_{Au} represents the final flotation concentrate gold grade in grams per tonne (g/t)

Au–S represents the (in g/t) to sulphur (in %) ratio in the feed to the processing facility.

Controlling the quality of the final concentrate is critical to its saleability.

Figure 13-5: Third Cleaner Concentrate Grade Recovery Relationship



Note: Figure prepared by Amec Foster Wheeler, 2016.

Recovery to Final Flotation Concentrate

For flotation recovery, the ratio of silver to gold was found to provide a relationship:

$$R_{Au} = -21.719\frac{Au}{Ag} + 81.212$$

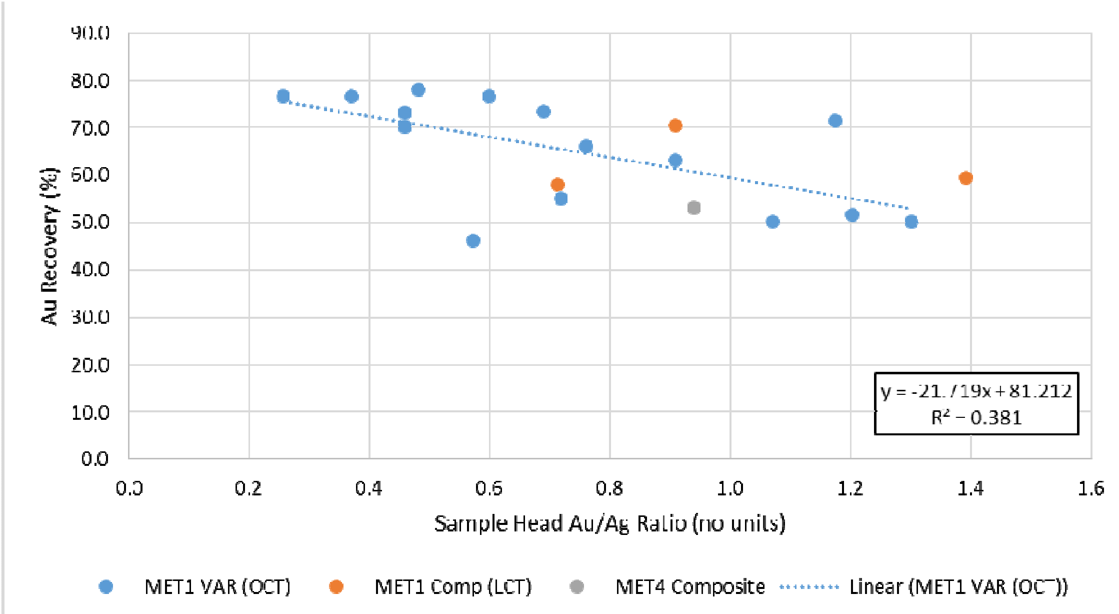
Where:

R_{Au} represents the overall final flotation gold recovery with respect to the processing facility feed

Au/Ag represents the gold to silver ratio in the feed to the processing facility.

This relationship is illustrated in Figure 13-6.

Figure 13-6: Global Gold Flotation Recovery Relationship



Note: Figure prepared by Amec Foster Wheeler, 2016.

To determine the number of tonnes of flotation concentrate produced, the flotation concentrate grade relationship and flotation recovery relationship were combined to yield the following equation for concentrate mass:

$$Mass = \frac{-21.719\dfrac{Au}{Ag} + 81.212}{34.604\dfrac{Au}{S} + 6.1098}$$

Doré Recovery

Recovery to doré also uses the gold to silver ratio as the predictor. The effect is opposite in that a higher recovery occurs with the higher level of silver. It should be noted that the testwork is on gravity concentrate and flotation tailings. With lower recovery to a flotation concentrate, there would be a compensating impact on leach recovery.

$$Dore_{Au} = 23.45\frac{Au}{Ag} + 9.1802$$

Where:

$Doré_{Au}$ represents the overall gold recovery, as doré from the gravity and CIL circuits, with respect to the processing facility feed.

Au/Ag represents the gold to silver ratio in the feed to the processing facility.

This relationship is illustrated in Figure 13-7.

Current recovery estimates are based on the MET program testwork results. The life-of-mine plan (LOMP) average gold metallurgical recovery is set at 91.7%. Actual gold recoveries are expected to range between 91.4–92.1%, peaking in 2023 when high-grade ore is processed, then reducing until 2031. The recovery increases for the last two years of operations (Figure 13-8).

13.4.2 Silver Recovery

The LOMP average silver metallurgical recovery is set at 81.6%.

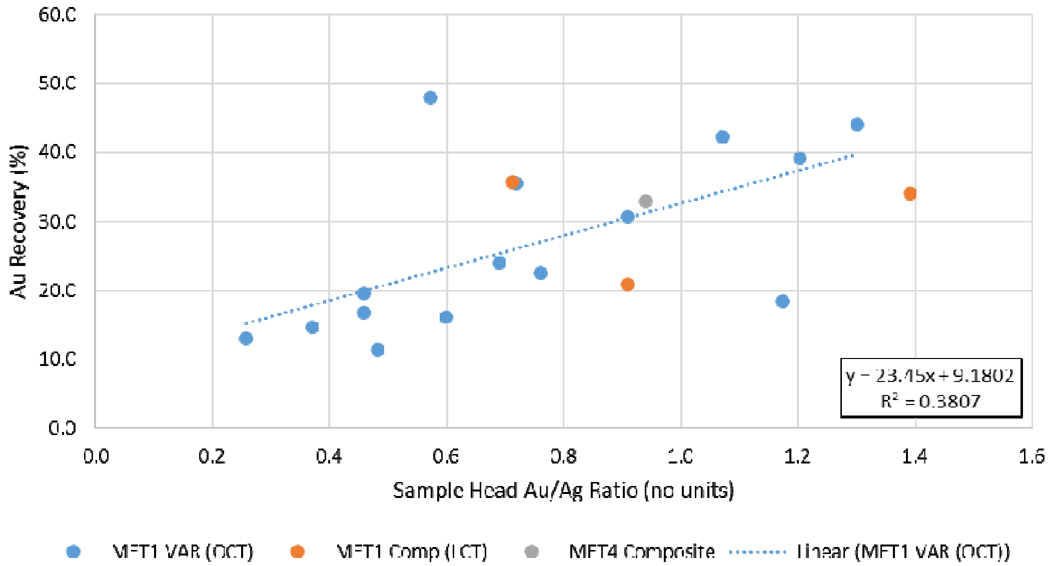
13.5 Metallurgical Variability

The metallurgical testwork completed to-date is based on samples which adequately represent the variability of the proposed mine plan.

13.6 Deleterious Elements

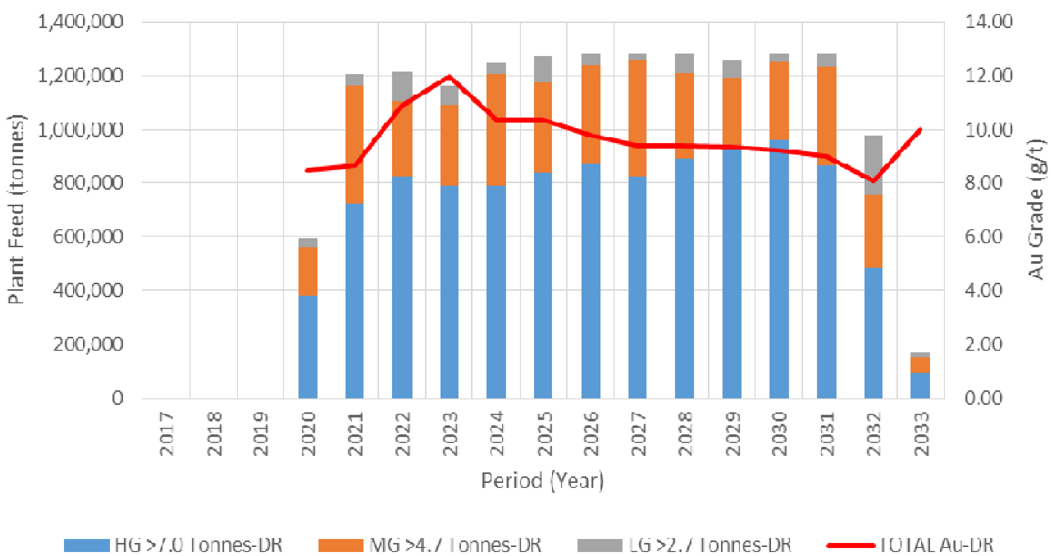
The two products of the plant, the concentrate and the doré, are considered to be saleable without major penalties. It is expected that the level of arsenic and mercury in the flotation concentrate can be maintained at acceptable levels. Final concentrate grades and impurities, based on the testwork samples, were as indicated in Section 13.3.8, in Table 13-12 and Table 13-13.

Figure 13-7: Global Gold Dore Recovery Relationship



Note: Figure prepared by Amec Foster Wheeler, 2016.

Figure 13-8: Projected Recoveries in LOMP



Note: Figure prepared by Amec Foster Wheeler, 2016. DR = diluted and recovered. HG = high grade, >7 g/t Au DR; MG = medium grade, >4.7 g/t Au DR; and LG = low grade, >2.7 g/t Au DR.

13.7 Comments on Section 13

The QP notes:

- The samples tested for the deposit are considered to be representative of the mill feed and LOMP on a grade and spatial basis
- Acceptable gold recoveries were achieved using a GFL (gravity, flotation and leaching circuit) flowsheet. This has resulted in a reasonable overall gold recovery
- A saleable gold–sulphide-rich flotation concentrate was obtained using a conventional sulphide flotation process. Reasonably high concentrations of silver were also obtained in the concentrate
- The concentrates produced during the most recent testwork phase contain relatively low levels of impurities and should not be subject to penalties (refer also to discussion in Section 17 and Section 19)
- Further fine free gold could potentially be recovered through the addition of a separate slimes flotation circuit. This would, however, have to be examined through further testwork and engineering
- Additional optimization testwork could be completed in the areas of flotation residence time and reagent dosages. This may lead to further reductions in operating costs for the plant
- Optimization testwork could also be completed for settling testwork, which may lead to a reduction in thickener area or optimized flocculant usage.

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The central core, located in the northern half of the deposit, was drilled at 35 m to 50 m spacing; the south half of the deposit was drilled on 100 m spaced sections. A total of 246 drill holes were used in the estimate. There was no drilling on the Project in the years 2013 to 2014 inclusive. Assay results from the 2015 drilling were not available at the time of the resource estimate update. Therefore the most recent drill holes used to estimate Mineral Resources were drilled in 2012, and the effective date of the current Mineral Resource model is 1 December, 2015.

Twenty-eight drill holes have no associated assay data since these did not intersect mineralization or the data are missing. Drill holes CP-07-116, CP-07-128, CP-07-145 and CP-08-229 collapsed during drilling and were not logged. Drill holes FN3400g01, FN3510g01, FN3510g02 and FN3650g01 intersect the interpreted mineralization but do not have associated assay or lithology data and were therefore treated as no data.

Forty-nine holes totalling 12,529 m were drilled in 2015 for various purposes including geotechnical, metallurgical, and structural geology. Assay data for these holes were not available at the time of resource grade interpolation and therefore were not included in the estimates. Subsequent to the completed estimate, RPA has tested these new data against the block model and has confirmed that these holes have no significant effect on the Mineral Resource estimate results.

14.2 Exploratory Data Analysis

Various geological versus metal attributes were tested by Kinross in 2011 to determine spatial relationships in an effort to facilitate domain creation. Since only a relatively minor amount of drilling has been done from 2012 to present, the Kinross studies remain current.

Additional work included principal component analysis (PCA) to provide the first step in the clustering of attributes for observing potential correlations, and K-mean clustering to assign attributes into groups called clusters so that the attributes in the same cluster are more similar to each other than to those in other clusters providing spatial relationships.

The correlation tests consisted of the following:

- Length statistics to determine dominance by geological attribute
- Dependency testing to develop baseline relationships for geological attributes
- Correlation matrices to link the geological dependents to geochemical attributes.

The correlation matrices defined a silicic gold-bearing core that consisted of four distinct sub-domains composed of distinctly different mineralogical, epithermal, and geochemical signatures. Conversely, there was a negative correlation of gold mineralization with respect to the clay alteration types that encapsulated the silicic core. The following key geological associations were used to facilitate the sub-domaining of the FDN mineralized core:

- Lithological: strong relationship between breccia types and gold mineralization
- Alteration: weak relationship between gold mineralization and clay alteration
- Alteration: strong relationship between gold mineralization and silica alteration.

14.3 Geological Models

14.3.1 Lithological Units

Logged rock types were grouped into one of 13 lithological units:

- Overburden
- Hollín Formation
- Fruta Andesite
- Suárez Formation mixed sedimentary rocks
- Machinaza Tuff (Suárez Formation)
- Suárez Formation conglomerate
- Sinter facies
- Misahuallí Andesite
- Feldspar porphyry
- Undifferentiated sub-volcanic/sub-intrusive rocks
- Phreatomagmatic breccia
- Hydrothermal breccia
- Zamora Batholith
- Veins.

These units were then divided into four main geological domains based on lithology, alteration and grade criteria. Each domain is distinctive in mineralogical, textural and geochemical character as well as in gold distribution:

- The Xh_Vn domain is dominated by the occurrence of hydrothermal eruption breccia
- The Xp_Ip domain is dominated by the occurrence of phreatomagmatic breccia and feldspar porphyry
- The M_South volcanic domain is located to the south of Xp_Ip and Xh_Vn
- The Silica_Halo envelopes the top and bottom of the three other domains. The Silica_Halo domain hosts some gold mineralization but not with sufficient grades and thicknesses to be modelled as Mineral Resources.

The four zones are believed to represent distinct hydrothermal events starting with the Xp_Ip domain, which is associated with late porphyry events. This was followed by the silica–(arsenopyrite)–marcasite alteration associated with hydrothermal brecciation (Xh) in the up-flow zone centred on section 3400N and "mushrooming" out below the Suárez unconformity. The later-stage quartz–carbonate phase (Vn) appears to have formed in the northern section of the deposit, wrapping partially around a flexure in the feldspar porphyry contact. Xh and Vn were grouped together for resource domaining purposes.

14.3.2 Lithological Models

SRK produced a new lithological model following the 2015 geotechnical and hydrological drilling program using Leapfrog Geo software. The lithological modelling program was an iterative process conducted concurrently with updating the structural model. As far as possible, observed fault kinematics and offsets were preserved within the lithological model. The interpretation was based on the integration of topographic and core hole data, and on direct observations made during two site visits. For the interpretation, Lundin Gold supplied SRK with assay and lithological data for 440 core holes, geophysical data, and LiDAR topographic data.

Modelling considerations applied in Leapfrog Geo software for the various lithological and structural units are summarized in Table 14-1.

14.3.3 Degradation Model

A degradation 3D model was constructed using Leapfrog Geo software. In general, degraded zones are contained within the Suárez Formation conglomerate and its interbedded tuff beds (see discussion in Section 9.6.2). Degradation also occurs to a lesser extent in the siltstone beds in the upper part of the Suárez Formation. Contacts of the Suárez Formation conglomerate were used to extrapolate degradation zones beyond areas of graphic logged intervals in order to construct a degradation shell with moderate confidence.

Table 14-1: Modelling Considerations in Leapfrog Software

Unit	Consideration
Overburden	Maximum thickness of 15 m allowed
Hollín Formation	Basal contacts identified, and a continuous surface produced.
Fruta Andesite, Suárez Formation, Machinaza Tuff and Sinter Facies	Basal contacts for the units in each fault block and deposit surfaces produced. Deposit surfaces compared against drilling and bedding data. If required, surfaces were adjusted by defining the orientation of a reference surface, or by manually adding polylines to produce a geologically reasonable surface. Below the basal contact of the Suárez conglomerate, the lithology was considered to be Misahuallí andesite or feldspar porphyry, depending on the location within the deposit. Due to the generally thin and sporadically discontinuous nature, the Machinaza Tuff and Sinter facies were built as veins. Reference surfaces for the veins defined such that the units correlated with the orientations of contacts between the major units.
Intrusive Units	For each fault block containing hydrothermal and phreatomagmatic breccia logged intervals, a structural trend was defined using the bounding north–south-trending fault orientations. This structural trend was applied to the intrusive bodies to produced geologically reasonable volumes. The upper limit of hydrothermal and phreatomagmatic breccia volume was defined as the contact between the Suárez conglomerate/Sinter facies and the underlying andesitic units. Lithological logging suggests that there may be abundant interlayering between feldspar porphyry and Misahuallí andesite in the central part of the deposit. However, this is more likely related to the ability to of differentiate between the feldspar porphyry and Misahuallí andesite where strong alteration obliterates original textures. This was modelled as a discrete undifferentiated unit that was built using polylines in areas of apparent interlayering. The Zamora Batholith was modelled using polylines, and was based on lithological and geophysical data and surface observations.
Veins	Veins were modelled as intrusive units. Only fault blocks with significant identified vein intersections were modelled. Vein orientation data supplied by Lundin Gold showed two preferred orientations for measured veins in the deposit. These orientations were 243°/-15° and 315°/-65° (dip direction/dip). The greatest continuity between logged vein intervals was observed using an orientation of 315°/-65°. An anisotropy in this orientation was applied to the modelled veins to produce the final volumes

Subsequently, areas of the degradation model that were extrapolated beyond approximately 55 m (locally up to 80 m) between drill holes were clipped out in order to produce a high-confidence degradation model.

14.3.4 Alteration Model

The alteration model was constructed from 18,930 infrared spectra collected from 148 drill holes at a 1.5 m interval for 2015 core holes and one spectrum per box for historical drill holes, and 3D modelling of the key infrared-active alteration minerals and features.

This included modelling sericite (paragonite and phengite), kaolinite, and chlorite shells.

14.3.5 Fault Models

The current fault model is a revision of a fault model produced by SRK for the FDN deposit in 2013 (SRK, 2013). Information supporting the model include site observations, selected drill core data (assay and lithological data; oriented geotechnical data; observations of core box photos), a Leapfrog geological model (FDN6) created by Kinross and maintained by Lundin Gold, and LiDAR topographic data.

A total of 14 faults were modelled and the fault model was used to define fault blocks for subsequent lithological modelling. Several iterations were constructed to refine the structural and lithological models.

14.3.6 Wireframes

Leapfrog and GEMS software were used to build the wireframe models representing the domains. Wireframe models of various alteration and lithology zones were first built in Leapfrog by Kinross in 2011. The second stage of the wireframe construction process was done in GEMS software. The alteration, silicic core and lithological surfaces created in Leapfrog were imported into GEMS for final solid creation. These domains were later modified to honour the lithological and structural domains built by SRK in 2015.

The FDN deposit was divided into four main geological domains based on lithology, alteration and grade (Table 14-2; Figure 14-1; and Figure 14-2).

The four zones are interpreted to represent distinct hydrothermal zones starting with the Xp_lp domain, which is associated with late porphyry intrusion events. This was followed by the silica–arsenopyrite–marcasite alteration associated with hydrothermal brecciation (Xh) in the upflow zone centred on section 3400N and mushrooming out below the Suárez Formation unconformity. The later stage quartz–carbonate phase (Vn) of the hydrothermal brecciation event appears to have formed in the northern section of the deposit, wrapping partially around a flexure in the feldspar porphyry contact. Xh and Vn were grouped together for resource domaining purposes.

14.4 Composites

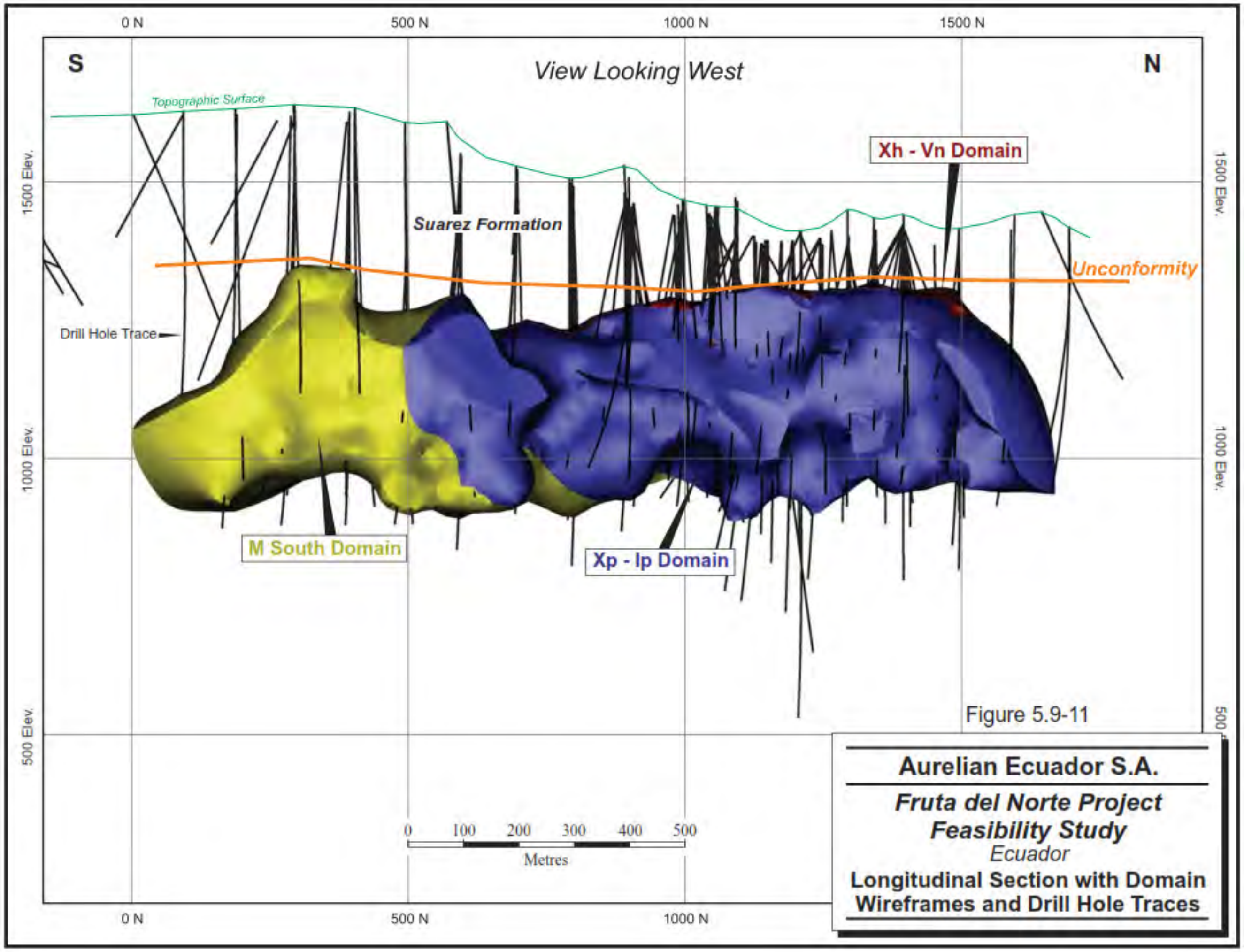
The average length of assayed samples within the mineralized domains Xh_Vn, Xp_Ip, and M_South is 0.99 m. Given the selected block size of 4 m by 10 m by 10 m, a 2 m composite was selected for grade interpolation purposes.

Compositing was performed in GEMS 6.7.1 modelling software. The processing was done by geological domain using intervals from a solid intercept table at 2 m intervals. Composites located at the end of the interval with lengths less than 0.5 m were removed from the database.

Table 14-2: Geological Domains

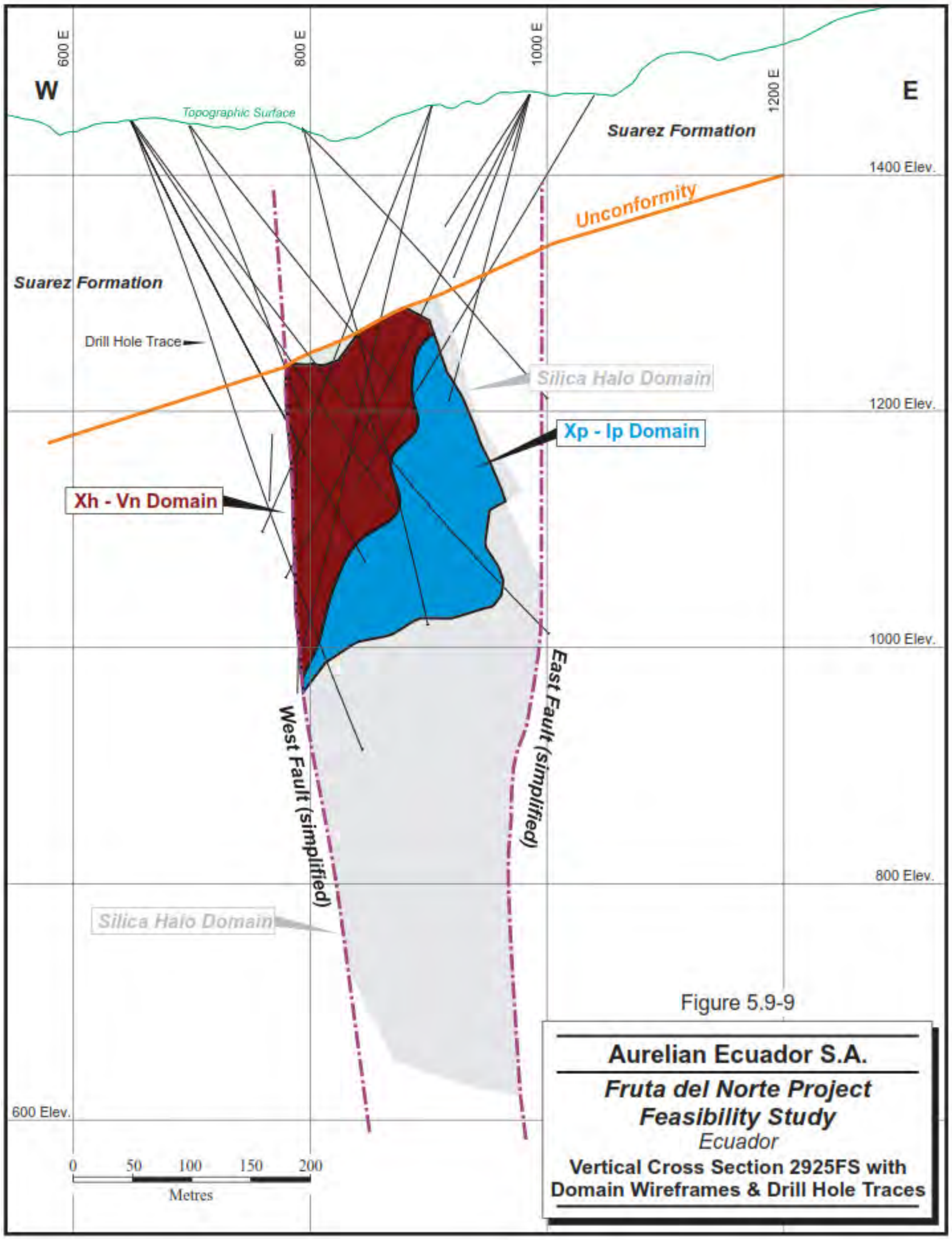
Domain Name	Characteristic	Timing	Comment
Xp_Ip	Phreatomagmatic breccia	Interpreted to pre-date the epithermal system because it does not include epithermal vein clasts, although it is silica altered and includes clasts of volcanic and porphyry rocks.	Vertically to sub-vertically oriented and elongated in the north–south direction, measuring approximately 400 m high by 120 m wide by 1,200 m long. Makes up part of the gold core of the FDN deposit and hosts almost 15% of the contained gold at FDN. Forms the eastern footwall of the deposit.
Xh_Vn	Hydrothermal eruption breccia with abundant veining and stockwork	Interpreted to be synchronous with the epithermal system as it contains clasts of veins and both mineralized and altered wall rocks	Vertically to sub-vertically oriented and elongated in the north-south direction, measuring approximately 370 m high by 130 m wide by 1,000 m long. Hosts approximately 75% of the contained gold
M_South	Volcanic domain		Located to the south of the gold rich core made up of the Xp_Ip and Xh_Vn domains. Approximately 300 m wide by 370 m high by 950 m long, elongated in the north–south direction
Silica_Halo	Envelops the top and bottom of the three other domains		Together with the three other domains, the FDN deposit has a strike length of 1,670 m by 700 m in height. Does not host Mineral Resources or Mineral Reserves

Figure 14-1: Longitudinal Section with Domain Wireframes and Drill Hole Traces



Note: Figure courtesy Lundin Gold, 2016.

Figure 14-2: Vertical Cross Section 2925FS with Domain Wireframes and Drill Hole Traces



Note: Figure courtesy Lundin Gold, 2016.

Assay and composite values located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. Results were used to help verify the modelling process.

14.5 Grade Capping/Outlier Restrictions

The FDN metal capping review consisted of disintegration analysis of the composite values in conjunction with histogram, log probability, and mean variance plots. The disintegration analysis ranks the metal data in ascending order and applies a percent change or step function of 10% to 15% between consecutive values to determine where population breaks occur. This was used in conjunction with the histogram and log probability plots to cross-validate the disintegration population breaks.

In order to preserve the grades within the high-grade zones with intense veining of domain Xh_Vn, composites were left uncapped, and instead a restricted search for gold values greater than 60.0 g/t was applied. A capping value was applied to the silver grades for this domain. The restricted search dimensions are described in Section 14.8. Capping grades are summarized in Table 14-3.

14.6 Density Assignment

The resource database includes 3,511 density measurements. Density data were reviewed by lithology and alteration type (Table 14-4). The average values were assigned to the block model to convert volume to tonnes.

14.7 Variography

14.7.1 Continuity Analysis

Continuity analysis was carried out by Kinross. The first stage involved analyzing a series of fans, in the horizontal, across-strike vertical and dip planes. Each fan was made up of variogram contours to analyze and select the directions of continuities. Each direction was used to determine the placement of the next plane. The selected strike, dip and plunge orientations were used to locate the three directions for which variogram models were constructed.

14.7.2 Variograms

Variography was carried out by Kinross within a 450 m long segment of the deposit with closely-spaced drilling, between northings 9,583,300N and 9,583,750N. Drilling and domains within this segment have not changed significantly since that time and therefore the variogram models remain current.

Table 14-3: Grade Caps

Domain	Element	Number of Samples	Grade Cap Applied (g/t)	Number of Affected Samples
Domain Xh_Vn	Gold	9,513	none	none
	Silver	9,513	148	19
Domain Xp_Ip	Gold	6,485	58.68	16
	Silver	6,485	98.19	14
Domain M_South	Gold	2,807	27.38	17
	Silver	2,807	87.83	20

Table 14-4: Density Data Summary Statistics

	Xh_Vn	Xp_Ip	M_South	All Data
No. Samples	321	335	752	3,511
Minimum (t/m^3)	2.21	2.11	2.28	2.06
Maximum (t/m^3)	2.98	3.08	3.01	3.08
Mean (t/m^3)	2.62	2.72	2.73	2.67
Standard deviation (t/m^3)	0.08	0.08	0.09	0.10
Coefficient of Variation	0.03	0.03	0.03	0.04

Note: "All data" column includes data outside the three domains.

The directions for the major, semi-major and minor axes were selected using a set of variogram maps generated in the horizontal, across-strike vertical and dip planes. Variograms for each direction were created and modelled for each domain. A "Normal Scores" transformation was also used to provide improved variograms. A back transformation from Normal Scores space was completed following the variographic analysis and used as input to the ordinary kriging (OK) interpolation in GEMS software.

14.8 Estimation/Interpolation Methods

Grade interpolations for gold and silver were performed using the OK algorithm and using search strategies individually adapted to each domain. The search ellipses generally have the same orientations, striking north–northeast, dipping west, and plunging north–northeast. A two-pass approach was used, with the first pass search ranges approximately equivalent to the variogram ranges at 80% of the sill. The first pass used a minimum of two drill holes. The second pass used a larger search with a one drill hole minimum.

Both hard and soft boundaries were used, based on various contact analyses and the geological interpretation. Pass 1 applied a hard boundary between domains. Pass 2 used a soft boundary between domains. The interpolation parameters for silver were similar to those for gold.

Table 14-5 summarizes the interpolation parameters for gold and silver.

14.9 Block Model

Two block models were constructed in GEMS Version 6.7.1 for the interpolation process consisting of a partial model set (percentage model) and a consolidated model.

14.10 Block Modelling of Other Elements

Block grades were estimated for sulphur, mercury, arsenic, antimony and lead using a similar database, domain boundaries and general block model set up as used for the gold resource model. Results were used for mine planning purposes. These block models were first generated in 2015 and more recently updated in January 2016. Some of the modelling analyses conducted in 2015 were relied upon for the January 2016 update.

Grades for arsenic, mercury, lead, sulphur and antimony were interpolated into the blocks using OK. For sulphur, no constraint was placed on the composite domain (i.e. all boundaries were considered to be soft boundaries). For the other elements, Domain Xh_Vn estimates were generated using composites from domains Xh_Vn and Xp_Ip. For Domain M_South, the composites used were from domains M_South and the silica halo.

The grade interpolations were validated by visual inspection and comparison with drill hole composite grades and comparison between global block kriged, nearest-neighbour (NN) and declustered composite means.

14.11 Block Model Validation

Validation tests included:

- Visually inspected composite grades versus estimated block grades
- Compared swath plots of composites versus block grades
- Used block variance to observe the degree of model smoothing
- Compared the OK model with NN and inverse distance models
- Compared the global mean grades between the OK model and the composite statistics
- Checked collar locations for unreasonable values
- Checked for reasonably similar shapes used on adjacent sections
- Checked for overlapping wireframes to determine possible double-counting

Table 14-5: Interpolation Parameters

Element	Domain/ Pass No.	Min. No. Samples.	Max. No. Samples	Max. Samples per Hole	Rotation (ZXZ)			Ellipse Range (m)		
					Z	X	Z	X	Y	Z
Gold	*Xh_Vn*									
	Pass 1	3	8	2	75	-75	25	45	30	15
	Pass 2	1	15	15	75	-75	25	250	200	100
	Xp_Ip									
	Pass 1	3	8	2	75	-70	25	45	24	12
	Pass 2	1	15	15	75	-70	25	175	130	80
	M_South									
	Pass 1	3	8	2	75	-75	25	45	30	15
	Pass 2	1	8	2	75	-75	25	250	200	100
Silver	*Xh_Vn*									
	Pass 1	3	8	2	75	-75	25	45	30	15
	Pass 2	1	8	8	75	-75	25	250	200	100
	Xp_Ip									
	Pass 1	3	8	2	75	-70	25	45	24	12
	Pass 2	1	8	8	75	-70	25	175	130	80
	M_South									
	Pass 1	3	8	2	75	-75	25	45	30	15
	Pass 2	3	8	2	75	-75	25	250	200	100
	Silica_Halo									
	Pass 1	3	8	2	75	-75	25	45	30	15
	Pass 2	1	8	2	75	-75	25	200	130	75

Note: In GEMS software, a positive rotation around the Z axis is from X towards Y, and around the X axis is from Y towards Z; capped grades for gold were used in all domains except Xh_Vn; within the Xh_Vn domain, gold grades greater than 60 g/t were restricted to a search ellipse of 30 m by 20 m by 7 m.

- Compared basic statistics of assays within wireframes with basic statistics of composites within wireframes for uncut and cut values
- Checked for missing assay intervals and missing assay values, and explained or verified discrepancies and treatment
- Checked that composite intervals start and stop at wireframe limits
- Checked that all drill hole intersections with the wireframes have been fully composited
- Checked that assigned composite rock type coding was consistent with intersected wireframe coding

- Checked if block model size and orientation were appropriate to drilling density, mineralization and mining method
- Checked interpolation parameters against variography
- Compared block statistics (zero grade cut-off) with assay/composite basic statistics
- Visually checked block Mineral Resource classification coding for isolated blocks
- Visually compared block grades to drill hole composite values on sections and/or plans
- Visually checked for grade banding, smearing of high grades, plumes of high grades on sections and/or plans.

14.12 Classification of Mineral Resources

Mineral Resources were classified into the Indicated or Inferred categories based on drill hole spacing and the apparent continuity of mineralization.

Variography has suggested a range of 35 m at 75% of the total sill. Infill drilling was designed at 35 m spacing. In general, areas of 35 m spacing or shorter were classified into the Indicated category. Other factors that were taken into consideration include the search distance to the nearest composite, estimation by the first-pass search ellipse, visual examination and general considerations of drill fan spacings. Classification was done in GEMS software guided by a 17.5 m (for 35 m spacing) distance buffer generated in Leapfrog software.

Parts of the Xh_Vn and Xp_Ip domains were classified as Indicated Mineral Resources. All of the M_South domain was classified as Inferred Mineral Resources. Due to the lack of exposures of mineralization for inspection on the surface or underground, there are no Measured Mineral Resources at this time.

14.13 Reasonable Prospects of Eventual Economic Extraction

Based on the assumptions in Table 14-6, Mineral Resources were reported at a block cut-off grade of 3.5 g/t Au. Silver was not included in the cut-off grade calculation due to its relatively low grade and small contribution to the value of the mineralization. Parameters used to calculate the cut-off grade were based on numerous metallurgical testwork and engineering studies, which assume an underground mining method and the processing method as described elsewhere in this Report.

14.14 Mineral Resource Statement

The QP for the estimate is Mr. David Ross, P.Geo., an RPA employee. The estimate has an effective date of 31 December 2015.

Table 14-6: Key Assumptions for Assessment of Reasonable Prospects of Eventual Economic Extraction

Item	Unit	Value
Metal recovery	%	94
Gold price	US$/oz	1,500
Mining cost	US$/t milled	60
Processing cost	US$/t milled	35
G&A cost	US$/t milled	15
Surface infrastructure	US$/t milled	28
Concentrate transport and treatment	US$/t milled	7
Total operating cost	US$/t milled	145
Royalties	US$/oz	71
Selling costs	US$/oz	65

Mineral Resources are reported inclusive of Mineral Reserves at a block cut-off grade of 3.5 g/t Au, assuming underground mining methods. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. No Measured Mineral Resources have been declared.

Mineral Resources are summarized in Table 14-7, are inclusive of Mineral Reserves, and have been classified using the 2014 Canadian Institute of Mining and Metallurgy (CIM) Definition Standards for Mineral Resources and Mineral Reserves (the 2014 CIM Definition Standards).

14.15 Factors That May Affect the Mineral Resource Estimate

Factors which may affect the Mineral Resource estimates include:

- Metal price and exchange rate assumptions
- Changes to the assumptions used to generate the cut-off grade value
- Changes in local interpretations of mineralization geometry and continuity of mineralization zones
- Density and domain assignments
- Changes to design parameter assumptions that pertain to stope designs
- Changes to geotechnical, mining and metallurgical recovery assumptions
- Assumptions as to the continued ability to access the site, retain mineral and surface rights titles, obtain environmental and other regulatory permits, and obtain the social licence to operate.

Table 14-7: Mineral Resource Statement

Category	Tonnage (Mt)	Grade (g/t Au)	Contained Metal (Moz Au)	Grade (g/t Ag)	Contained Metal (Moz Ag)
Indicated	23.8	9.61	7.35	12.9	9.89
Inferred	11.6	5.69	2.13	10.8	4.05

Notes:

1. The Qualified Person for the estimate is Mr. David Ross, P.Geo., an employee of RPA. The estimate has an effective date of 31 December 2015.
2. Mineral Resources are reported inclusive of Mineral Reserves; Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Mineral Resources are reported at a cut-off grade of 3.5 g/t Au; which was calculated using a long-term gold price of US$1,500/oz.
4. Mineral Resources are constrained within underground mineable shapes that assume a minimum thickness of 2 m; metallurgical recovery of 94%; total operating costs of US$145/t milled (mining cost of US$60/t milled; process costs of US$35/t milled; G&A costs of US$15/t milled; surface infrastructure costs of US$28/t milled; concentrate transport and treatment costs of US$7/t milled); royalties of US$71/oz and selling costs of US$65/oz.
5. Numbers may not sum due to rounding.

14.16 Comments on Section 14

Mineral Resources have been estimated using the guidance in CIM (2003) and classified using the 2014 CIM Definition Standards.

There are no other known environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Resources that are not discussed in this Report.

15.0 MINERAL RESERVE ESTIMATES

15.1 Block Models

The Mineral Reserve Block Model was prepared by combining the Resource Block Model and the Geotechnical Block Model.

- The Resource Block Model was provided by RPA (November 2015) and consisted of density, grades (gold, silver, arsenic, mercury, lead, sulphur and antimony), rock types (geometallurgical resource domains), resource confidence categories and other impurities. It was built in GEOVIA GEMS software
- The Geotechnical Block Model was developed by SRK. It utilized assessments of lithology, alteration and structure to model three domains that encompassed Poor, Fair–Poor, and Good–Fair rock mass conditions (Figure 15-1). This model was built in Leapfrog.

The models were imported by NCL via ASCII files into DESWIK software. Validation was carried out with 99.9% of the original block model data for Indicated and Inferred Mineral Resources in terms of tonnes, gold ounces and silver ounces.

The Inferred Mineral Resources grades were set to zero for the purposes of Mineral Reserve estimation.

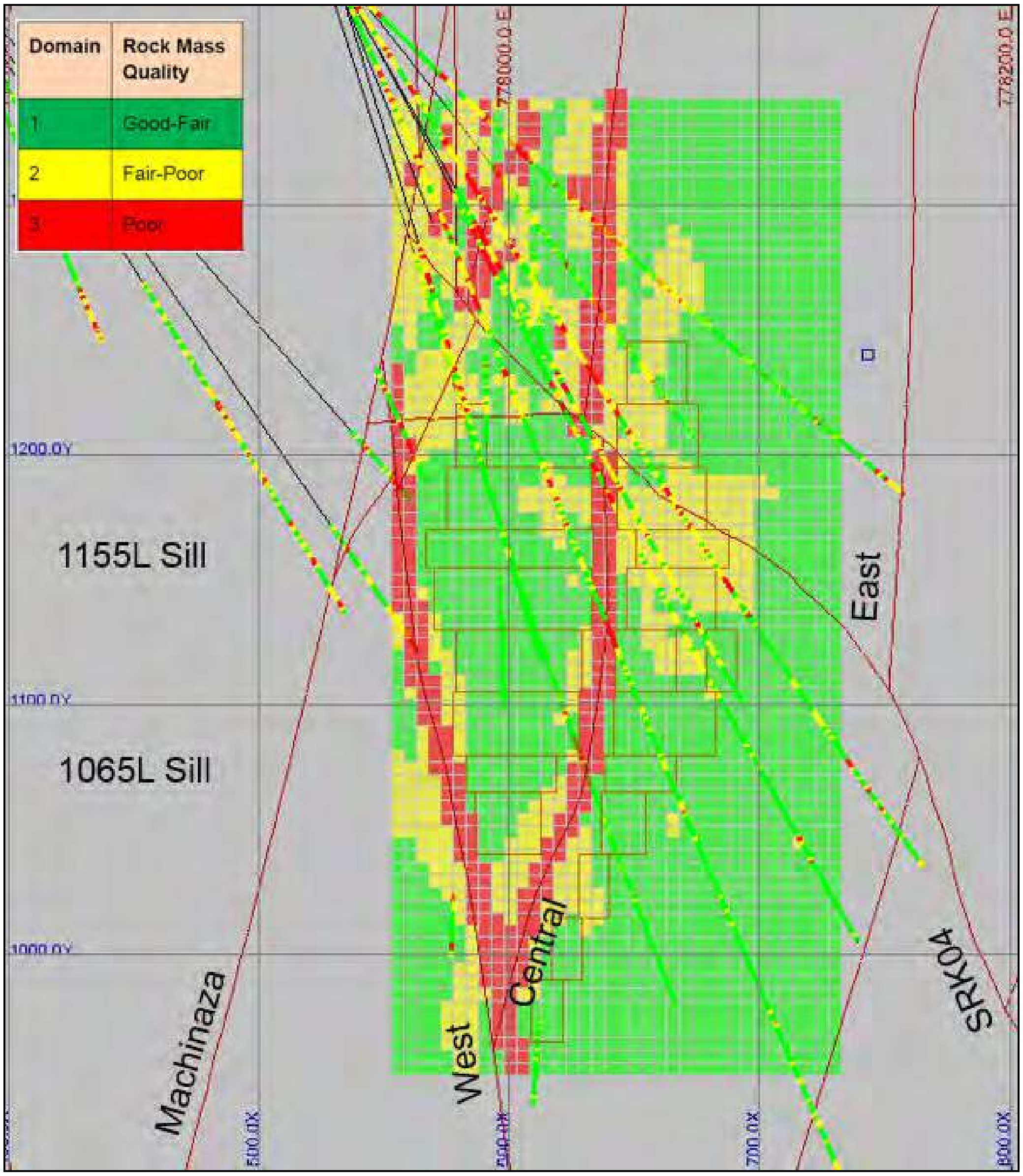
15.2 Assumed Mining Methods

The mining methods for FDN will be long-hole transverse stoping (TS) in Fair to Good ground, and drift-and-fill (D&F) stoping in Poor ground. Dilution was applied following the geotechnical recommendations. The shape optimizer from DESWIK software was used to determine practical mining shapes.

The deposit was divided into horizons that were classified both vertically and by mining method (Figure 15-2).

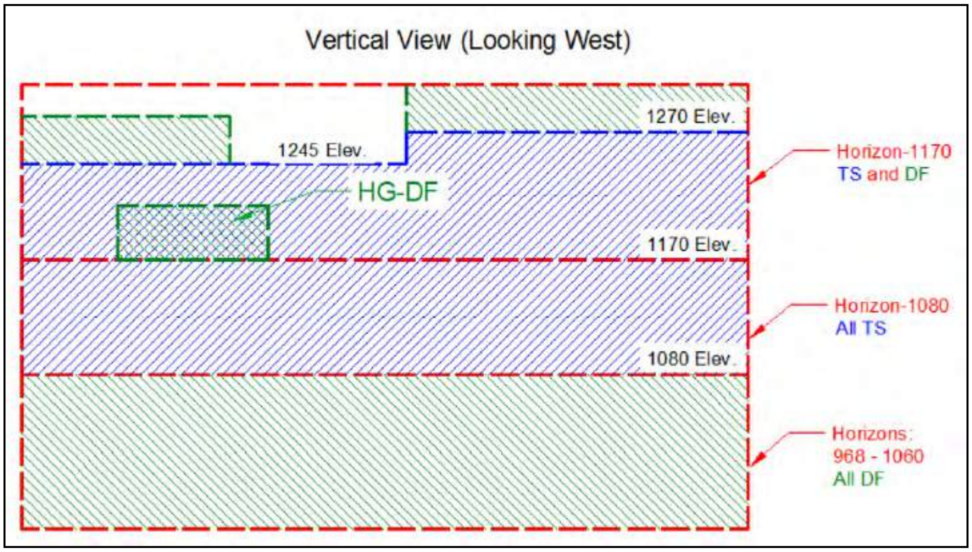
The top D&F horizons (DF-1245 and DF-1270) will be the crown pillar areas. The D&F south area at elevation 1170, is a high-grade zone within Poor domain rock quality. The upper part of horizon TS-1080 (stopes between elevation 1155L and 1170L) will be a sill pillar with 15 m high stopes. The bottom horizons (below 1080L) will be mined as D&F.

Figure 15-1: Representative Vertical Section through the Geotechnical Block Model for Stope Design



Note: Figure prepared by SRK, 2016.

Figure 15-2: Horizon Distribution



Note: Figure prepared by SRK, 2016.

15.2.1 Throughput Rate and Supporting Assumptions

The process plant feed will start in February 2020. Throughput was set at 3,320 t/d for three years, and then will reach 3,500 t/d. Additional detail is provided in Section 16.

15.2.2 Stope Sizing

The recommended dimensions for TS are 12 m wide x 20 m long x 25 m high. D&F was designed with vertical cuts of 4 m each. Additional detail is provided in Section 16.

15.2.3 Geotechnical and Hydrogeological Considerations

Geotechnical and hydrogeological considerations are discussed in Section 16.

15.2.4 Dilution and Mine Losses

TS Method Dilution

The dilution material for the primary stopes was estimated using the resource block model; dilution material for lateral stopes was assumed to be zero grade on one side and the grade from the resource block model on the other side. A summary of the dilution estimate is presented in Table 15-1.

Table 15-1: TS Grade Dilution Factors

Dilution	Regular TS		Sill Pillar TS	
	Starting Stope (Primary)	**Lateral Stope (Secondary)**	**Starting Stope (Primary)**	**Lateral Stope (Secondary)**
Ore	13.6%	7.3%	14.4%	7.7%
Waste	2.5%	7.3%	2.5%	7.7%
Total Operational Dilution	16.1%	14.6%	16.9%	15.4%
Grade Dilution (factor)	97.6%	93.2%	97.6%	92.9%

Note: Dilution percent is calculated as additional tonnage over insitu tonnage.

The total maximum dilution reaches to 16.9% (sill pillar starting stope); for scheduling and reporting purposes the waste dilution is applied (a maximum of 7.7% in sill pillar lateral stopes) so as not to duplicate tonnage because of the stope arrangement at FDN.

The grade dilution factor applied is a factor by which grades are adjusted because of dilution, in this case the waste reduces the grades because it adds no content for the following elements: gold, silver, mercury, lead, sulphur and antimony.

D&F Method Dilution

The D&F dilution estimate includes the primary, secondary and tertiary drifts. Results for estimates of D&F dilution are presented in Table 15-2. A grade dilution factor of 95.3% was used for D&F.

Mining Losses

For TS primary and secondary stopes, the geometry of the production ring blast results in shoulders that can only be partially blasted, resulting in losses. The load-haul-dump machine (LHD) will have difficulty mucking the stope corners and near the walls, particularly under remote control operation. This will result in an additional loss.

It is assumed that one ore truck per day will be misclassified as waste, resulting in an overall loss of 1.3%. Sometimes skins are left in the secondary stopes; an overall loss of 2.7% has been allowed for this type of loss.

Unplanned stope failures and safety issues were also considered to account for further losses. Higher values were estimated for secondary and sill pillar stopes.

Overall, in primary TS stopes, the total losses are estimated at 8.8%, resulting in a mining recovery factor of 91.2%. In secondary TS stopes, the total losses are estimated at 11.9%, resulting in a mining recovery factor of 88.1%.

Sill pillar mining recovery is assumed at 50%.

For D&F areas, the mining recovery was assumed as 100%.

Summary of Dilution-Mining Recovery & Grade Dilution

The final LOMP weighted-average dilution applied in the estimation (including TS, D&F and development) is 5.63%. This material is backfill material and is applied at zero grade.

The final LOMP weighted-average mining recovery applied to the estimate is 90.9%.

The grade dilution factor (the factor by which the drop in grade from insitu to diluted and mining recovered is reported) varies depending on the mining method and extraction sequence. The weighted-average grade dilution factor applied (the weighted average dilution of 5.63% and weighted average mining recovery of 90.9%) is 95.14%. This is estimated using the ratio of:

(Diluted & mining recovered)/(Insitu).

15.2.5 Cut-off Grades

Cut-off grades (COGs) are used to identify whether material is classified as ore (at or above the COG) or waste (below the COG). The COG is a function of operating costs, dilution, metal prices, royalties and process recoveries.

Cut-off grades were estimated for each mining method (TS and D&F) using the technical and financial information provided in Table 15-3 and Table 15-4. The cut-off grade data were calculated using metallurgical recoveries and other data that were fixed as of December 2015 for mine design and planning purposes. Some of these data have subsequently been revised, and the December 2015 data may differ from the final values used in the financial analysis in Section 22. Silver is not used as an input when calculating cut-off grades.

Two different COG have been used, the breakeven COG (BECOG) and the mill COG (MCOG). The BECOG is one of the key parameters needed for mine and stope design. The estimate of BECOG considers mining, processing, royalties and overhead operating costs. The BECOG calculation is shown in Table 15-5.

The MCOG is applied after the stopes and the accesses are defined, at this stage there could be some low grade material that has to be mined and hauled to surface. A decision has to be made whether to send this material to the process plant or to the waste dump. If the material has enough grade to pay for processing and other surface costs, it is sent to the processing plant (the mining cost is considered a sunk cost). The MCOG calculation is shown in Table 15-6.

The calculated COGs are listed in Table 15-7.

Table 15-2: D&F Grade Dilution Factors

Stope Type: Primary		Stope Type: Lateral (Secondary)		Stope Type: Lateral (Tertiary)	
Total Operational Dilution	11.0%	Total Operational Dilution	9.4%	Total Operational Dilution	9.9
Ore	9.0%	Ore	4.7%	Ore	4.9
Waste	2.0%	Waste	4.7%	Waste	9.9

Table 15-3: Operating Cost Detail by Mining Method

Area	TS (US$/t)	D&F (US$/t)	Comments
Mining cost	61.0	80.0	Preliminary mine cost NCL estimate, includes backfill
Process costs	34.5	34.5	Process plant, tailings transport to TSF, Ore Concentrate transport not included
Surface Infrastructure	25.8	25.8	Includes Hollín Borrow Pit, WTP, reclaim water, assay laboratory, surface infrastructure
G&A costs	15.5	15.5	Site G&A included but no other indirect costs
Total Operating Cost (US$/t)	**136.8**	**155.8**	
Dilution Factor	10%	10%	Preliminary waste dilution estimates
Diluted Cost (US$/t)	**150.5**	**171.4**	**Calculated**
Concentrate Transport & Treatment	6.7	6.7	Estimated by Lundin (see selling costs in Table 15-4)
Cost Inflation Factor	0.0	0.0	Was not applied
Total Cost (US$/t) Treated	**157.2**	**178.1**	

Note: Data current at December 2015.

Table 15-4: Financial Parameters

Item	Unit	Value
Gold Price	US$/oz	1,250
Payable	%	100
Gold Payable	US$/oz	1,250
Selling Cost	US$/oz	65.90
Royalty	US$/oz	71.1
Gold Process Recovery	%	93.9
Release Revenue	US$/oz	1,045

Table 15-5: Breakeven Cut-off Grade

$$\text{BECOG (g/t)} = \frac{\text{(Mining Cost + Process Cost + G\&A) x Dilution}}{\text{(Product Price - Selling Cost - Royalty) x Process Recovery}}$$

Table 15-6: Mill Cut-off Grade

$$\text{MCOG (g/t)} = \frac{\text{(Process Cost + G\&A) x Dilution}}{\text{(Product Price - Selling Cost - Royalty) x Process Recovery}}$$

Table 15-7: Mining Methods Cut-off Grade

COG		TS	D&F
BECOG	g/t	4.7	5.3
MCOG (1)	g/t	2.7	2.7

Note: (1): MCOG: The marginal cut-off grade. Mining costs are excluded in this calculation

A BECOG of 4.7 g/t Au was used for TS and an elevated BECOG of 6.8 g/t Au was used for D&F. A MCOG value of 2.7 g/t Au, excluding the mining costs, was used where production development was already built.

15.3 Mineral Reserves Statement

Mineral Reserves were classified using the definitions in CIM (2014). No Proven Reserves are reported. The Qualified Person for the Mineral Reserve estimate is Mr. Alejandro Sepúlveda, RM CMC, an employee of NCL.

Mineral Reserves have an effective date of 30 April, 2016.

Mineral Reserves are summarized in Table 15-8.

15.4 Factors that May Affect the Mineral Reserves

Factors that may affect the Mineral Reserves include:

- Long-term commodity price assumptions
- Long-term exchange rate assumptions
- Long-term consumables price assumptions.

Other factors that can affect the estimates include changes to the Mineral Resources input parameters, constraining stope designs, cut-off grade assumptions, geotechnical and hydrogeological factors, metallurgical and mining recovery assumptions, and the ability to control unplanned dilution.

Commencement of operations will require grant of the exploitation licence.

Table 15-8: Probable Mineral Reserves Statement

Material Source	Tonnes (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)
Transverse Long-Hole Stope	8,404	8.97	2,423	10.4	2,813
Drift & Fill	5,533	11.15	1,984	16.9	3,003
Development >4.7 g/t Au	1,158	9.70	361	11.6	434
Development >2.7 g/t Au	394	3.72	47	7.4	94
Total	***15,490***	***9.67***	***4,816***	***12.7***	***6,344***

1. The Qualified Person for the Mineral Reserve estimate is Mr. Alejandro Sepúlveda, RM CMC, an NCL employee.
2. Mineral Reserves have an effective date of 30 April 2016. All Mineral Reserves in this table are Probable Mineral Reserves. No Proven Mineral Reserves were estimated.
3. Mineral Reserves were estimated using a US$1,250/oz gold price. Mining cost assumptions for transverse stoping (TS) US$61.0/t; mining costs for drift-and-fill (D&F) stoping US$80/t. Other costs and factors common to both mining methods were process and other costs US$75.8/t, dilution factor 10%, concentrate transport and treatment charges of US$6.7/t. A royalty of US$71.1/oz/t Au and a gold metallurgical recovery of 93.9% was assumed.
4. Gold cut-off grades were 4.7 g/t for TS and 5.3 g/t (elevated to 6.8 g/t) for the D&F.
5. Silver was not used in the estimation of cut-off grades but is recovered and contributes to the revenue stream. The average silver metallurgical recovery is 81.6%. The silver price assumption was US$20/oz.
6. Tonnages are rounded to the nearest 1,000 t, gold grades are rounded to two decimal places, and silver grades are rounded to one decimal place. Tonnage and grade measurements are in metric units; contained gold and silver are reported as thousands of troy ounces.
7. Rounding as required by reporting guidelines may result in summation differences.

15.5 Comments on Section 15

Mineral Reserves have been estimated using the guidance in CIM (2003) and classified using the 2014 CIM Definition Standards.

There are no other known environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors that would materially affect the estimation of Mineral Reserves that are not discussed in this Report.

16.0 MINING METHODS

16.1 Overview

The following key considerations influenced the mine design:

- The Project is located in an environmentally-sensitive area. Although an open pit mining method or a caving method might be possible, the subsequent impacts were assessed not to be feasible. Hence, selective underground mining was considered for the 2016 FS
- The host rock for the deposit appears competent but the resource zone is less competent with a small portion in poor rock (less than 10%). Geomechanically, the rock mass quality varies from Poor to Fair (RMR range 40 to 55), with the intact rock strength averaging 60 MPa. The deposit is also relatively close to surface (within 140 m of surface in some locations)
- Given the variable conditions likely to be encountered, a range of methods and or support regimes was considered appropriate for FDN. The primary methods of extraction selected are TS in the better ground conditions and D&F in the more geotechnically-challenging areas
- Incorporation of backfill to reduce the risk of geotechnical failure and maximize extraction
- Consideration of dewatering requirements and proximity of the Machinaza River.

16.2 Geotechnical Considerations

The geotechnical data and resulting recommendations in this subsection were generated by SRK, and are based on work completed by SRK (2013, 2015–2016), Itasca Consulting Canada Inc. (2010–2011), and Golder Associates (2008).

16.2.1 Structural Geology

Faults

The faults present in the 2015–2016 structural model form a complex network of west–northwest- to northeast-trending, moderate dipping to sub-vertical faults that variably truncate and offset lithology and gold mineralization. Observations of core from the 2015 geotechnical drilling include numerous other faults that have not been modelled, either due to the narrow widths, lack of ability to correlate between adjacent drill holes, or the lack of significant offset of the lithological units or gold mineralization.

Faults generate the widest zones of gouge and breccia where they cross the Suárez Formation. In comparison, faults have well defined margins where they cross the Misahuallí Formation.

The West, Central, and portions of the East Fault are significant fault structures that represent a risk to the stability of an open stoping method and subsequently these areas are considered suitable only for a man-entry mining method where conditions can be well controlled such as D&F. Not all structures could be modelled, and the influence of the secondary and tertiary level structures are not well understood in some areas. Several East Fault-parallel secondary structures have been interpreted within the east footwall ramp area, and will need to be further defined during underground drilling programs.

Figure 16-1 shows the 1170L with mine infrastructure, modelled faults and interpreted structural domains.

Degradation Zones

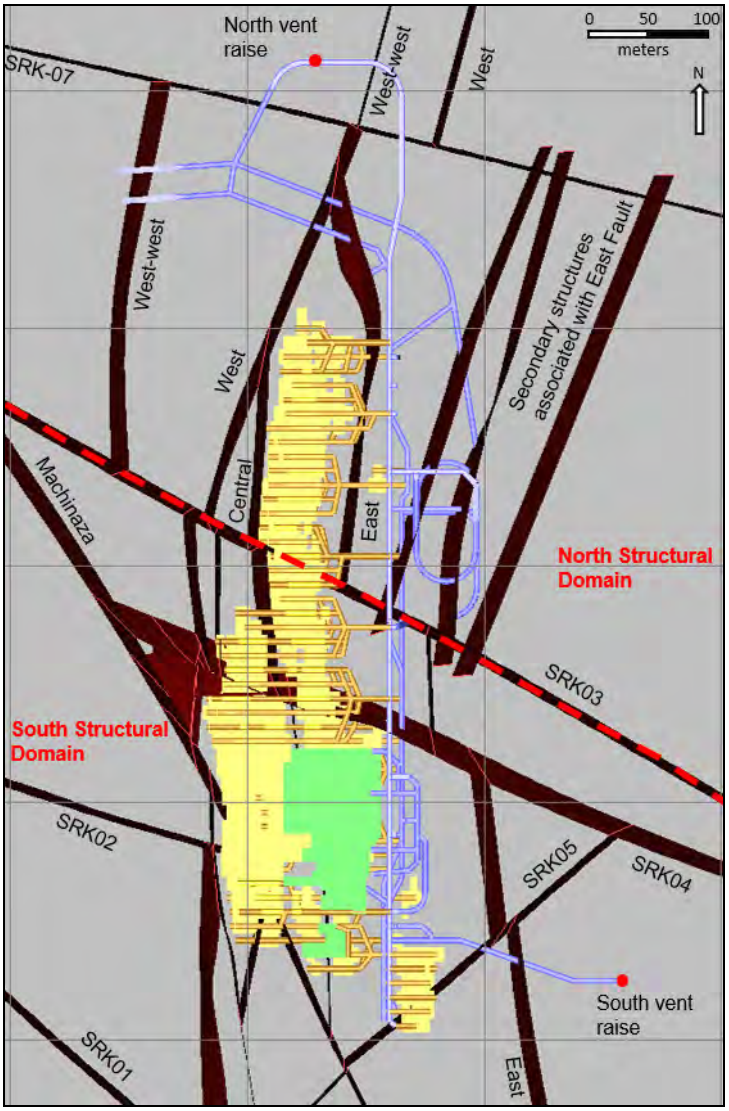
Time-dependent degradation of drill core has previously been identified at FDN. In general, degraded zones are contained within the Suárez Formation conglomerate and interbedded tuff beds. Degradation is strongest in intervals that are observed or interpreted as having contained disseminated pyrite. Contacts of the Suárez Formation conglomerate were used to extrapolate degradation zones beyond areas of graphically-logged intervals in order to construct moderate and high confidence degradation shells. Across the deposit, the West Fault produces significant degradation above and below the conglomerate and tuff strata, and the East Fault or parallel faults to the east appear to displace or bound the degradation zone.

Degradation of Suárez Formation conglomerate results in difficult mining conditions that can be mitigated through extraordinary ground support (full shotcrete lining and invert) which will be a high mining cost with slow advance rates. The mine layout has been optimized to avoid intersecting the Suárez Formation.

16.2.1 Insitu Stress

Stress measurements are not currently available for FDN. In the absence of this information, a stress regime based on SRK's evaluation of the structural geological setting and the World Stress Map have been used to provide a range of estimates. Uncertainty in the stress magnitude has been assessed through a K-ratio sensitivity analysis.

Figure 16-1: Plan View of Structural Model and Structural Domains (1170 Level)



Note: Figure prepared by SRK, 2016.

The ground stress is relatively low based on the shallow depth, and rock damage due to higher mining-induced stress concentrations is only anticipated in high extraction or sequence closure areas, and weaker rock mass areas. However, reduction in the mining stresses around excavations is likely to adversely affect the stability of large, open-span areas. Tensile failure and gravity-induced unravelling are foreseen as the main failure mechanisms. The pre-mining stress field should be further evaluated during the pre-construction and early construction phases, and the designs should be adjusted if required.

16.2.2 Geotechnical Assessment

The FDN deposit is in a structurally-complex, clay-altered environment, adjacent to a river. Rock mass conditions in the infrastructure and production areas vary from Poor to Fair quality (RMR 20 to 60) with the poorest conditions present within major structures that run longitudinally through and bound to the deposit. Outside of these fault areas, rock mass conditions are generally Fair (RMR 40 to 60; intact rock strength 50 to 70 MPa); however, localized zones of Poor ground potentially associated with secondary structures or locally elevated alteration intensity are present throughout the planned mining area.

A geotechnical domain model has been constructed that incorporates the 2015 structural model, degradation assessment, upgraded core photo re-logging, and 2015 geotechnical drilling data. The core photo re-logging has been used to build a geotechnical block model to represent the variability in ground conditions underground. Three domains have been developed that encompass Poor, Fair–Poor, and Good–Fair rock mass conditions.

The block model extents were designed to encompass the stoping and majority of infrastructure areas using search ellipsoids based on anisotropy extracted from the structural model. The objective of the block modelling work was to generate a tool to spatially assist with stope design (particularly stoping 'no-go' zones), with infrastructure, and development rates.

16.2.3 Excavation Design

Stability Assessments

Excavation stability assessments were completed using industry-accepted empirical relationships, with reference to analogue operational mines where possible. The rock mass conditions in the Poor to Fair and Good domains are considered suitable for open stoping mining methods. The ground conditions within the Poor domain (and crown pillar area) are considered suitable only for a man-entry method where conditions can be well controlled such as D&F.

SRK noted the following:

- The recommended open stope geometry is 12 m wide x 25 m high x 20 m long, mined in a transverse orientation (west to east) across the deposit perpendicular to the dominant structural orientation. Stability of the stope back is critical for maintaining stable mining conditions, and stope widths have been reduced to 12 m wide to provide better support coverage of the back from a single cross-cut. This is dictated by the range of conditions expected in the Fair–Poor domain. Smaller stopes imply less drill and blast damage, lower incidence of stope run-away, and a more predictable mine plan. SRK considers this to be a prudent approach until underground exposure indicates that this view is too conservative

- Based on the prevailing ground conditions in the Poor domain, drift and fill headings are recommended (4 m wide x 4 m high). These dimensions will ensure good quality backfill practices can be maintained through tight filling to manage the open spans, side wall stability, and ultimately stability of the mining area. The smaller spans will require lower levels of ground support to ensure that cycle times and productivity are maintained
- The stand-off distance for long-term critical excavations including ventilation shafts and the workshop is recommended to be 60 m from stoping. For permanent foot wall drives, a 30 m stand-off is recommended.

An end-slicing (primary–primary) sequence should be used to promote stability, maintain control of mining and reduce the potential for stope run-away. This is primarily dictated by the potential range of rock mass conditions in the Fair–Poor domain interspersed between the better Good–Fair domain and the Poor domain (especially in close proximity to fault areas). SRK considers that the best approach to manage risk in this environment is to plan a more conservative approach to the stope design and extraction sequence. The high-grade nature of the deposit means that recovery of ore is critical to maintaining the grade profile, and the stability and final recovery of secondary stopes in the variable rock mass could be very challenging.

Global Extraction Sequence

The guiding philosophy for stress management is to lead the initial stopes vertically as quickly as possible, maintaining a steep echelon formation parallel and perpendicular to the strike. Production cross-cuts should be driven with a 'just-in-time' approach, and retreated back as quickly as possible based on the limited pillar size between them. Stopes on the west side should be mined as early as possible to cut off east–west principal stresses. Where possible stoping should retreat towards solid abutments, and closure areas between mining faces should be limited.

Compromises have been made in the extraction sequence as a result of the need to balance grade and production profiles, extraction of wide orebody areas, and other geotechnical constraints. Ultimately several aspects of the sequence may not be geotechnically optimal, and additional design may be required.

Numerical Modelling

Numerical assessments using Map3D and Itasca codes FLAC3D and 3DEC have been completed to evaluate the extraction sequence, stope stability, stress migration, potential damage to infrastructure, and subsidence. Even though the stoping will be at relatively shallow depths (200 m to 500 m below ground surface), stope closure areas and adverse extraction geometries may generate elevated levels of stress that result in rock mass damage and possible poor excavation performance related to low rock mass strength. Early mining of hanging wall stopes will help to cut off the high horizontal stresses acting across the deposit.

Rock mass damage will be particularly prevalent in the excavation intervals located within the fault zones, adjacent to advancing stoping. These cross-cut intervals will need to be well supported during initial development and may need rehabilitation in the more critical closure areas.

In general, the reduction in the mining stresses around excavations is more likely to adversely affect the stability of long-hole open stopes as well as the areas immediately above the D&F mining areas. The failure mode in these areas is more likely to be tensile failure and gravity induced unravelling. Preventing this type of failure will require high levels of support.

3DEC and FLAC3D codes were specifically used to review the potential for movement along faults and the potential for surface subsidence. Subsidence caused by extraction could cause dilation or fracturing above the deposit and an increase in hydraulic conductivities and water inflows to the mine. Some level of dilation of fault and joint systems within the Suárez Formation can be expected as a result of mining. Under the current extraction sequence this is expected to be during the late stages of mining.

16.2.4 Portals and Ventilation Shafts

North Portal

Design of the north portal excavation and soft ground tunnelling was completed by Alan Auld Canada Ltd. (Alan Auld, a specialist in weak-ground tunnelling) under the guidance of SRK. Weak saprolite and saprock ground conditions at the north portal require that a shallow box-cut excavation is established to form a suitable face where tunnelling can occur. Specialized soft ground tunnelling techniques with full concrete lining and concrete floor will then be required to advance the tunnel for an approximate 70 m horizontal distance, to a point where conventional hard rock drill and blast tunnelling can begin.

Alan Auld's current design is considered suitable for the feasibility level. Additional work has been proposed to bring the design to construction level including site investigations, design and modelling work to be completed prior to the start of pre-construction.

Ventilation Shafts

Elysium Mining Limited (Elysium) completed appraisals of the planned north and south ventilation shaft site conditions.

Pilot hole drilling has not been completed for the raise locations, and the current scoping level appraisals considered nearby drill holes as a guide to the conditions in the area of the proposed excavations. At both the north and south shaft locations it is considered technically feasible to construct a raise bore with maximum diameter of 5 m with limited post-construction ground support. At the north shaft limited data exists for the upper 100 m portion of the shaft and it is likely that permanent sealing of the top 49 m will be necessary to eliminate groundwater ingress and to provide permanent long term support. Additional near-surface investigations are required to confirm ground conditions. Both locations require geotechnical investigation of the immediate 60 m shaft pillar area for the entire raise profile.

16.2.5 Backfill

Free-standing paste backfill walls are required in the transverse long-hole stoping areas. The recommended unconfined compressive strengths of 300 kPa for main stope filling, and 2,000 kPa for sill mats are based on previous modelling work completed by Itasca Consulting Canada Inc. (Itasca), benchmarked values, and SRK's experience in paste backfill at other mining operations. Sill mats will be required on 1080L and 1170L, with a planned thickness of 12 m.

For D&F areas, cemented rock fill (CRF) has been recommended with unconfined compressive strength between 3 MPa and 5 MPa (estimated 5% binder content; testing has not been completed). A high-quality, stiff CRF will allow tight filling to maintain stability of the open spans in poor ground conditions and critical crown pillar areas.

Additional information on the backfill considerations for the Project is included in Section 16.7.

16.2.6 Ground Support

Ground support design considers industry standard empirical guidelines and SRK's experience in variable ground conditions. The ground support philosophy for underground excavations is sprayed concrete lining (fibrecrete or shotcrete) with anchors installed through the concrete. Sprayed concrete was selected for overall simplicity and speed of application, longevity of surface support, and sealing of intrusive units that may potentially oxidize over the long term.

Enhanced ground support for poor ground areas includes the installation of spiling (pre-support), thicker shotcrete, reduced anchor spacing, and Swellex and split-set type anchors.

Long-standing temporary development, over-stressed cross-cuts, and cross-cuts in closure areas will require some level of rehabilitation. This has been estimated at 30% of cross cuts (in the Poor and Fair–Poor domains). A rehabilitation requirement of 5% for permanent development has been estimated based on the linear metres of development completed in the Poor domain.

16.3 Hydrogeological Considerations

The hydrogeological data and resulting recommendations in this subsection were generated by SRK, and are based on work completed by SRK (2013, 2015–2016), Itasca Consulting Canada Inc. (2010–2011), and Golder Associates (2008). Additional information on water management is included in Section 20.

16.3.1 Groundwater Conditions

Groundwater is expected to inflow into the underground mine from the fractured bedrock around the mine itself and from geological structures. The total groundwater inflow will not be large compared with many other mines around the world, and could be dealt with by in-mine pumping, but the combination of the water with poor ground conditions and the mining methods could have an influence on mining productivity. Rock within the mining area is potentially acid generating (PAG); hence, water that flows through the mine is assumed to need treatment before being discharged to the environment.

Data have been collected over three programs: Golder (2007–2008); Itasca (2010–2011); and SRK (2015). The hydrogeological dataset for the Project includes 99 hydraulic tests with 42 incorporating geological structures; water level data from 38 standpipes or vibrating wire piezometers; and 30 groundwater quality samples:

- The 99 hydraulic tests include short duration packer injection tests (less than one hour), short duration airlift tests (less than one hour to about 10 hours) and longer duration airlift tests (three days to five days). Most airlift tests had numerous observation points at various distance from the test hole. A large-scale, long-duration pumping test has not been completed for the Project (e.g. pump at 1,000 m^3/d for one month). The lack of large-scale test results has been addressed using the available data set and conservative assumptions
- Forty-two hydraulic tests were completed over intervals logged as incorporating a geological structure. Results suggest that, on average, tests incorporating geological structures are not much different than tests in other rock formations. Cross-hole test responses suggest some potential connectivity along the Machinaza and Central Faults
- Water level or pore pressure data are available from 38 standpipe or vibrating wire piezometers distributed across the site, and from 109 measurements inferred from

hydraulic testing data. Near the mine footprint, water levels from the Suárez Formation are relatively flat, suggesting relatively high hydraulic continuity. Compartmentalization is possible. Artesian conditions have been noted from exploration drill holes collared near the Machinaza River and drilled towards the east, underneath higher-elevation ground. Artesian pressures could be encountered during mining and would persist until the source of pressure has drained

- A total of 30 samples has been collected from the mine area, from wells installed in 2015, and from the Itasca three-day airlift test completed in 2010. In general, background groundwater quality is good. Water in the Suárez Formation may be corrosive over long periods of time (i.e. many years). Geochemical studies indicate that some lithologies are potentially acid-generating.

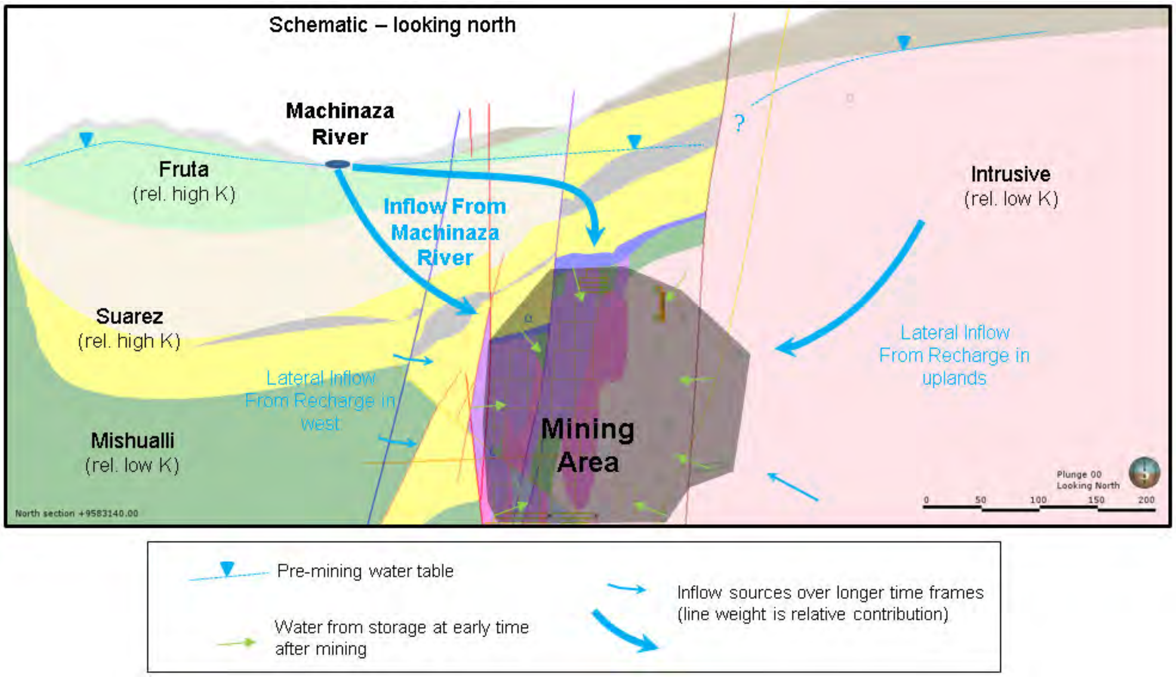
In general, the groundwater system is fracture-controlled with overprinting by geological structures. Groundwater is recharged at high elevations, flows generally transverse to topographic contours (i.e. downhill) towards lower elevation areas and the Machinaza River (Figure 16-2). Under pre-mining conditions, groundwater discharge occurs as springs and seeps in the slopes; as discharge to the Machinaza River or its tributaries; or flows towards the north, parallel to the Machinaza River, and exits the Project area.

The sources of mine inflow would be:

- Recharge from the Machinaza River (the underground mine reverses the hydraulic gradient toward to the mine due to dewatering and this factor is expected to play a significant role in controlling inflow since the mine is in close proximity to the river)
- Depletion of groundwater storage
- Capture of groundwater flow originating as recharge from precipitation, and which currently discharges in to the Machinaza River and its tributaries.

Groundwater inflows and dewatering well discharge rates were estimated using a groundwater numerical model constructed using MINEDW software (Azrag et al., 1998). Models were completed for multiple scenarios with different hydraulic conductivity distributions. Each model was generally calibrated to the average water level data. The modelled scenarios include the dewatering wells and galleries, but not drain holes. The different scenarios provide a reasonable range of potential conditions based on the data. Model results are considered conservative; it is expected that inflows will be less than predicted and pumping/dewatering capacities will be higher than required.

Figure 16-2: Hydrogeological Conceptual Model



Note: Figure prepared by SRK, 2016.

16.3.2 Dewatering and Mine Drainage

Groundwater inflow risks and potential effects will be managed in multiple ways, including cover and probe drilling, localized grouting, dewatering wells and underground drainage galleries. The selection of these methods was based on the results of a groundwater management trade-off study that compared costs to benefits in terms of pumping system requirements, water treatment plant sizing and overall water-related risks.

As mine development proceeds, the groundwater system will start to drain down, but since the geological units only have moderate hydraulic conductivity and flow will be fracture controlled, it is expected that drainage performance will be highly variable over different parts of the mine. The combination of dewatering wells and drainage galleries with drain holes provides flexibility and some degree of redundancy to reduce the risk of areas not being sufficiently dewatered prior to production mining.

Summary descriptions of the dewatering wells and drainage galleries are provided in Table 16-1.

Table 16-1: Dewatering Wells and Drainage Galleries

Item	Design Consideration	Comment
North Decline	Groundwater inflow is expected to be minimal. The North Decline will pass under the Machinaza River and is assumed that it will cross the Machinaza Fault. Cover drilling ahead of the decline will be completed well before the decline reaches the river. Under the river, cover grouting is not believed to be necessary, but will be confirmed by the cover drilling. Sumps will be located at multiple positions along the decline to keep water away from the face	During portal construction water will be collected in sumps and diverted away from working areas
Dewatering Wells	A fence of six dewatering wells will be positioned between the mine and the Machinaza River to draw down the water table within the mine area, to intercept water from the river before reaching the mine, and to reduce water entering the mine that will then require treatment. These wells target the thickest sections of the Suárez Formation and will terminate within the underlying Mishaullí Andesite. A seventh dewatering well will be located over the central part of the mine, within the crown pillar, to provide additional draw down capacity. This well will be completed within the Suárez Formation. Additional dewatering wells will be completed as needed	Dewatering wells will be drilled from the surface during the construction period to start dewatering of the mine area.
Drainage Galleries	Three north–south-oriented drainage galleries will be located at specific levels on the west side (hanging wall) of the mine. Drain holes will be drilled off the galleries to enhance drainage. Drain holes will typically be oriented west from the galleries towards the West Fault and Suárez Formation, though other orientations may be used as necessary.	The galleries will be 5 m x 5 m development drifts, providing access for drills to install passive drain holes and space for routing of water to the mine sumps

An underground groundwater management plan has been developed to provide a framework for monitoring and risk management. The plan will be updated prior to operations and reviewed annually during operations. Implementation of the planned monitoring will allow for ongoing assessment of actual site conditions. If conditions are better or worse than expected, the plan will provide the opportunity for water management activities to be adjusted.

16.4 Mining Method Selection

Supported and artificially supported mining methods are geomechanically applicable for the deposit. In general, unsupported or mining methods such as sub-level or blast hole mining, are more productive and have a lower unit cost than the more selective cut-and-fill methods, but they result in lower reserve recovery. The shape of the deposit (a massive block) tends to favour open stoping blast hole mining. To ensure stability of the stopes, it is important to select a stope size that is appropriate for the strength of the rock mass. Weaker rock masses support smaller excavations than stronger rock masses, but it is important to maximize the stope dimensions in order to optimize productivity and economics. Due to the nature of the development cycle in cut-and-fill mining, the quantity of ore per blast in these headings is far less than that of open stopes.

Other mining methods, such as stope-and-pillar and room-and-pillar where the excavations are left open after mining, were excluded from further consideration due to the thickness of the deposit, the weak rock mass and the potential failure of the surface crown pillar (presents a flooding risk). The stopes must be backfilled to ensure long-term stability of the mine, safe extraction and to maximize recovery of the Mineral Reserves.

The recommended mining method for the Fair to Good ground conditions is open stope blast hole stoping, with delayed fill. Typically, this mining method is applied either transverse or longitudinal to the deposit. Due to the width of the deposit, the transverse open blast hole stoping method (TS) is preferred. The recommended mining method for areas with Poor ground conditions is a drift and fill (D&F) method.

16.5 Consideration of Marginal Cut-off Grades and Dilution

This is discussed in Section 15.

16.6 Design Assumptions and Design Criteria

Long section and isometric views of the planned mine development layout are provided in Figure 16-3 and Figure 16-4 respectively.

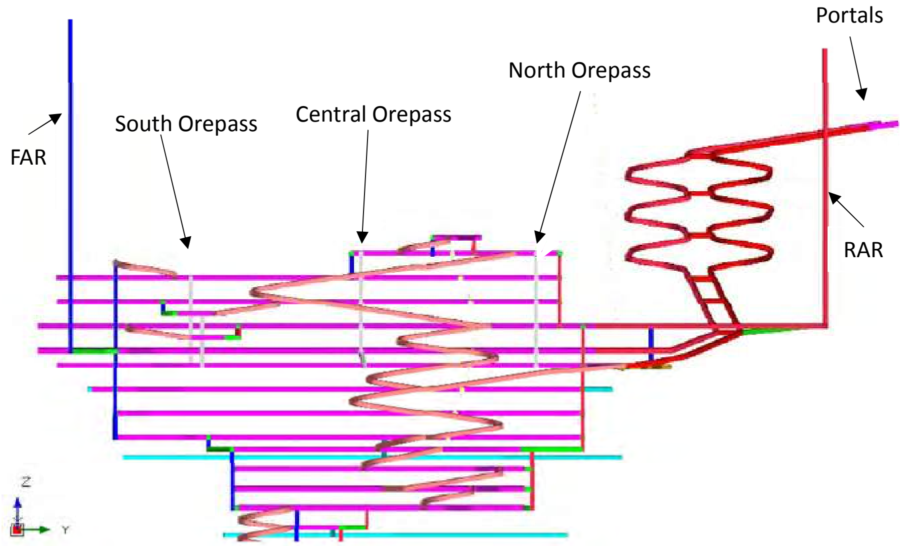
16.6.1 Stope Design Criteria

SRK defined the non-entry excavation design parameters for the stopes (Table 16-1). The orientation of the excavations has a limited impact on the stability of wedges (blocks of material between faults that would be unsupported during mining). The transverse mining orientation (east–west) is associated with fewer wedges but the calculated factor of safety is similar for both the transverse and longitudinal mining directions. In both scenarios the long axes of the wedges are aligned with the north–south direction which is also the trend of the dominant structural orientation.

Initiation points are especially important in the TS areas. Given the structural complexities at FDN, these were carefully selected in more competent zones to minimize the risk of uncontrolled caving.

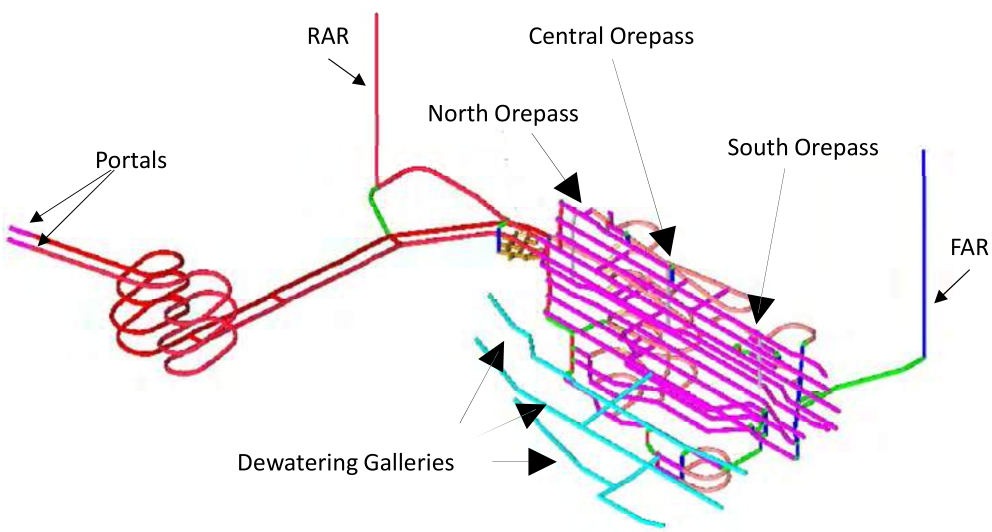
For excavations within the Poor ground a D&F method is recommended. Dimensions for this method are 4.0 m wide x 4.0 m high. For FDN, empirical design indicates that the stable man-entry span is 3 m to 5 m (RMR 30 to 40). Consideration has been given to extracting these zones in a geotechnically sound way and also to the interaction between other mining zones to minimize areas of stress concentration.

Figure 16-3: Schematic Long Section of Mine Development Layout Looking West



Note: Figure prepared by SRK, 2016.

Figure 16-4: Isometric View of Mine Development Layout (looking northeast)



Note: Figure prepared by SRK, 2016.

Table 16-2: Design Summary of Non-Entry Openings

Domain	Stope Face	Hydraulic Radius	Sub-Level Spacing (m wide)	Strike Span (m wide)	Dip Span (m high)
Fair–Poor	Back	3.8	25	12	20
	Vertical End	5.6	25	12	20
	Hanging Wall	4.1	25	12	20
	Footwall	4.1	25	12	20
Good–Fair	Back	3.8	25	12	20
	Vertical End	5.6	25	12	20
	Hanging Wall	4.1	25	12	20
	Footwall	4.1	25	12	20

16.6.2 Crown Pillar

The crown pillar is located at the highest level of the mine workings. There are often considerations related to the stability of this critical piece of infrastructure. These and other considerations related to the challenging geotechnical and hydrogeological environment mean that the crown pillar design must be included in the mine design criteria.

To mitigate the risks associated with the challenging conditions and recognizing the importance of stability in this zone, it was determined that D&F with CRF was the appropriate method of extraction for these areas.

16.6.3 Sill Pillar

In the sill pillar areas the following criteria should be applied:

- The final mining method will depend on the level of stress within the sill pillar and the prevailing rock mass conditions
- In the lower stress, stronger rock mass areas, long-hole stoping can likely be applied using standard dimensions
- In the higher stress, weaker rock mass areas, the method may need to be modified to shorter stope heights and in some areas reduced extraction may result
- For the 2016 FS, it was recommended that 50% recovery of the sill pillar areas be used.

16.6.4 TS Designs

SRK recommended TS where there is no Poor domain rock quality. The recommended dimensions for TS are: 12 m wide x 20 m long x 25 m high.

Stope Optimizer was configured for the following elevation limits: 1080L, 1105L, 1130L, 1155L, 1170L, 1195L, 1220L, 1245L and 1270L. With the primary–primary end slicing mining method and the TS layout, dilution was set to zero, to avoid duplicating mass when reporting the final results. Dilution was estimated and reported later. The optimization by SO was run along the X axis (east coordinate). The other two axes were fixed at 12 m (Y) and 25 m (Z), except for the sill pillar stopes (1155L) where the Z axis was set to 15 m.

16.6.5 D&F Designs

Outside the long-hole stoping area, where the COG allows the D&F mining method will be used. The dimensions provided in SRK's geotechnical recommendations, is a section 4 m wide x 4 m high. Hence D&F was designed with vertical cuts of 4 m each. The process for the final design of the D&F was as follows:

- SO run optimizing the X axis with fixed Y of 4 m and fixed Z of 4 m. This gives an idea of possible Mineral Reserves to be mined by D&F within the defined D&F horizons
- Detailed design of the drifts as development. This includes passing through waste areas
- Finally, the drift average Au grade was reported and filtered to include no more than 20% of waste (grade <5.3 g/t Au), resulting in a minimum drift grade of 6.8 g/t Au. This effectively high grades the D&F areas. The exclusion of lower-grade material can be both a risk and an opportunity which must be considered in later studies.

16.6.6 Crown and Sill Pillar Designs

The crown pillar is from the 1240 L (south area of the mine) to the 1270L (north area of the mine). Because of instability risk associated mainly with the rock quality, the mining method for these areas will be D&F.

A sill pillar was included between the TS horizons 1080L and 1170L at 1155L, which allows for earlier production. The mining method for this sill pillar will be TS with a stope height of 15 m (instead of 25 m in the regular stopes).

16.6.7 Mine Development Layout

The layout work was completed using CAD software, and imported to DESWIK software for scheduling.

Portals

Portals will be constructed using a cut and cover method for the first 34 m (chainage) from surface at a nominal gradient of 14%. They will be 20 m apart (25 m from axis). It is likely that these designs will be modified in later stages of design to leverage the ventilation opportunities that a twin portal/decline allows.

Twin Declines

The decline will be 5.0 m wide by 5.5 m high with an arched back. These dimensions will provide enough clearance for loaded 40 t capacity haulage trucks to move safely. The declines will be developed at a nominal gradient of -15%. Cross cuts will be established nominally each 250 m with some closer together depending on ground conditions and infrastructure requirements. The distance between the two declines is nominally 25 m (centre to centre).

The declines will use a spiral to gain depth to maximize the distance from the surface, so that a vertical distance of approximately 155 m below the Machinaza River can be obtained.

Mine Ramp and Level Access

The mine ramp dimensions are 5.0 m wide by 5.5 m high with an arched back. The mine ramp will be located central to and will be approximately 50 m offset from the main workings to the east of the deposit. The ramp configuration will be an elongated spiral, with straight sections and an inner turning radius of 50 m; the ramp will cross structures as perpendicular as possible while maintaining operational functionality. Where this is not possible, additional support will be required and the advance rate will be adjusted (related to rock quality domains).

The ramp configuration will enable haulage trucks to achieve higher average haul speeds and maintain safety standards. The ramp will be developed nominally at a -15% gradient. The ramp is designed to flatten out at each level access in order to provide easier turning conditions for mobile equipment.

Levels (Headers) and Haulage

The level dimensions are 5.0 m wide by 5.5 m high with an arched back. These are the same dimensions as the ramp and will provide sufficient clearance for the mobile equipment. Levels will be developed for the transverse stope horizons (TS-1170L and TS-1080L) from the mine ramp to the haulage drifts at a 25 m vertical spacing, except for the 1155L and 1170L which are at 15 m (sill pillar). For D&F areas and horizons, the vertical spacing will be 20 m. Levels will contain additional development for items such as escape way accesses, electrical substations and storage.

Levels will be developed to access the strike extents of the deposit and connect the development to the return air raise (RAR) in the north and fresh air raise (FAR) in the south in order to establish flow-through ventilation.

Ventilation Layout

The mine ventilation system will consist of two vertical raises; the FAR and RAR. These raises will be developed by raise borers from 170L (FAR/south) and 1195L (RAR/north) to surface. The raises will have a diameter of 5 m; the RAR will have an overall length of 290 m, and the FAR will have an overall length of 345 m. A 90 m/month vertical development rate was assumed.

The remaining sections of the mine ventilation system will consist of the two declines, the mine ramp and the internal raises connecting levels. Additional information is included in Section 16.8.

Transverse Stope Cross-Cuts

The cross-cuts for the TS will be 5.0 m wide by 5.0 m high, with a flat back. Slot (vertical) development is assumed to be four by four, slashed out to a 12 m stope width. Stope cross-cuts are required to access sill development from the haulage drifts, as well as connecting sill development within a given stope line separated by waste. The cross-cuts will generally be oriented in a west–east direction. In an effort to minimize the development requirements, primary and secondary cross-cuts will be developed using a pitchfork design. Sill development dimensions will be 5.0 m wide by 5.0 m high with a flat back. Development will be centrally located within a given stope. The top development in a stope will initially serve as the drill horizon for the stope below, and then as the mucking horizon for the stope above. The bottom development in a stope will serve as the mucking horizon for the stope above.

Drift and Fill Access (Pivots)

The D&F access (pivot) dimensions are 4.0 m wide by 4.0 m high with a flat back. Pivots will be required to access every lift of D&F, and they will be slashed when a D&F lift is developed and backfilled. They are developed from level drifts (headers) at ±15% gradient.

Development Quantities

Lateral (horizontal) development includes ramps, re-mucks, electrical substations, sumps, level accesses, ventilation accesses and haulages and will total about 180,640 m. Vertical metres include raise bores and drop raises, and totals an estimated 981 m.

Southern Exploration Decline

From 2011–2012, Kinross constructed a 5 m wide x 5.5 m high exploration decline, which reached a length of 600 m. This facility was assessed for inclusion in the 2016

FS mine plan, was dewatered, and inspected to check the rehabilitation needs and any requirements to continue with construction. However, the southern exploration decline is situated in an area that has strongly reactive pyrite and degradation issues. As a result of a review of the costs of covering this material, it was determined that use of the decline was cost-prohibitive. The southern exploration decline is not included in the current mine plan.

16.6.8 Construction Philosophy

Due to the specialized nature of the construction or development phase and the lack of skilled mining labour regionally, an experienced, qualified mining contractor will be required to develop the declines. Initially this will require standard construction equipment such as excavators and a shotcrete machine when establishing the portal. For the twin declines, conventional drill, blast, load, haul and bolter equipment and support equipment will be required to achieve high speed development. Once this critical development has been completed the contractor will demobilize. Other contractors will be required from time to time with specialized equipment such as a raise boring machine.

Contract mining will continue until the critical underground infrastructure has been constructed. The contractor will then demobilize. There will be a transition period as Owner mining equipment is introduced when access to additional ventilation and the mineralized zone is reached. Owner mining will eventually operate both development and production equipment.

The portals and main declines will be built by contractors. FDN will start to use its own equipment and operators in Year 2 of the mine construction period. It was assumed that mobile equipment for this will be leased. The remainder of the owner mining equipment is assumed to be purchased outright. This philosophy is incorporated into the operating and sustaining capital estimates in this report.

Underground infrastructure will be built with a combination of mine contractors and Owner mining.

16.6.9 Mine Production Plan

Criteria and assumptions used in preparing the production plan include:

- The production plan has been developed on a monthly basis from Year 2017 to Year 2022 and annually thereafter
- The mine will operate 360 d/a with five days allowed for delays due to weather conditions
- The plant is scheduled to operate 365 d/a

- Production will be a combination of TS and D&F methods
- The process plant is designed to treat 3,500 t/d.

TS Production

For scheduling purposes, stopes were assigned eight days for the slot construction, 800 t/d for stope productivity (excluding backfilling), and 23 days for backfilling.

D&F Production

The 2016 FS assumes that D&F backfill will use CRF. The cycle for CRF is driven by the ramming activity, which was estimated to have a productivity of 21 m/day. The curing time for the CRF is seven days.

Ore Passes

The materials handling for the upper levels (from 1155L up) has been designed with three ore passes. There will be two ore passes in the south designed to connect the 1155L with the 1245L and the 1208L. In the central and north areas, one ore pass has been designed for each area connecting the 1155L to the 1270L.

Sequence and Schedule

The mine development was sequenced and scheduled to meet the following criteria:

- Development will be done “just in time”, hence only the required development will be ready before a production unit is activated
- Production will start as soon as possible
- Ventilation circuits will be established as soon as possible
- The dewatering galleries will be developed six months prior to production areas.

The lateral development rarely reaches 500 m/month with the exception of later in the mine life. This is associated with higher production from D&F and less from TS later in the mine life. Excluding the D&F, the lateral development requirement averages between 133 m/month and 266 m/month.

The TS sequence was configured as follows:

- Production face advance from west to east (retreating from the west fault)
- Horizontally stopes should grow from a centre line (starting point, primary stope) to the north and south in an arrow fashion

- Vertically the same criteria will be followed, but each stope in a row (west to east direction) must finish its production cycle.

The macro-sequence was configured to meet the geotechnical recommendations for the location of the starting and closure points. For TS this is horizon-1170 and horizon-1080.

The D&F sequence was configured as follows:

- From the main infrastructure levels, a pivot drive will be developed to access each D&F slice vertically
- From the pivot a header drive (D&F-Header) will access the mineralized zone in the west boundary
- From the headers each 24 m (south–north direction) a primary drift will be developed
- After the primary drift is backfilled another drift will be developed to the north side and the same process will be repeated.

The maximum active stopes reaches 12 units in 2024 and 2025. The D&F areas have up to four drifts producing ore at any given time, it is assumed that for each six drifts there is one primary; and for each access pivot there are one or two areas available. The maximum number of active areas for D&F is 18 in 2024.

The production plan is summarized in Figure 16-5.

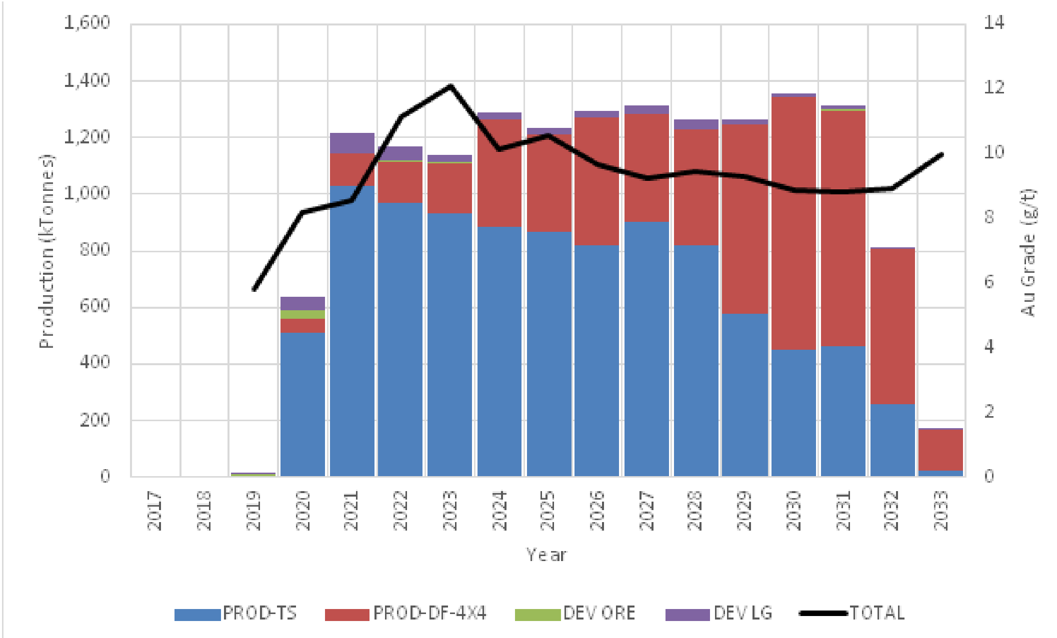
16.6.10 Plant Feed Plan

Calendar year dates noted in this section are provisional, and used for illustrative purposes only.

The plant feed was estimated from production from the mine as follows:

- Ore is taken from the mine to the three stockpiles:
  - High grade (HG): >7.0 g/t Au
  - Medium grade (MG): >4.7 g/t Au
  - Low grade (LG): >2.7 g/t Au
- Ore from the stockpiles feeds the plant. HG feed is the preferred feed and when the HG is exhausted, MG would be treated, and lastly the LG.

Figure 16-5: Mine Production Plan



Note: Figure prepared by NCL, 2016.

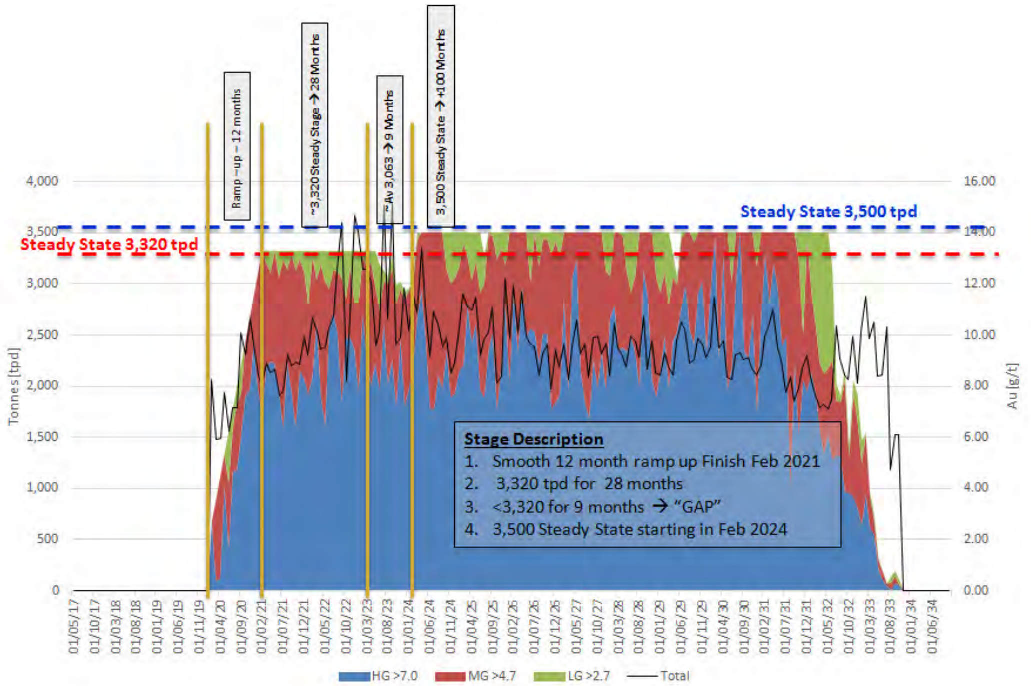
Plant ramp-up was assumed to be 12 months (this is considered to be reasonable and achievable). Because of mine production plan (stockpile generation) and the ramp-up, the plant feed will start in February 2020 (Figure 16-6). Throughput was set at 3,320 t/d for three years and then at 3,500 t/d.

There are some gaps in the planned flow of feed so as not to delay the plant start-up. The gaps in production can be seen in Figure 16-6 particularly towards the last four months of 2023 and first quarter of 2024 when the plant is scheduled to treat about

3,250 t/d. Other minor gaps can also be identified after February 2024 when the plant is operating at 3,500 t/d. Detailed mine planning and optimization needs to be done to minimize these gaps.

Table 16-3 is a summary of the projected annual production rates showing gold and silver production and the contaminant elements projected on an annual basis.

Figure 16-6: Mine to Plant Feed Production Plan



Note: Figure prepared by NCL, 2016.

Table 16-3: Plant Feed by Year with Penalty Elements

Year	Ore-DR (t)	Au-DR (g/t)	Ag-DR (g/t)	As-DR (ppm)	Hg-DR (ppm)	Pb-DR (ppm)	S-DR (%)	Sb-DR (ppm)
2020	594,944	8.47	9.2	319	1.1	166	1.9	30
2021	1,204,939	8.65	10.5	386	1.0	244	2.3	29
2022	1,211,800	10.89	13.6	538	1.1	326	3.1	40
2023	1,160,895	11.97	12.4	677	1.1	318	3.0	44
2024	1,250,327	10.36	12.6	526	1.3	327	2.8	38
2025	1,270,316	10.37	13.2	349	1.4	271	2.3	29
2026	1,277,500	9.76	12.9	329	1.5	243	2.1	29
2027	1,277,500	9.39	11.8	370	1.5	313	2.3	36
2028	1,281,000	9.39	12.6	376	1.7	361	2.1	41
2029	1,257,592	9.34	15.5	356	2.0	304	2.3	48
2030	1,277,500	9.21	14.2	306	1.4	197	2.0	34
2031	1,277,500	9.01	12.9	265	1.2	195	1.9	30
2032	977,300	8.07	11.9	286	1.3	200	2.0	27
2033	170,510	10.00	12.7	457	1.7	317	2.4	42
Total	***15,489,622***	***9.67***	***12.7***	***394***	***1.4***	***272***	***2.3***	***35***

Note: DR = diluted and recovered

16.7 Backfill

Paterson & Cooke (P&C) completed a study of the backfill systems for the FDN Project. The following backfill capacities and strength targets were set:

- The paste plant has been designed to cater for a nominal throughput of 70 m^3/h and will operate at an average utilization rate of approximately 60%
- Main pour target strength of 300 kPa after 14 days with a plug pour target strength of 434 kPa after three days
- The nominal design production rate of the CRF plant is 180 m^3/h
- CRF target strength of 3 MPa to 5 MPa after seven days.

16.7.1 Paste Fill Testwork Summary

P&C conducted the paste fill testwork using FDN tailings and Hollín Formation sandstone. The following summarizes the main outcomes of the testwork which influenced the design process:

- GFL tailings particle size distribution: very fine with 48% passing 20 µm and 30% passing 8 µm
- Predominant minerals in GFL tailings: quartz, muscovite. The presence of muscovite poses challenges to filtration, tailings transport and strength gain
- Thickening testwork: 64%m solids was achieved after four hours consolidation, up to 68%m solids was achieved after 24 hours consolidation
- Pressure filtration: the membrane filter press configuration using a partial form step only and no air blow drying step can achieve a maximum filtration rate of approximately 240 kg/h/m^2 (at a cake moisture of 20% and a feed solids mass concentration of 62%)
- Leaching potential: the acid rock drainage potential is low for the tailings and paste samples tested
- The most cost-effective paste recipe has an aggregate content of 50%.

The final paste recipe for the 2016 FS is as follows:

- Main pour: 50% aggregate, 6.58% binder content (Guapan Type IP binder), 77.8%m
- Plug pour: 50% aggregate, 12.4% binder content (Guapan Type IP binder), 77.8%m.

The final CRF recipe for the 2016 FS is based on previous testwork and has been benchmarked against similar projects. The recipe contains 5% binder.

16.7.2 Paste Plant and Paste Delivery

The paste plant will be located 1.3 km directly southeast of the process plant site, on the right bank of the Machinaza River, slightly north of the northern border of the deposit.

All tailings leaving the process plant will be thickened to about 55%m solids. When no paste fill is required underground, the entire tailings stream will be pumped to the TSF. When paste fill is scheduled for underground, approximately half of the tailings stream will be pumped 3.4 km to the paste plant for further dewatering. Excess process water will be pumped back from the paste plant to the process plant using a second pipeline.

The paste plant will be a batch-type backfill plant. The tailings received at the paste plant will be dewatered through a series of dewatering technologies. First the tailings will be fed to a 15 m diameter high rate thickener and from there the tailings will be fed to one of two vertical plate pressure filters to achieve a final moisture content of 20%. The filter cake will be stored temporarily, and when required will be batch-fed to the paste plant batch mixer.

Aggregate from the Hollín Borrow Pit will be crushed, screened and transported to the paste plant aggregate stockpile. When required, the aggregate will be conveyed to an aggregate weigh hopper and batch-fed to the paste plant batch mixer.

Dry binder will be stored in three 250 m^3 storage silos positioned in a horizontal configuration near surface elevation. Binder will be screw-fed from the silos, transported via a bucket elevator into the paste plant mixing building and discharged into a binder weigh hopper located above the batch mixer. Seven days of cement storage has been provided.

The batches of filter cake, tailings, aggregate and binder will be weighed independently before being fed to the paste plant twin shaft batch mixer. Slump control water will be added to the mix to control the slump which will be monitored via power draw on the mixer shafts. The paste recipe target yield stress will vary depending on the part of the underground mine requiring paste fill.

After being mixed for a predetermined period of time, and meeting the power draw specification, cemented paste fill will drop from the mixer into a paste hopper. A piston pump will pump paste fill on a continuous basis down a drill hole for deposition underground at the nominal design flow rate of 70 m^3/h.

The underground paste fill distribution system will include two surface to underground cased drill holes to supply paste to the underground workings. Once underground, the paste fill will travel through a network of pipelines to reach the location where the paste is needed. Two dedicated inter-level drill holes (one duty and one standby) will be drilled to connect all levels between 1270L and 1105L as a means of transferring paste fill to the various working levels. Paste fill will only be used to backfill bulk TS, meaning that paste fill is not envisaged to be required below 1080L, above 1270L, and in certain areas between 1170L and 1220L.

16.7.3 Cemented Rock Fill Plant

CRF is comprised of crushed rock, cement (binder) and trim water. A loader will dump crushed rock into aggregate bins at the CRF plant. An extracting conveyor will take the aggregate to the main incline conveyor which will transport the aggregate up to the mixer level, and discharge directly into the mixer. Binder from bulk silos will be screw fed into a binder dosing hopper where the product is weighed, then discharged into the mixer. A process water tank and pump will send water to the water dosing hopper where the water is weighed then discharged into the mixer. The CRF from the mixer will be discharged into a mine truck which will transport the CRF underground.

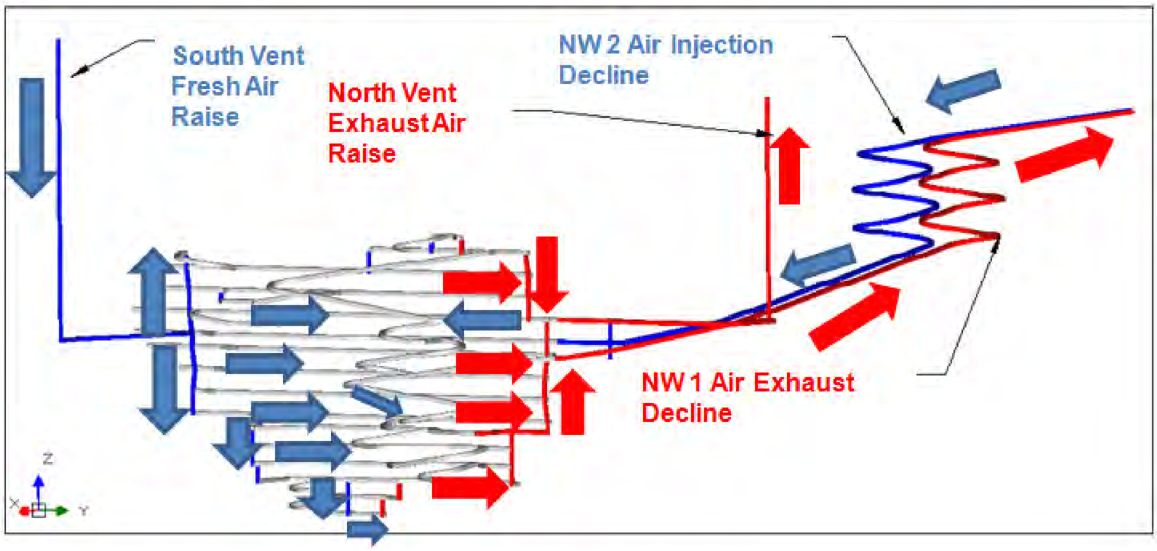
The CRF recipe will contain 5% Guapan Type IP binder, which has been based on industry norms, until future testwork can provide additional supporting information.

16.8 Ventilation

The main ventilation system is designed to accommodate the initial ore production rate and the ramp-up to the required tonnes per day. The ventilation system proposed is a mechanical exhaust ventilation system (pull) where fresh air will enter by suction through a 5.0 m wide x 5.5 m high injection ramp and a 5.0 m diameter main FAR. The contaminated air will be extracted from the underground mine through a 5.0 m wide x 5.5 m high exhaust and access ramp and a 5.0 m diameter main RAR with exhaust fans that will be installed in the airways (one fan unit per airway). Ventilation on the levels flows from the FAR to the RAR, as well as air coming from the ramp via the level accesses. Regulators located in the FAR and RAR will be used to control the air coming onto each level. Auxiliary fans will be used to push air into the sills for drilling, blasting and mucking activities.

A figure showing the schematic ventilation layout is included as Figure 16-7.

Figure 16-7: Ventilation Schematic



Note: Figure prepared by NCL, 2016.

During production, ventilation regulators placed on the intake and exhaust raises on each level will provide a balanced air flow on each production level. Auxiliary fans located in the haulage drifts will direct air off the haulage drift and into the individual stopes.

The ventilation system is balanced using regulators and fan volumes. Doors were not used because they hinder production in a truck haulage mine.

16.8.1 Initial Development

During the development of the declines, a blowing or forced air system will be used to ventilate the declines. This forced air system will be used with the cross cut connections established between the twin declines.

The main fan will be installed on surface at the portal collar of the NW2 decline and then, approximately every 250 m connections with the NW1 decline will be established. The NW1 decline will be used as the exhaust air route. The NW2 decline will be ventilated directly from the main ventilation fan using a booster to impel fresh air to the face with the exhaust air returning through the cross-cut connection and the NW2 decline.

This system will be used until the development of the north RAR system from 1195L to surface is complete.

16.8.2 Production Start

Stope production in the northern area will begin six months before the final ventilation circuit can be established connecting the south ventilation raise. A temporary ventilation circuit will be used including the intake through the two declines and exhausting the air through the RAR.

The total air required prior to full production was estimated on a month-by-month basis, with a peak of 178 m^3/s.

16.8.3 Full Production Requirement

Air Requirement

The ventilation air requirement dimensioning is based on Chilean and Ecuadorian regulations using the higher value of the following estimates:

- Number of people/diesel horse power (hp) equipment
- Explosives consumption.

The total air flow required for equipment and labour is 384 m^3/s. This value must be compared with the air flow required for ventilating the production blasting in order to determine the ventilation requirement. The production blasting considers that in Year 5 of production, there is 65% of the total production from stopes and 35% from D&F. Blasting indices of 0.77 kg/t and 1.78 kg/t were used. Two blasts per day were allowed for stopes and three blasts per day for D&F.

The higher value is the air requirement for people and diesel equipment (384 m^3/s). For the purposes of the ventilation system design, a value of 450 m^3/s was used; this value allows for approximately 17% in losses and leakage.

The intake side of the main ventilation system will deliver 450 m^3/s into the mine; 328 m^3/s will be delivered by the FAR and 122 m^3/s will be provided by the NW2 decline. A main extraction fan will be located near the bottom of the 5.0 m diameter raise bore hole (RAR) providing 488 m^3/s exhaust air flow, a secondary fan, in parallel, will be located in the NW1 decline exhausting 62 m^3/s of air to surface. Variable speed drives (VSD) will be used to control the fan outputs and allow for a phased approach to the mine ventilation requirements, as well as energy management.

Production Level Ventilation

Once a level has access to both the intake and exhaust raises, level ventilation will be controlled via plank-type regulators on the RAR and FAR accesses. The regulators will be adjusted to provide the air volume required for operating equipment on the level. Based on the air volume requirements per level, the regulators will be adjusted to supply approximately 75 m^3/s of air.

Auxiliary Ventilation

Up to three 30 hp axial fans are allotted for each active production level for auxiliary ventilation. Based on the stope sequencing, these auxiliary fans ventilate the active stopes and will not all be operating at the same time. Auxiliary fans will be hung in the haulage drift, forcing air into the accesses. Exhaust air will exit these accesses, will be pulled into the haulage-level air stream, and exhausted off the level through the exhaust raise.

The maintenance service facility that will be located off the ramp on 1170L will be ventilated via a raise connecting to the 1195L main access level to the RAR.

16.9 Underground Infrastructure Facilities and Services

16.9.1 Shops and Warehouses

It is proposed to keep material handling as simple as possible, relying on mobile equipment for transport instead of permanent infrastructure and facilities. Minimal storage will be developed underground. Much of the supplies such as ground support, cables can be stored on surface in covered areas or sheds and brought underground as needed. Underground storage areas will be located strategically and will service the daily/weekly operational requirements.

Haul trucks will be repaired in a surface maintenance facility. LHDs, drills, explosive carriers and scissor trucks will be repaired/maintained underground in a service facility located on 1170L or driven/hauled to the surface shop for major work. The underground service facility will contain two service bays with a wash bay located nearby. There will be a small warehouse and office. Fuel and oils will be located close by and contained in a satellite station (SatStat) arrangement. A mobile repair and servicing (fuel and oil) truck will be required to service underground equipment not brought to surface.

16.9.2 Automation and Communications

The radio communication system is based on laying leaky cable feeder antenna through the main tunnels and access ramps at various levels to ensure coverage in the areas of greatest traffic of people and vehicles.

A fibre-optic network will provide a communication highway for control and data management systems inside the mine.

16.9.3 Fuel Supply and Storage

The total estimated daily underground fuel consumption for diesel mobile equipment is about 13,500 L. Most of the mobile equipment, trucks and LHDs, and vehicles parked on surface will be fuelled from the surface facility. The rest of the fleet will be fuelled by the fuel/service vehicles or at the underground service facility.

16.9.4 Compressed Air

The air compressor system will consist of two compressors in operation and one on standby. An air accumulator will be store compressed air to regulate the air pressure. Compressed air will be used by in shotcrete operation, jacklegs, pumps, the explosives charger, refuge station, and garage, as well as for miscellaneous purposes.

The main pipe will run from the centralized air unit (located on the surface close to the portals) to the mining ramps. A total of about 4,000 m of pipe will be required for the main branch. Secondary pipes feeding the equipment in the drifts will be fed by 75 mm diameter pipe and a total of approximately 12,500 m will be required for the life of mine.

16.9.5 Dewatering

The dewatering system has been designed based on a hydrogeology study completed by SRK. All the water flow generated in the mine (infiltrated, industrial and paste fill water) will be managed in a single dewatering system. The system assumes that water flows running on ramps, declines and drifts is collected by gravity in a sump on each production level. Where gravity flow is not possible, a sump pump will be used to conduct water to the sump.

During the initial stage of development (first three years of pre-production) a temporary dewatering system will be installed; this will be replaced by a definitive dewatering system once the declines are finished. The temporary system will consist of one drainage water sump every 40 vertical m in the declines and ramps. These sumps will receive the water flow from the ramp and will operate as a decanter, promoting solids sedimentation in the sump. Each sump will be equipped with a sump pump which transports water to the next higher sump and so on, cascading the water up until it reaches the portal and is discharged to the surface water treatment plant.

The final dewatering system will be operative from Year 4 (ore production start) until the mine closes. This system includes a main pump station located at 1130L, which will pump water to the water treatment plant on surface (approximately 1410L), and three auxiliary pump stations that will feed the main station. The auxiliary pump stations are located on 1090L, 1030L and 970L. This system is designed to conduct the water flow to the main station at 1130L and then pump it to the surface.

16.9.6 Process Water

The underground process water system has been designed to deliver water for drilling and other equipment via an underground distribution network.

During the construction period, a temporary supply system will be used, distributed by a pipe through the ramp. Once the ramp arrives at 1170L, the permanent system will be used with a pipe descending by the ventilation raise.

16.9.7 Power

The estimated power demand for the mine was determined on the basis of the preliminary mechanical equipment requirements. This includes mining, dewatering, ventilation and low voltage miscellaneous loads. The resulting power demand for the underground loads is estimated to be 5.3 MW.

During the development of the declines, power will come from diesel generators located near the portal area.

In February 2020 permanent power from the grid will be available and underground electrical distribution will then be via two 13.8 kV double ended feeders from a surface substation adjacent to the portals, another surface substation located close to the RAR.

16.10 Blasting and Explosives

Underground magazines for explosives and detonators will be located on a selected level underground and will store approximately 3 days inventory. A more substantial surface magazine will be utilized in addition to this underground facility. A typical underground magazine bay will be 34 m wide x 5.0 m high x 35 m long. The bulk of the explosives used will be bulk emulsion, some cartridge explosives will also be used. The detonator magazine will be close to the explosive magazine. Auxiliary ventilation will be installed as required.

16.11 Mining Equipment

16.11.1 Equipment Requirements

Lundin Gold plans to Owner-operate. Mine operations will use the same equipment for development for TS and for D&F. Drilling, support, loading and hauling equipment are the same for both methods. Different equipment is required for loading for production because TS is 5 m wide x 5 m high and D&F is only 4 m wide x 4 m high.

Although specific equipment manufacturers and equipment models are described in this Report, no final equipment selection has been made. The equipment listed is indicative of the type and size of the equipment planned and was used as the basis for the capital cost estimate.

Contractor equipment requirements are not included in the equipment lists that follow.

The equipment for each lateral development crew will be primarily made up of a jumbo and bolter. The development drilling equipment fleet has been selected to primarily develop lateral and ramp excavations 5.0 m wide by 5.5 m high, and 5.0 m wide by 5.0 m high in the top and bottom stope cross cuts on a routine basis. Vertical development will be done using raise bore drills and vertical crater retreat (VCR) methodology. In general, raise boring will be contracted out to specialists who will use their own equipment and personnel.

TS production work will be done using radial jumbos for blast holes; drilling in D&F will use frontal jumbos shared with the development work. Loading (mucking) will be done using LHDs and low profile trucks will be used for hauling. Production crews will share emulsion loading vehicles for blasting of the stopes.

The following material-handling equipment is proposed:

- Loading and transportation with 12 yd^3 LHDs between TS and stockpiles in the headings of the production levels of the transverse stopes
- D&F stope loading to the stockpiles will be performed using 10 yd^3 LHDs
- From the stockpiles, direct load to truck with 12 yd^3 LHDs
- Transportation to the process plant on the surface using 45 t trucks.

The proposed mucking/haulage fleet has been primarily selected to accommodate excavations 5.0 m wide by 5.5 m high. For the purposes of the 2016 FS, the recommended loader and truck combination consists of a Caterpillar R2900G 12 yd^3 LHD and the Caterpillar AD-45 haulage truck (this combination provides adequate performance). For waste haulage and CRF haulage, the same trucks will be used. The haulage trucks will be equipped with ejector boxes to facilitate dumping CRF into stopes and for re-mucking. The capital cost estimate was prepared using the cost for this type of truck.

For D&F (with 4.0 m wide x 4.0 m high drifts) the primary loading equipment will be a Caterpillar R1700G 10 yd^3 LHD.

The production drilling fleet selected for this study to drill TS nominally 12 m wide by 20 m long by 25 m high consists of the Sandvik DL411 (top hammer) for production drilling and the Maclean MEM 928 Scissor Lift for cable bolting. Drifting and slashing in the D&F areas will be done using the same equipment used for development drilling (frontal jumbo, Sandvik DD421). Support equipment will be the same as that used for TS stoping and development.

Explosive handling equipment will be required in the development and production processes. For this study, the primary explosive handling equipment will consist of the Maclean EC3 Emulsion Carrier for lateral development and bulk stope blasting. The use of emulsion explosives will help to reduce the amount of nitrates/nitrites that may be dispersed into the drainage water system. It is assumed that ground will generally be wet and hence ANFO could be ineffective.

In addition to the development and production equipment, other equipment will be required to support mine operations. The type and quantity of equipment were estimated considering the following:

- Number of stopes in production; the maximum active stopes at a given time
- Overall requirements for a modern standard operation.

16.11.2 Equipment Performance

The number of jumbos was estimated as a function of metres developed (ramps, accesses and pivots). Estimates were based on the production and development plans. An advance rate was estimated for each type of activity and then used to estimate the number of units.

For the stope rings, the production drilling will be done using a production jumbo for TS. For D&F and level developments for the stopes it is planned to use frontal jumbos.

Taking into account the width of the drifts, two different types of loading equipment have been selected. The loading equipment requirement was estimated based on the mine plan. This equipment is allocated to material loading in the production stopes, mucking in the development galleries and loading trucks.

Hauling activities include ore and waste. Ore will be hauled from ore passes or loading stations to the stockpiles at the process plant, to be located about 500 m from the portal. The average hauling distance will be 2.67 km. Waste will be hauled from the mine to the waste stockpile, which will be located 2 km from the mine portal. The average waste transport distance is projected to be 3.0 km. CRF will be hauled an average distance of 3.0 km. The distances for each origin to destination will vary from period to period. Hauling performances have been estimated on a period basis, depending on distances.

For CRF transport it was assumed that the truck used for extracting ore or waste will haul CRF into the mine. Additional time for washing and other activities has been added to this cycle.

16.11.3 Equipment Numbers

In order to calculate the number of pieces of equipment per fleet, the performances estimated were used in conjunction with the requirements defined in the mine and preparation plans. Availabilities ranging from 70% to 80% have been used in the estimates, as per industry databases.

A maximum of four 10 yd^3 LHDs, four 12 yd^3 LHDs and nine 45 t trucks will be required for production and development (Table 16-4). Support equipment requirements are included in Table 16-5.

16.12 Comments on Section 16

The QP notes:

- The Project is located in an environmentally sensitive area, and open pit or block cave mining operations are not considered to be the optimal approach
- The host rock for the deposit appears competent, but the resource zone is less competent with a small portion in poor rock; the deposit is also relatively close to surface and is in close proximity to the Machinaza River
- Groundwater inflow will not be large compared with many other mines around the world, and could be dealt with by in-mine pumping, but the combination of the water with poor ground conditions and the mining methods could have an influence on mining productivity
- Two underground mining methods are proposed, TS and D&F. TS will be employed in areas of reasonable ground; D&F will be used in poor ground conditions areas. A crown pillar and a sill pillar have been included in the design
- Access will be via twin declines from surface
- The mine ventilation system will consist of two vertical raises; the FAR and RAR; ventilation design assumes a mechanical exhaust ventilation system (pull) where fresh air will enter by suction
- The mine plan assumes use of conventional equipment, which will be Owner-operated.

Table 16-4: Owner Main Equipment Fleet

Equipment	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033
LHD 10 yd^3	-	1	2	2	3	3	3	3	3	3	3	3	3	3	3
LHD 12yd^3	2	4	4	4	4	4	4	4	4	4	4	4	3	3	2
Truck 45t	2	7	7	7	7	8	8	8	8	8	8	10	10	6	2
Frontal Jumbo	2	4	4	3	3	3	3	3	3	3	4	5	5	3	1
Rammer Jammer	-	-	2	4	4	4	4	4	4	4	4	5	5	5	3
Radial Jumbo	-	2	3	3	3	3	3	3	3	3	3	3	3	3	1
Support Jumbo	2	5	6	5	4	4	4	4	4	4	4	5	5	3	1
Development Explosive Loader	2	2	3	3	3	3	3	3	3	3	4	6	6	6	1
Production Explosive Loader	-	2	3	3	3	3	3	3	3	3	3	3	3	3	1

Table 16-5: Support Equipment Fleet

Equipment Make and Model	Service Life (years)	Units Purchased
MacLean Scissor Lift	5	2
Marcotte Crew Vehicle	3	4
Rescue Vehicle	10	1
Caterpillar FEL	5	1
Atlas Shotcrete Sprayer/Hauler	5	2
Atlas Shotcrete Transmixer	5	2
Jacklegs	2	10
Atlas Scaler	5	5
MacLean BT-3 Boom Truck	5	3
Telehandler	5	2
Sandvik DE 140 UG Diamond Drill	5	2
Sandvik LH307 Utility LHD	5	1
Kubota tractors	5	7
MacLean Mobile Rock Breaker	5	3
Caterpillar Dozer	5	1
Caterpillar Grader	5	1
MacLean Fuel/Lube Truck	5	2

17.0 RECOVERY METHODS

17.1 Introduction

The circuit design size is derived from data and design criteria provided by Lundin Gold, Amec Foster Wheeler, NCL, KCB, P&C, vendor data, testwork and regulatory/permitting requirements. These data and criteria provided the basis for the calculations and design criteria derived from the mass balance.

The crushing and grinding circuit design is based on the throughput, ore hardness, clay content and moisture content. The inclusion of the SAG–ball mill, pebble crusher (SABC) circuit is due to the requirement for clay and moisture handling. The grinding circuit design considers the target primary grind size for the flotation circuit which is based on the testwork. The sizing of the crushing area was simulated using Bruno; the sizing of the SAG and ball mill are based on first-principle calculations.

The gravity circuit design is based on the large variability in free gold and vendor information for sizing and configuration.

The design of the flotation circuit is based on testwork results with standard flotation design configurations. Results from the MET1 program provided the basis for mass pull and recoveries, and historical laboratory tests provided the basis for retention times.

The carbon-handling circuits sizing is based on the carbon stripping batch size and frequency, gold grades of material processed, expected dissolution rates and maximum expected loading onto carbon. The gold recovery rate determined the sizing of the electro-winning and smelting circuits to produce the doré. The cyanide destruction system sizing is based on standard design for such a system incorporating the CIL tailings design flow rate and indicated residual cyanide content from testwork.

The carbon preparation and regeneration systems are based on estimates of carbon losses and the carbon stripping cycle. The reagent systems are sized to cover the indicated requirements per testwork or guidelines for similar batch circuits using cold acid wash and the modified Zadra process for hot stripping of gold and silver from activated carbon.

For other equipment the peak production rate was used for specific equipment sizing. Equipment for which sizing is mainly influenced by the volumetric flow rate are sized to cover a design flow at 115% of the instantaneous flow, except in the case of batch systems where the peak requirements are based on the batching frequency.

There are four water types defined for the process including process, treated, gland and domestic. Process water will be reclaimed from the tailings storage facility and overflow from thickeners. Process water will generally supply the grinding, flotation, leach and cyanide destruction areas where a specific quality is not required. Treated water will be supplied from the water treatment plant. Treated water will generally supply the adsorption, desorption and regeneration (ADR), intensive leach reactor (ILR) and reagent areas where the requirement for the quality of water is higher. Gland water will be supplied from treated water. Domestic water will be supplied from the domestic water plant and will be used for process plant safety showers, the camp and the mine.

17.2 Process Flow Sheet

The FDN process plant feed will need to accommodate recovery of gold in the following forms:

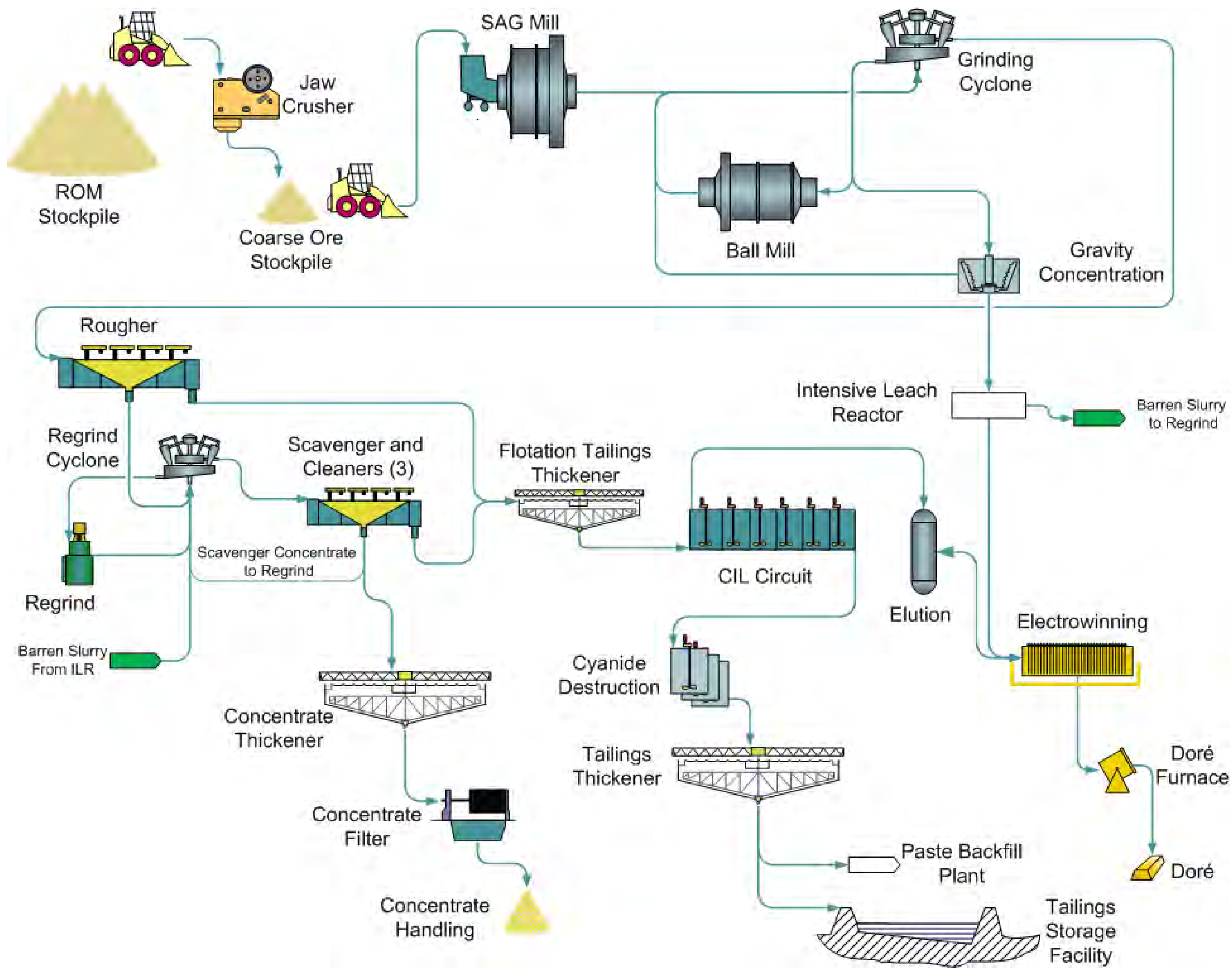
- Fine free gold
- Coarse free gold
- Gold contained in sulphides (refractory)
- Gold contained in other forms (e.g. silicates).

The GFL flowsheet (Figure 17-1) was selected for the Project because of the nature of the gold in the plant feed. The up-front gravity circuit is essential to recover the coarse free gold and small amounts of fine free gold. The gravity circuit is essential to reduce spikes in coarse gold content in the feed, ensuring that the flotation feed grade stays relatively uniform. The flotation circuit is capable of recovering the gold associated in sulphides (pyrite). The flotation circuit is able to reduce spikes in sulphide gold grade and provide a consistent feed to the CIL circuit. Typically, CIL circuits function best on a uniform feed, this can be provided by the combined gravity and flotation circuits.

Run-of-mine ore (ROM) will be transported to ROM stockpiles. A front end loader will load a bin through a grizzly which will feed the jaw crusher via a vibrating grizzly. The crushed ore will be transferred to a discharge conveyor followed by the stockpile feed conveyor which transfers it to the coarse ore stockpile.

Feed will be reclaimed from the stockpile via a front end loader; transferred to an apron feeder and conveyed to feed the primary SAG mill. Oversize from the SAG mill discharge will be recycled back to the SAG feed. The SAG circuit product will be fed to a cyclone cluster which is in closed circuit with the gravity concentrators and ball mill. Oversize from the gravity concentrator feed screen will be fed into the ball mill discharge which is pumped to the cyclone feed. Undersize will feed the gravity concentrators. Gravity concentrate will report to the intensive leach reactor (ILR) and the gravity concentrator tailings will return to the cyclone feed.

Figure 17-1: Simplified Process Flowsheet



Note: Figure courtesy Lundin Gold, 2016.

The ILR will produce pregnant solution which will be directed to electro-winning cells to produce a gold–silver precipitate. After washing, the barren slurry will report to the flotation regrind circuit.

The overflow from the grinding cyclone will report to the flotation circuit. The flotation circuit will consist of three stages of flotation, and regrind. Rougher and scavenger concentrate combined with ILR barren slurry will be directed to a regrind mill in closed circuit with a cyclone cluster. Final concentrate from the third cleaning stage of the flotation circuit will be thickened, filtered and bagged as product. Overflow from the concentrate thickener will be recycled to the process water tank.

Flotation tailings will be thickened and then report to the leach circuit while the thickener overflow will be recycled to the process water tank. The slurry will continue through pH conditioning before reporting to a series of CIL tanks where the slurry is leached with cyanide. Discharge from the leach train will report to cyanide destruction.

The loaded carbon generated from the CIL tanks will be pumped to the carbon elution and regeneration circuit. Once gold has been eluted, the carbon will be sent to regeneration. After quenching and screening to remove small particles, the reactivated carbon will be reintroduced to the CIL circuit.

Gold eluate will be sent to electro-winning cells using stainless steel cathodes to produce a gold-silver sludge. This is combined with sludge from the separate ILR electro-winning cell, filtered and dried. It is then mixed with fluxes and smelted to produce gold-silver doré.

Slurry discharged from the CIL tanks will report to cyanide destruction. A two-stage Inco SO_2/air process will be employed with the addition of lime. Sulphur dioxide will be provided as sodium metabisulphite. Slurry discharged from cyanide destruction will report to the tailings thickener. Underflow from the thickener will be sent to the tailings storage facility (TSF) or to the paste backfill plant. Overflow from the thickener will be recycled back to the process water tank.

17.3 Plant Design

Plant design criteria are summarized in Table 17-1.

Table 17-1: Plant Design Criteria

Design Criteria	Unit	Value
Operating time	d/a	365
Operating time	h/d	24
Nominal daily production	t/d	3,500
Ore Grade		
Gold (average)	g Au/t	9.7
Silver (average)	g Ag/t	12.7
Gold (minimum)	g Au/t	4.7
Silver (minimum)	g Ag/t	7.1
Gold (maximum)	g Au/t	15.1
Silver (maximum)	g Ag/t	17.5
Gravity Stage	% Au	19.0
	% Ag	12.0
Flotation Stage	% Au	79.3
	% Ag	75.7
CIL Stage	% Au	50.6
	% Ag	14.1
Overall	% Au	91.7
	% Ag	81.6
Feed Physical Properties		
Moisture	%	4 to 8
Specific Gravity	-	2.77
Bulk Density	t/m^3	1.6
Clay Content	%	5 to 8
Area Utilization		
Crushing	%	70
Grinding	%	90
Flotation	%	90
Leach	%	92
Cyanide Destruction	%	92
Crushed Ore Storage		
Storage Capacity	t	3,500
Grinding		
Feed Size F80	mm	125
SAG Mill Product Size P80	µm	2,756
JK Parameter Axb (85^{th} percentile)	—	41.6
Pebble Circulating Load	% of Fresh Feed	21

Design Criteria	Unit	Value
Ball Mill Product Size P80	µm	60
Bond Work Index	kWh/t	23.5
Cyclone Feed Density	% Solids	58
Cyclone Underflow Density	% Solids	75
Cyclone Overflow Density	% Solids	35
Gravity Concentrator		
Number of Concentrators	-	1 Operating, 1 Spare
Feed Source	-	Cyclone Cluster Underflow
Gravity Throughput	% of Cyclone U/F	33
Concentrating Cycle Time	min	30 to 40
Mass Pull	%	0.1
Destination of Flow	-	Intensive Leach Reactor
Flotation		
Overall Stage Recovery		
Au	%	79.3
Ag	%	75.7
Mass Pull	%	4.16
Carbon In Leach		
Feed	-	Flotation Tailings
Feed Density	% Solids	40
Overall Stage Recovery		
Au	%	50.6
Ag	%	14.1
Loaded Carbon Grade - Au	g/t Carbon	650
Loaded Carbon Grade - Ag	g/t Carbon	734
Carbon Movement	t/d	9.0
Carbon		
Carbon Type	-	Coconut Shell
Carbon Size	mesh	6 x 12
Dry Carbon Density	t/m^3	0.5
Soaked Carbon Density	t/m^3	1.02
pH	-	10.5 to 11
Oxidant	-	Air
Lime Addition (CaO based)	kg/t CIL Feed	0.39
Cyanide Addition Rate	kg/t CIL Feed	0.44
Cyanide Consumption	kg/t CIL Feed	0.05
Carbon Concentration	g/L	12
pH Conditioning		
Role	—	pH Modification

Design Criteria	Unit	Value
Residence Time	h	2
Number of Tanks	quantity	1
Leach Tanks		
Residence Time	h	12
Number of Tanks	quantity	6
Agitator Type	—	Axial turbines
Aeration Gas	—	Air
Aeration Rate	$Nm^3/h/m^3$	0.6
Cyanide Destruction		
Type	—	INCO
Oxidant	—	SO_2/Air
pH	—	8.5 to 9
Residence Time	h	2.5
Number of Tanks	quantity	2
Residual Cyanide	ppm	<1
Discharge pH	—	10 to 10.5
Carbon Treatment		
Number of Strips	quantity/d	1
Nominal Carbon Bed Volume	m^3	18
Carbon Movement	t/d	9.0
Carbon Reactivation Kiln		
Carbon Feed Moisture	%	4
Type	—	Horizontal, Rotary, Electrical
Availability	%	85
Kiln Capacity	kg C/h	500
Operating Temperature	ºC	550-650
Intensive Leach Reactor		
Number of Cycles per Day	quantity	1
Capacity ILR Module	t/batch	3.0
Recovery		
Au	%	95.0
Ag	%	90.0
Refining		
CIL Electro-winning		
Number of Units	quantity	2
ILR Electro-winning		
Number of Units	quantity	1
Mercury Retort		
Capacity	kg	500

Design Criteria	Unit	Value
Sludge Mercury Content	t/year	0.86 - 1.82
Induction Furnace		
Feed Precipitate	kg/week	630
Batches Per Week	batch/week	1
Capacity	kg of mix	750
Flux Addition	kg/kg ppt.	1
Tailings Thickener		
Unit Area (Continuous Rate)	m^2/t/d	0.20
Flocculant Addition	g/t Thickener Feed	30
Thickener Underflow		
Density	% Solids	55
Solids SG	—	2.72

17.4 Energy, Water, and Process Materials Requirements

17.4.1 Reagents

The following reagents will be required:

- Primary collector (xanthate)
- Secondary collector (AP208)
- Frother
- Carboxy methyl cellulose (CMC)
- Lime
- Sodium cyanide
- Sodium hydroxide (NaOH)
- Hydrochloric acid (HCl)
- Sodium metabisulphite (SMBS)
- Anti-scalant
- Flocculant.

Reagent mixing will be done in a designated area within the process plant. The design of this area includes features such as section bunding with dedicated sump pumps for individual reagents, segregated ventilation, and dust and fume control around reagents with potential for dust or fume release. The layout and general arrangement of the reagent area accounts for the need to prevent contact of incompatible reagent types. In general, reagent unloading, hopper loading and mixing will be carried out manually.

17.4.2 Air

Two low pressure air blowers will supply the process air needed for all the flotation tank cells. One of these will be operational; the other will be on standby.

Two air blowers will supply the process air needed for all the CIL and cyanide destruction tanks. One of these will normally be operating; the other will be on standby.

Compressed air for plant distribution will be provided by the plant air compressor via the plant air receiver.

Compressed air for instrument use will be provided by the instrument air compressor; and also this can be supplied by the plant air compressor if required. Instrument air will be dried by the instrument air dryer to remove moisture and distributed via the instrument air receiver.

The truck shop air compressor will supply compressed air to the truck shop. The truck shop air will be dried by the truck shop air dryer and distributed via the truck shop air receiver.

Compressed air for the primary crushing area will be supplied by the primary crushing air compressor and distributed via the primary crushing air receiver.

17.4.3 Water

The bulk of the water requirements for the process plant will be met by process water. Process water will be recycled from the thickener overflows and from the TSF as required. Water can also be reclaimed from the plant area sedimentation ponds, if necessary.

The process water tank of total volume 1,050 m^3 will also be used to provide fire water and thus must have a minimum water volume of 500 m^3 at all times. Treated water will be obtained from the water treatment plant and will be primarily utilized in the ADR, ILR and refinery circuits, as well as for gland water due to water quality requirements. Treated water will also generally be used for reagent dilution. Treated water will used for the grinding mill cooling systems and in the mine.

Gland seal water will be supplied from the treated water tank. Gland seal water is supplied to the plant in addition to the paste backfill feed pumps and tailings storage feed pumps.

The domestic water tank will supply the camp, mine and process plant. Domestic water requirements in the plant include the plant safety showers and bathrooms.

Domestic water will not be drinkable (potable); drinking water will be supplied as bottled water.

17.5 Production Summary

A production summary for the proposed LOM is provided in Table 17-2.

The concentrate production rate is expected to be 160 t/d at a feed rate of 3,320 t/d and 140 t/d at a feed rate of 3,500 t/d. During the MET1 and MET4 metallurgical testwork programs the final concentrates were analyzed using standard AAS, 32 ICP analysis and isotope evaluation for radioactivity. The results presented in Table 17-3 provide a guide to the predicted concentrate quality; the actual quality could vary from month to month based on ore variability, mine planning and sequencing as well as the geometallurgy.

The total gold expected to be produced as doré varies from 90 koz to 145 koz per year during steady state and is 1,323 koz during the LOM. The doré is expected to contain above 98% precious metals with the remainder made up of base metals and impurities. The precious metals portion is expected to contain approximately 70% gold and 30% silver.

17.6 Comments on Section 17

The QP notes:

- The flowsheet design is conventional, and assumes a gravity–float–leach approach
- The grinding circuit is designed at the 85th percentile
- The flotation circuit is capable of recovering the gold associated in sulphides (pyrite). The flotation circuit is able to reduce spikes in sulphide gold grade and provide a consistent feed to the CIL circuit. Typically, CIL circuits function best on a uniform feed, this can be provided by the combined gravity and the flotation circuits
- The concentrate production rate is expected to be 160 t/d at a feed rate of 3,320 t/d and 140 t/d at a feed rate of 3,500 t/d. The concentrate quality could vary from month to month based on ore variability, mine planning and sequencing as well as the geometallurgy
- The total gold expected to be produced as doré is 1,323 koz during the LOM. The precious metals portion is expected to contain approximately 70% gold and 30% silver.

Table 17-2: Production Summary

Description	Unit	Value
Plant Feed		
Length of Production	years	14
Life of Mine Feed	tonne	15,489,622
Average Feed Gold Grade	g/t	9.67
Average Feed Silver Grade	g/t	12.74
Concentrate		
LOM Concentrate Production	tonne	644,804
Average LOM Concentrate Gold Grade	g/t	149.3
LOM Concentrate Contained Fine Gold	tonne	96.3
LOM Average Concentrate Silver Grade	g/t	203.7
Doré		
LOM Gold in Doré Production	M oz.	1.323
LOM Silver in Doré Production	M oz.	0.953
LOM Doré Production	M oz.	2.276
Total Production		
LOM Gold Production (concentrate and doré)	M oz.	4.418
LOM Silver Production (concentrate and doré)	M oz.	5.177
Gold in concentrate/gold in doré	—	70%/30%
Silver in concentrate/silver in doré	—	82%/18%

Table 17-3: Concentrate Quality

Element	Unit	Min	Max
Al	%	0.68	0.89
As	%	0.34	1.1
Ba	ppm	17	60
Be	ppm	<0.2	<0.5
Bi	ppm	8	20
Ca	%	0.09	0.35
Cd	ppm	38	76
Co	ppm	83	106
Cr	ppm	48	196
Cu	%	0.17	0.38
Fe	%	35.2	42.9
Ga	ppm	30	34
K	%	0.3	0.48
La	ppm	4	5

Element	Unit	Min	Max
Li	ppm	8	14
Mg	%	0.04	0.09
Mg	ppm	380	865
Mn	ppm	224	1224
Mo	ppm	<50	80
Na	%	0.01	0.06
Nb	ppm	<1	<1
Ni	ppm	24	89
P	%	<0.03	0.04
Pb	ppm	2,484	6400
S	%	38.7	53.4
Sb	ppm	482	1019
Sc	ppm	1	2
Se	ppm	<12	<12
Sn	ppm	<5	<30
Sr	ppm	9.7	18
Te	ppm	22	59
Ti	%	0.11	0.14
TL	ppm	11	<30
V	ppm	20	54
W	ppm	67	160
Y	ppm	2.9	6
Zr	ppm	13	15.6
Zn	%	0.5	1.291
Hg	ppm	11	14.4

18.0 PROJECT INFRASTRUCTURE

18.1 Introduction

An infrastructure layout plan for the Project is included as Figure 18-1.

18.2 Road and Logistics

18.2.1 Access

The planned route to access the FDN site is by the Troncal Amazonica road to Los Encuentros and from this point to the Project site by a new main access road, a section that will be public road near the El Pindal village, and another new section of road through the Ecuadorean jungle that will be private (Figure 18-2). After km 15, the new 22 km long road will be used exclusively for access to the Project site. The access control facility will be located at km 15.

The Project will have limited temporary construction roads that will need to be decommissioned after construction. The current temporary road from the exploration camp to the Project will be upgraded in terms of width, gradients and road surface. This road will remain open for exploration activities during operations.

Roads within the mine site will be gravel, consisting of crushed base course and select granular sub-base, and will be engineered for light and/or heavy vehicles as applicable.

18.2.2 Cargo Transport Route

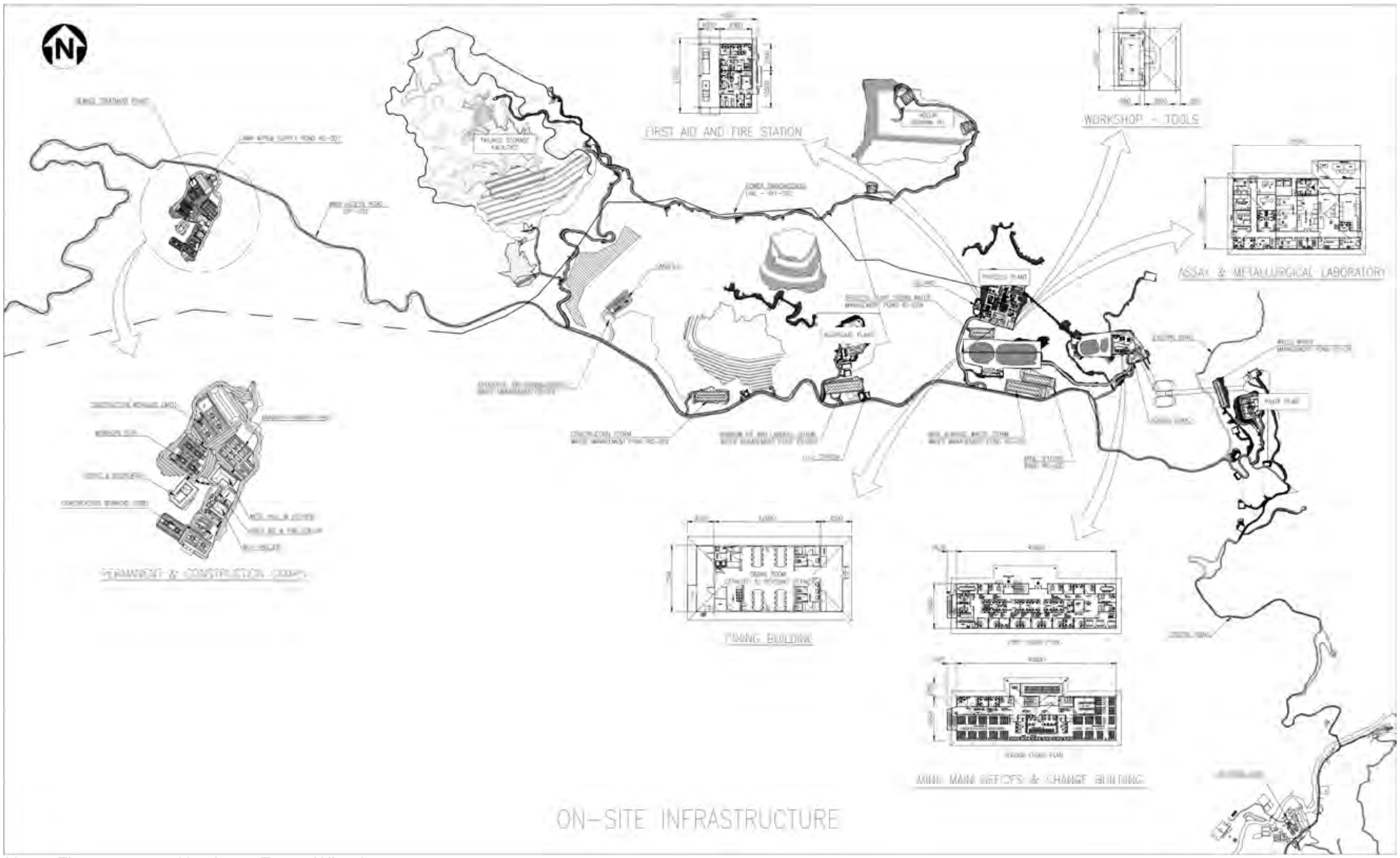
The main port for international cargo arrival will be Guayaquil. The route from the port is: Guayaquil Port, Virgen de Fatima, Cañar, Azogues, Paute, Mendez, Bomboiza, Los Encuentros. This route does not present significant height or weight restrictions, except for those which can be managed by changing poles, cables, and rails. The cargo weight limitation for all bridges is 40 t approximately. The Port of Bolivar may be used as an alternative.

18.2.3 Air Services

The nearest airport for commercial national flights is Loja Airport, located 157 km from the FDN site. The Project will include a helipad near the process plant area for doré transportation.

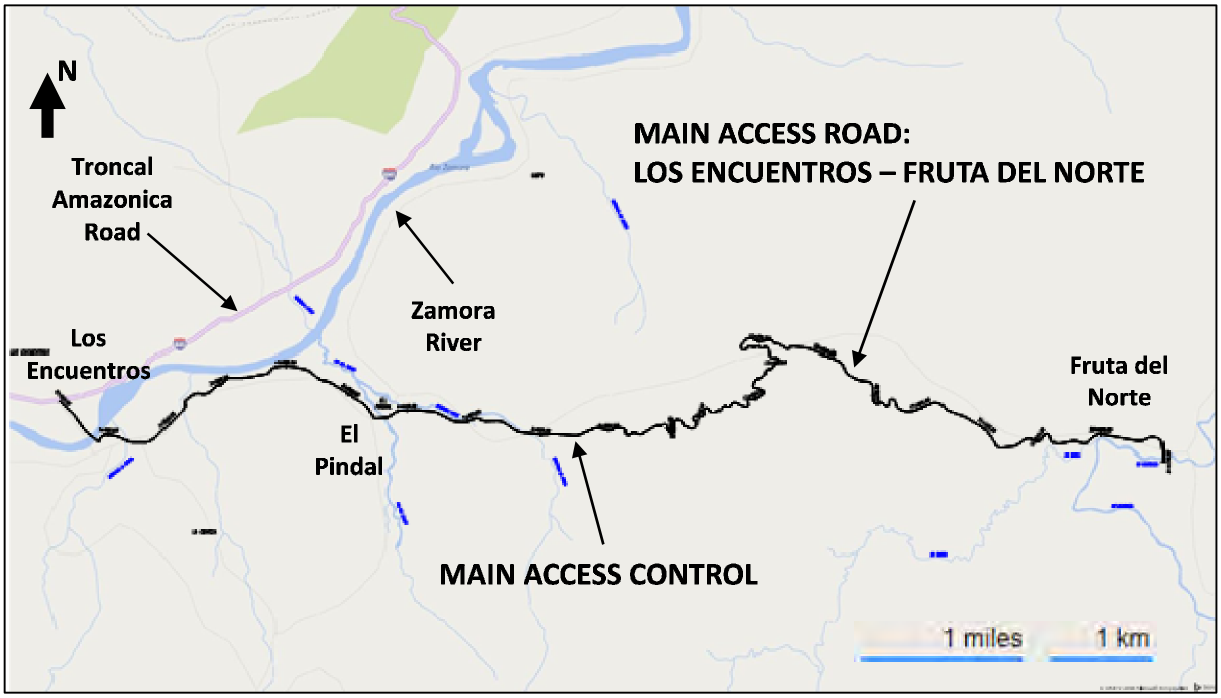
A space has been reserved at the Project site for an airstrip, but this will not be constructed as part of the initial installations.

Figure 18-1:On-site Infrastructure Layout Plan



Note: Figure prepared by Amec Foster Wheeler, 2016.

Figure 18-2: Access Route



Note: Figure courtesy Lundin Gold, 2016

18.3 Waste Storage Facilities

Waste storage facilities are discussed in Section 20.

18.4 Stockpiles

Stockpiles are discussed in Section 20.

18.5 Tailings Storage Facility

The tailings storage facility (TSF) is discussed in Section 20.

18.6 Hollín Borrow Pit

The Hollín Borrow Pit is discussed in Section 20.

18.7 Water Management

Water management plans and the basis for these plans are provided in Section 20.

18.8 On-site Infrastructure

On-site non-process services such as the camp, greenhouse, sewage treatment plant and mobile equipment will support the operation (refer to Figure 18-1). There will be fresh water, domestic water and process water systems and a fire detection and protection system.

The utilities and services include compressed air supply and distribution, process control system, closed circuit television (CCTV) system, supervisory control and data acquisition (SCADA) system, waste management systems and fuel storage and distribution.

Mobile equipment for maintenance, operations services and transportation includes tractors and loaders for stockpile re-handling, mobile cranes, buses and utility vehicles.

18.8.1 Site Buildings

The buildings designed for the site include the following:

- Truckshop (two truck bays, one welding bay, tire shop and truck wash). Mobile equipment (equipment and trucks) will be serviced in the mine surface maintenance area, which includes offices and kitchenette
- Mine office building with change house. This will be a two-storey building with offices, meeting rooms and kitchenette on the first floor. The change house will be on the second floor, with an independent access stairway
- Process plant office. This will be a single-storey building with offices for process plant personnel and a kitchenette
- Electrical room and control room
- Workshop
- Canteen
- First aid station and fire station
- Laboratory
- Camp facilities.

18.8.2 Camps and Accommodation

The permanent camp facilities will be located close to the main access control to the Project site at an altitude below 1,500 masl. Camp services will include kitchen and mess hall, toilet and shower units for workers, recreation and sports areas, fire station and first aid post, sewage system, water treatment plant, bus shelter, parking, laundry and lockers, warehouse and maintenance, administration office, and guard house.

The camp will have capacity for a peak of approximately 830 people and will include services facilities, and food preparation facilities. The camp facilities will be constructed early in Project construction in order to accommodate construction personnel as the construction manpower grows.

The temporary camp will be located alongside the permanent camp and will provide 1,184 beds in tents. The temporary camp will have only accommodation, toilets, and showers; other services will be provided from the permanent camp. The temporary camp will be closed when operations manpower has reached steady-state.

18.8.3 Greenhouse

There is already a plant nursery on site but this will need to be expanded for closure. The final greenhouse will be built after the Closure Plan has been approved, two years before the planned end of operations. Shelters will be provided with space to grow vegetation, which will be used to cover disturbed areas during closure. These facilities will be built on the temporary construction areas that have previously been disturbed.

18.8.4 Security

The Project access control will be located at km 15 on the access road to the site, which starts at the Troncal Amazonica road near Los Encuentros. This main access control will provide security and control for the site, and will be the only point where authorized personnel will be checked into the site.

Security chain link fence will be provided for those areas that require a physical barrier for security or to prevent the ingress of animals.

18.9 Off-site Infrastructure

In order to reduce the impact of the Project footprint, the 2016 FS assumed that some support facilities would be located off the main Project site. No specific site has been determined, as the actual final location will be subject to more detailed study in a later Project stage. Therefore, for the purpose of the 2016 FS, the off-site facilities were considered to be conceptually located 12 km from the main Project site. These facilities would include the following:

- Guard house
- Light vehicle shop (Contractor)
- Warehouse and laydown area
- Main office building

- Canteen.

Lundin Gold has established administration offices in Quito and in Los Encuentros. These existing offices provide administrative and logistics support to the Project, and are not part of the Project capital costs.

18.10 Power and Electrical

The Ecuadorian electrical system is based on a high quality electricity service matrix, the distribution system is called the Sistema Nacional de Distribución (SND, National Distribution System). The SND is controlled by CELEC EP Transelectric, a government institution in charge of power transmission and distribution.

The Project site is located within the supply concession area of the Empresa de Energia Regional del Sur (EERSA, Regional Electric Company of the South). SND has no substation near to the Project with sufficient capacity or reliability to feed the Project. Lundin Gold is participating in a public infrastructure investment to reinforce the SND matrix in the area, and is contributing to the installation of a transmission line between Taday and Bomboiza.

The overall Project power requirements are expected to be met via a 230 kV double circuit transmission line from the Bomboiza substation. The contract for this substation has been awarded and it will be built at the same time as the Taday–Bomboiza transmission line. The Bomboiza substation will be situated approximately 50 km away from the FDN site. A transmission line will be built from the Bomboiza substation to the El Pindal substation, near Los Encuentros. This system will be a public transmission line and substation, owned and operated by CELEC EP Transelectric, with installation paid for by Lundin Gold.

From the El Pindal substation, a single-circuit, 230 kV dedicated transmission line will be built to feed the Project. It is planned to build the FDN main substation on the process area platform. This substation will step down the power to 13.8 kV, and will distribute power throughout the plant site at this voltage.

The annual average power demand is estimated to be about 222,000 MWh.

18.11 Communications

The communications system for the Project will consist of:

- Fibre optic network infrastructure
- Telephony system
- Radio communications
- Mobile telephony

- Satellite communications.

The data management system will be connected to the communications systems.

18.12 Fuel

The diesel storage and distribution system will be located just off the main access road, close to the process plant access road. The fuel storage is designed for one week's storage capacity for surface vehicles. It includes supply stations and a transfer tank.

Another diesel storage and distribution system will be provided at the mine surface infrastructure area, to supply the mine trucks.

18.13 Water Supply

Domestic (non-potable) water for the camp and process plant will be supplied from fresh water intakes in nearby creeks. This water will be for sanitary usage (e.g. showers, toilets, emergency showers).

Lundin Gold already has a permit for 1.15 L/s for domestic non-potable use (refer to Section 4) and is in the process of obtaining a permit for a second intake point for 1.6 L/s. Both permits will be used for supply of domestic non-potable water during construction.

A water management pond will be located at the permanent camp site which will collect direct precipitation and runoff from the camp area. During construction the pond will be used for unaffected contact water management and domestic non-potable water supply. After the construction phase, the pond will be used for domestic non-potable water supply only. This will be the main domestic non-potable water source. A secondary source will be an intake located in a nearby creek. Lundin Gold is currently applying for a permit for the creek for the main camp. The requested intake flow is 6.3 L/s.

The process plant platform will be located about 5 km east of the main camp area, and a separate source of domestic non-potable water is required. The intended source is a creek located north of the process plant. Lundin Gold is currently applying for this permit. The requested intake flow is 1.1 L/s and the permit is expected to be required only during the Project operations phase.

There will be no potable water produced on site. All drinking water will be bottled and supplied by a local supplier.

Industrial water for construction will be required mainly for concrete mixing, shotcrete application, excavation works and maintenance activities. It is estimated that a peak volume of 9,870 m^3/month and a peak flow of 7.6 L/s of industrial water will be required during the construction phase. Four water management ponds will be used for industrial water supply during construction. The ponds will remain operational through operations as industrial water supply back-up; three of them will receive affected waters during operations.

To maximize water reuse and recycling and minimize demand for external make-up water, TSF effluent and paste plant effluent will be used as make-water for the process plant. Approximately 120 m^3/h will be used as process plant make-up meeting water quality and quantity requirements.

18.14 Comments on Section 18

The QPs note:

- The infrastructure design is in line with the planned process and mining rates, and is appropriate for a greenfields development in a remote area
- In order to reduce the impact of the Project footprint, some support facilities will be located approximately 12 km from the process plant and mine site. A new main access road is planned
- A borrow pit will be required to provide the materials needed for construction and mine backfill, from construction through to mine closure
- Lundin Gold is participating in public infrastructure investment in order to ensure power from the national grid is available to the Project.

19.0 MARKET STUDIES AND CONTRACTS

19.1 Introduction

The Project will produce doré bars and gold–silver concentrate.

Requests for expressions of interest in the gold–silver concentrate were sent to a number of different potential off-takers. Preliminary terms were received by Cliveden Trading AG (CTAG), and form the basis for the indicative terms used in the financial model.

19.2 Market Studies

Lundin Gold, through Aurelian, commissioned CTAG to prepare an independent marketing study on the planned products from FDN. The study was prepared on the understanding that the most probable method for metallurgical treatment was a conventional crushing and grinding circuit, followed by a gravity separation and flotation, producing a precious metal doré and a gold–silver concentrate for sale to third party off-takers.

Specifically, Lundin Gold requested information as to whether the gold–silver flotation concentrate and the precious metal doré would be marketable and the likely terms expected from potential off-takers of the products.

Potential receivers of gold–silver concentrate product include the following in general:

- Copper smelters
- Gold roasters
- Gold pressure oxidation plants (POX)
- Bacterial leach plants
- Lead smelters
- Commercial buyers (metals traders).

As part of the study, CTAG contacted a number of smelters, roasters and metals traders for information regarding gold–silver concentrate sales, and received indicative smelting terms based on the assay data of the concentrate produced from early testwork performed by Kinross.

Subsequent to receipt of the indicative terms in August 2015, CTAG and representatives from Lundin Gold met with a number of these smelters, roasters and traders to establish relationships and lay the ground work for future sales.

In February 2016, a comprehensive metallurgical testwork update was completed by SGS under the supervision of Amec Foster Wheeler, Santiago. Following the completion of this new composite testing, revised assay data was provided to the potential off-take partners for updated indicative sales terms.

Gold–silver doré bullion is typically sold through commercial banks and metals traders with sales price obtained from the World Spot or London fixes. These contracts are easily transacted, and standard terms apply. CTAG expects that the terms of any sales contracts would be typical of, and consistent with, standard industry practices and would be similar to contracts for the supply of gold–silver doré elsewhere in the world. Limited additional effort is expected to be required to develop the doré marketing strategy.

Information provided to CTAG assumed the following:

- The doré is expected to contain >98% precious metals and <2% base metals and impurities
- The precious metals portion is expected to contain approximately 60% gold and 40% silver
- It is also assumed that the doré bars will be produced with a mass of approximately 35 kg; however, there is a possibility that larger bars may be produced due to the high silver content (silver being lower in density than gold)
- No deleterious elements have been identified in the doré.

Based on the bids received for the gold–silver concentrate and the standard terms for precious metal bullion, the CTAG report concluded that there is significant interest in both products, and the indicative terms are as follows:

- Gold–silver concentrate
  - Pay 97.0% gold content
  - Pay 95.0% silver content
  - A treatment charge of US$275/dmt of concentrate is applied
  - Refining charge of US$6.00/oz for gold and US$0.50/oz for silver
  - Penalties for deleterious elements total approximately US$8.00/dmt of concentrate.
- Gold–silver doré
  - Pay 99.9% gold content
  - Pay 99.0% silver content
  - A refining charge of US$0.35/oz is deducted.

The information included in this section and the marketability of the products are based on CTAG's in-depth knowledge of the gold concentrate and doré markets, indicative terms received based on the FDN assay data, and various documents in the public domain.

19.3 Commodity Price Projections

This Report utilized consensus metal prices derived from: bank analysts long term forecasts, historical metal price averages and prices used in publically disclosed comparable studies. Gold and silver price projections are detailed in Section 22.

- Gold price: US$1,250/oz
- Silver price: US$20/oz.

Metal prices were kept constant throughout the life of the Project.

19.4 Contracts

19.4.1 Sales and Marketing Contracts

No contracts are currently in place for any production from the Project.

Amec Foster Wheeler notes that most precious metals concentrates are traded on the basis of term contracts. These frequently run for terms that have durations from one year to 10 years, although many long-term contracts are treated as evergreen arrangements that can continue indefinitely with periodic renegotiation of key terms and conditions.

19.4.2 Other Contracts

No other contracts are currently in place for the Project.

19.5 Comments on Section 19

The QPs have reviewed the information provided by Lundin Gold and CTAG on marketing and contracts, and note that the information provided is consistent with the source documents used, and that the information is consistent with what is publicly-available on industry norms. The information can be used in mine planning and financial analyses for the Project in the context of this Report.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

Baseline studies have been undertaken to support the FDN Environmental Impact Study (EIS), and subsequent updates to the FDN EIS. The most recent update to the FDN EIS was prepared by Cardno Entrix, Lundin Gold's environmental consultant. The update to the FDN EIS was prepared in support of applications for the exploitation phase, and includes the beneficiation, smelting and refining phases.

The physical (abiotic), biotic, social, economic and cultural baseline has been characterized for the Project using primary information gathered in the field, and secondary information from official sources such as Ecuadorian Government records. Field studies and data gathering for the baseline studies were undertaken between 2008 and 2016.

A summary of the baseline studies is provided in the following sub-sections.

20.1.1 Abiotic Baseline

Climate

The average annual temperature for the area is 17.9℃, and the relative annual humidity is 89.9%.

The average annual precipitation is 3,414 mm, with monthly precipitation varying between 191 mm (August) and 342 mm (May).

The estimated potential annual evapotranspiration is 896 mm (the pan evaporation is estimated to be 1,103 mm).

The maximum probable precipitation (MPP) in 24 hours is estimated to be 400 mm based on seven years of data. This should be updated when there is a precipitation record spanning at least 10 years.

Geology, Geomorphology, and Soils

The area is part of the Ecuadorian Eastern Sub-Andean Zone, which is made up of a series of marine-continental sedimentary rocks. It is located on the large landscape area called the Sub-Andean Region. Horizontal to inclined structures are present, which are dissected irregularly by ravines and slopes, anticlinal domes, and small synclines, with karstic modelling in some areas. Physiographically, these form a group of mesas, inclines, ravines, mountains and hills with moderate to very steep slopes.

From a physical–mechanical point of view, the soils have a residual and sedimentary residual origin, fine to medium grained. They are more than 2 m thick and are mainly silty clays and clayey silts. The soils have natural low to medium density. Hence, they are potentially vulnerable to erosion, they have medium to high expansion and contraction, and low permeability. To a lesser extent, granular soils of alluvial origin are found, including sandy silts and well-graded sands. These are fine- to medium-grained and medium to high density, making them highly permeable.

Hydrology

The hydrological and hydraulic model was mainly developed to evaluate peak flows and flood levels for the Machinaza River (adjacent to the plant and mine) for several return periods. The models show that the maximum flows vary between 1,266 m^3/s and 1,466 m^3/s for the 100 year and 500 year events; this is equivalent to average flood levels between 1,394 masl and 1,396 masl, respectively.

Surface Water Quality

In order to characterize the water resource, water bodies that could be directly or indirectly influenced by Project activities were examined. Water samples were taken, analyzed, and compared against criteria provided in Ecuador's Ministry of Environment *Agreement No. 97-A, Book VI, Annex 1* (water quality criteria or WQC). The WQC provide for various water uses including:

- Human and domestic consumption
- Preservation of aquatic and wild life in fresh water
- Agricultural irrigation
- Livestock.

The Machinaza River has been assessed as being used for agricultural irrigation and livestock in the vicinity and downstream of the Project. Therefore parameters were assessed against these criteria (i.e. agricultural irrigation and livestock). The parameters sampled and analyzed were: pH, dissolved oxygen, aluminum, arsenic, barium, boron, cadmium, cobalt, mercury, chrome, cyanide, copper, iron, manganese, nickel, silver, lead, selenium, residual chlorine, oil and grease, total petroleum hydrocarbons, phenols, surfactants, fecal coliforms, nitrites and nitrates, and zinc.

Table 20-1 illustrates the general baseline results against the WQC (note that 28 monitoring locations were sampled along the Machinaza River, and the table is an average compliance indicator). Findings included:

- The water in the zone has an acidic tendency, which may be due to the high level of sulphates occurring in the Project area, or due to the influence of artisanal mining activities that occur in the vicinity of the Project. SRK has noted that many tropical rivers have acidic tendencies
- Concentrations of dissolved oxygen are suitable for the preservation of aquatic life
- Arsenic, barium, boron, cadmium, cobalt, mercury, chromium, nickel, silver, nitrate, and nitrite concentrations are below the WQC for all water uses

- Cyanide concentration is below the WQC
- Copper and residual chlorine concentration is higher than the WQC; however, the laboratory detection level is above the WQC
- Aluminium, iron, manganese, lead and selenium concentrations are higher than the WQC to preserve aquatic life
- Oil and grease, total petroleum hydrocarbons (TPH), phenols and surfactants have concentrations below the WQC
- Coliforms concentration is higher than the WQC
- The zinc concentration is below the WQC to preserve aquatic life.

Sediments

The results were mostly lower than the criteria. Some parameters were higher than the criteria, and these parameters also exceeded surface water criteria. Free cyanide was not detected in any of the sediment samples.

There are no industrial, agricultural or livestock activities near the area where the samples were collected, hence, the study concluded that the values reported in the sediments were due to the natural physical-chemical composition, as well as the edaphological (influence of soils on flora) and geological characteristics in the areas of study.

Hydrogeology

Permeability tests indicated that the rock has very low permeability (ranging between 1×10^{-9} m/s and $1x\ 10^{-8}$ m/s). According to the hydro-geochemical analysis, the water in this area is from recent infiltrations, and is generally made up of sodium bicarbonates, with some hydrogeochemical alteration due to the mineralization in the area.

The aquifer system is defined by low potency and discontinuous strata, with very superficial phreatic levels and replenishment mainly from infiltration from the surface drainage systems. Replenishment of aquifers detected in other lithological units is regional, and comes from precipitation in higher areas where there is low-potency saprolite with larger grain sizes, and also coincides with high fracture zones.

Air Quality, Noise and Vibration

During the study period, all the analyzed parameters for air quality (CO, NO_2, SO_2, O_3, and particulate material (PM10 and PM2.5) were below the Ecuadorian Air Quality Guidelines for all of the monitoring points.

Table 20-1: Baseline Results

	Preservation Of Aquatic And Wild Life In Fresh Water	Agricultural Irrigation	Livestock
DO	✓	×	-
Al	×	×	✓
As	✓	✓	✓
Ba	✓	✓	✓
Bo	✓	✓	✓
Co	✓	✓	✓
Hg	✓	✓	✓
Cr	✓	✓	✓
CN	✓	-	-
Cu	×*	✓	✓
Fe	×	✓	-
Mn	✓	✓	-
Ni	✓	✓	-
Ag	✓	✓	-
Pb	×	✓	-
Se	×	✓	-
Cl	×*	-	-
Oils & Greases	✓	-	-
TPH	✓	-	-
Phenols	✓	-	-
Surfactants	✓	-	-
Faecal Coliforms	-	×	×
Nitrites & Nitrates	✓	✓	✓
Zn	✓	-	-

Notes:
✓ General Compliance with criteria
× General non-compliance with criteria
- No WQC available
* Laboratory detection limit is higher than the criteria value

Noise levels were measured in the baseline to obtain referential background noise levels for comparative evaluation with noise levels generated during operations.

Based on 2011 measurements, there are no vibrations above 0.3 mm/s of particle velocity, which is the human perception limit.

20.1.2 Biotic Baseline

Flora

In general terms, the study area contains two types of vegetation: natural forest (Bn) covers most of the study area, and natural forest with low disturbance (Bnpi) is seen in the outlying regions where vegetation has been cleared for exploration activities. These areas have been revegetated with species that are characteristic of the site.

The Ecosystem Classification System for Continental Ecuador proposed by the Ecuador Environmental Ministry (MOE) in 2013 was used to define the ecosystems in the Project operating area. Three types were identified:

- Evergreen foothill forests of the Cóndor–Kutukú Cordillera: Dense forests with a 20 m high closed canopy with taller trees 30 m high with abundant epiphytes. This is found in the Cóndor and Kutukú mountain ranges on hills and in depressions between 350 masl and 1,400 masl
- Evergreen low mountain forest of the Cóndor–Kutukú Cordillera: This is a woody ecosystem between 1,400 masl and 1,900 masl, occupying the steep slopes and high hills around the plateaus in the Cóndor and Kutukú mountain ranges, sitting on metamorphic and igneous rocks. The trees are covered in moss, and there is a large amount of plant litter on the ground
- Low mountain evergreen forest on sandstone plateaus of the Cóndor–Kutukú Cordillera: These forests are located in the high areas of the Cóndor and Kutukú Cordillera. This dense forest with an average canopy height of 10 m. The underforest is very dense, with abundant hemi-epiphytes, epiphytes, vines and ferns.

Vegetation Units

The following vegetation units are found in the study area where permanent monitoring areas have been established. These units follow a preliminary vegetation classification proposed for the Cóndor Cordillera (Neill, unpublished, cited in Entrix, 2010):

- Valley forest on poorly-drained soils: This type of vegetation is found between 1,400 masl and 1,450 masl, and is characteristic of the bottom of the Machinaza River valley, located on relatively flat land with alluvial soils and some marshy sites
- Mature forest with springs under a sandstone plateau: The vegetation in this classification is similar to that of the previous vegetation type, except that the soil is much more fertile; it is made up of sand from erosion of the sandstone and clay layer in the outcropping of bedrock under the sandstone plateaus. The eroded sand flows into the springs. Diversity, diameter and height of the trees are much greater than those of the mature forests on sandstone plateaus
- Mature forests on sandstone plateaus: This vegetation type is located on plateaus of sandstone rock from the Hollín Formation (Neill, 2009). The forest is characterized by being very thick, and the substrate is crystalline sand that is very acidic and poor in nutrients. There is a large abundance of bromeliads. The vegetation grows very slowly except for fast growing grass species
- Mature forest outside of sandstone: This vegetation type is located between 1,400 masl and 1,500 masl. This type of vegetation is found on the geological substrate of the Zamora Batholith. The soil has a high clay and a low sand content.

Protected Areas

The El Zarza Wildlife Refuge and El Cóndor Mountain Range Protected Forest are located just outside of the Project area; no Project activities will be carried out in these areas. There are no other protected areas in the vicinity of the Project.

Species

Baseline studies (Entrix, 2015) identified 209 species and 808 individuals, 103 genera and 54 families in the study area. The dominant species is the *Dictyocaryum lamarckianum* (Arecaceae) palm, with 68 specimens. Between 41 and 23 specimens of the following tree species were abundant in the area: *Alchornea grandiflora* (Motilón*), Croton pachypodus, Graffenrieda uribe*i (Duco), *Neea spruceana, Clusia elliptica* (Duco) and *Graffenrieda emarginata*. There were 91 species with one specimen each, which were considered to be scarce.

The Shannon Diversity Index showed that the study area is made up of highly diverse forests. Generally speaking, it is highly heterogeneous, with a large number of species with a single specimen. The "mature forest outside of sandstone" vegetation type is the most diverse.

Twenty-one species were recorded during the baseline studies that are of ecological interest. This is based on species that are:

- New species or probably new species
- Endemic to the area
- Listed on the International Union for Conservation of Nature (IUCN) Red List of Threatened Species
- Listed on the Convention on International Trade in Endangered Species of Wild Flora and Fauna (CITES)
- Listed on the Red Book of Ecuador.

As required by the existing Environmental Licence for the Project, a Rescue and Relocation Plan for Biotic Species (Entrix, 2015) has been developed for these species, which provides the list of species targeted for rescue, the prioritization for rescue, and the justification.

Terrestrial Fauna

Mammals

Thirty species of mammals were registered, grouped into eight orders and 14 families. By number of species, the most representative order was Chiroptera (bats), which included two families and 13 species, representing 44.82% of the total registered. At the family level, the Phyllostomidae family was most abundant, with 12 bat species.

The rest of the orders and families presented a lower percentage of the species identified.

Sturnira oporaphilum (eastern yellow-shouldered bat) and *Carollia brevicauda* (silky short-tailed bat) with 17 specimens each, dominated in the study area. The *Carollia perspicillata* bat species, with 11 specimens, was one of the most abundant species registered.

The Andean slender mouse opossum (*Marmosops impavidus*), the silver fruit-eating bat (*Dermanura glaucus*) and the mountain paca (*Cuniculus taczanowskii*) were classified as common species, registered throughout the sampling area.

There were no endemic species recorded (Entrix, 2015); however, 22 species are listed on either the IUCN Red List or CITES. The *Puma concolour* (mountain lion) and *Eira barbara* (tayra) are listed as Vulnerable on the Red Book of Ecuador. Management of these species is described in the Rescue and Relocation Plan for Biotic Species.

Birds

There were 59 species recorded during the baseline studies (Entrix, 2015) that are listed on either the IUCN Red List, CITES, or in the Red Book of Ecuador. Of these, one species is listed as Endangered under IUCN, *Heliangelus regalis* (royal sunangel, a type of hummingbird). Two species are listed as Vulnerable under IUCN: *Patagioenas subvinacea* (ruddy pigeon) and *Pyrrhura albipectus* (white-breasted parakeet). The white-breasted parakeet is also listed as Vulnerable in the Red Book of Ecuador, together with the *Aburria aburri* (wattled guan) and *Chamaepetes goudotii* (sickle-winged guan). Management of these species is described in the Rescue and Relocation Plan for Biotic Species.

There are human settlements in the El Pindal vicinity, and the habitat has been modified by crop-growing and grass-growing for livestock. Omnivorous bird species were observed in disturbed areas, such as the *Coragyps atratus* (black vulture).

Reptiles and Amphibians

There are more amphibians than reptiles; 11 of the 40 families of registered herpetofauna are reptiles, and most of these are snakes (Viperidae and Colubridae). Two of the snake species are venomous (coral snake and X snake).

Nearly all of the herpetofauna recorded in the baseline studies (Entrix, 2015) are in threatened categories according to IUCN or CITES. There is one species listed as Endangered under IUCN: *Pristimantis prolatus* (a frog endemic to Ecuador), and one species listed as Vulnerable: *Pristimantis serendipitus* (frog). Management of these species is described in the Rescue and Relocation Plan for Biotic Species.

Insects

Diurnal butterflies and coprophagous beetles were studied; 70% of the species identified were beetles and 30% were butterflies.

Aquatic Fauna

Ichthyofauna

At the sampling points, 20 fish species were found in total. Eight of the 20 species were found at three of the sample points in the Machinaza River watershed. Although there is a low number of species, more than 100 specimen were found at two points; however, the diversity is low (low number of species). This was confirmed by a point-by-point interpretation of the result using the Shannon Index. Low fish species diversity could be due to the physical–chemical characteristics of the water, which may explain why the ichthyologic resource is not exploited in the area. None of the species recorded in the surveys are listed in the IUCN, CITES or Red Book of Ecuador.

Aquatic Macro-Invertebrates

Rare species (between one and three specimens) dominate the composition of macro-invertebrates in the area. The species accumulation curve and the Chao 1 index lead to the conclusion that more than 60% of the total number of expected species for the area have been registered, and that in future studies 48 additional species could still be reported. None of the species recorded in the surveys are listed in the IUCN, CITES or Red Book of Ecuador.

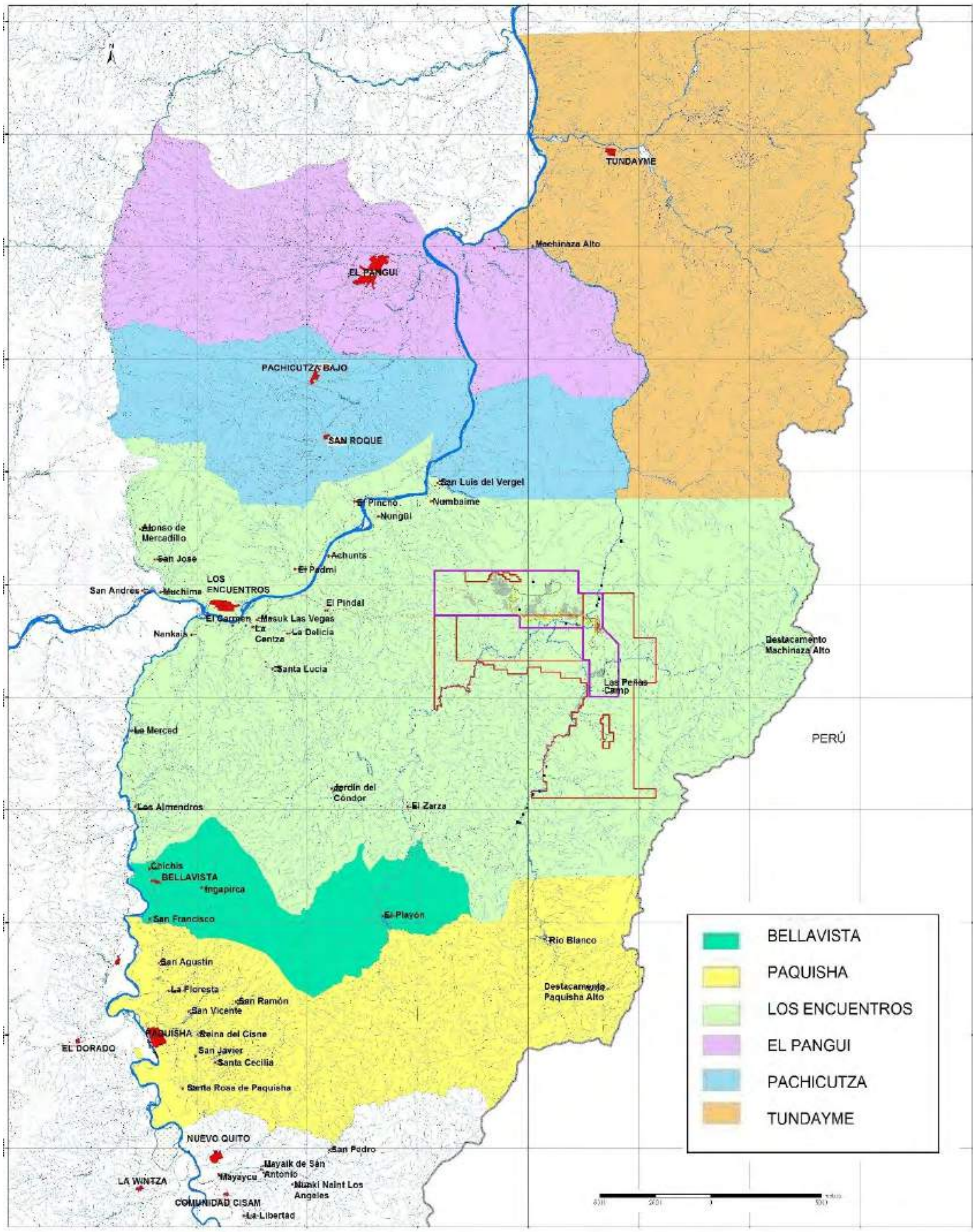
Aquatic species monitoring is done approximately every six months or during seasons of higher or lower intense rains. Hence, it should be possible to establish which changes are due to climatic conditions, and therefore not caused by Lundin Gold's activities.

20.1.3 Social Baseline

The Project is located in the parish of Los Encuentros in the Yantzaza county of the province of Zamora Chinchipe, in the southern Amazon region of Ecuador, not far from the Ecuador/Peru border. The Project's indirect influence is expected to extend to some neighbouring communities.

Figure 20-1 shows the major administrative areas in relation to the Project.

Figure 20-1: Political–Administrative Location of FDN Project



Note: Figure courtesy Lundin Gold, 2016.

Lundin Gold advised Amec Foster Wheeler that the Project is not located on ancestral land, and there are no Indigenous communities within the Project area. Only two people (non-Indigenous) were identified as residing within the Project area, and Lundin Gold has indicated that the company is already in discussions regarding voluntary resettlement. Therefore, in Lundin Gold's view, Free and Prior Informed Consent is not applicable to the Project. Amec Foster Wheeler concurs that, based on the findings by Lundin Gold and its advisors, because there are no indigenous peoples in the immediate Project area, no indigenous peoples appear to be impacted by the Project development and require resettlement efforts, and the Project is not on ancestral lands, Free and Prior Informed Consent is not applicable to the current Project configuration.

Demographic Aspects

Population

According to data from the 7th Population and 6th Housing Census by the National Institute of Statistics and Census (INEC) in 2010, Zamora Chinchipe province has a total of 91,376 inhabitants (0.63% of the total national population). Yantzaza county has a population of 18,675 residents, putting it second in size of the nine Zamora Chinchipe counties. The county represents 20.44% of the provincial population.

Los Encuentros is a rural parish located in Yantzaza county, and has one main population centre (the parish seat and home of the parish government). The parish population totals 3,560 inhabitants with a slightly higher male population (54.09%) than female. There are also several scattered population centres, known as communities, neighbourhoods, and sectors.

Ethnicity

The parish population is mostly mestizo (mixed race) (68.96%), followed by indigenous nations (27.60%); most of whom are Shuar, and to a lesser extent, Saraguro. The majority of the population is native to the county, but a considerable portion of the population has emigrated from the neighbouring province of Loja.

There are few indigenous communities; they have integrated into the Ecuadorian dynamic through interaction. Although they have conserved their main cultural heritage, they face the challenge that the new generations are not interested in continuing traditional practices.

Migration

Most of the population is native to the county, although a considerable percentage of the population has emigrated from the neighbouring province of Loja. According to the 2010 census, the Zamora Chinchipe province registered a total of 2,094 immigrants, most of whom were men.

In the parish of Los Encuentros the distribution of immigrants was opposite to the provincial trend, more women immigrated than men. The main reasons for immigration were a lack of employment and forming their own homes (leaving the parental home). There was also immigration for education; because of the rural characteristics of the area there are few institutions where students can continue their high school or university studies, especially the latter.

The economically active population (EAP) is mostly made up of men, reflecting the traditional gender inequality in the area in terms of economic activities. Gender inequalities in access to paid productive activities are highly significant. For men, the average EAP is 77.06%; for women it is 46.65% (these are higher than the parish indicators).

Most people work in agriculture or provide unskilled labour (73.72% of the population on average). The population is primarily dedicated to farming and artisanal mining. The local economic structure of the communities is based on subsistence farming.

A total of 46.48% of the population stated that the household income was between US$251 and US$500 per month (minimum wage range); 30.11% reported income between US$101 and US$250 per month (below minimum wage); and 15.93% reported income between US$500 and US$1,000 per month.

At the provincial level, the percentage of the population that is poor with unsatisfied basic needs (UBN) is 73.84%; Bellavista parish (96.35%) is the parish with the highest percentage of population with UBN, followed by Los Encuentros parish (89.15%).

Education

Data from the last population census show that in all cases, illiteracy is higher in the female population, with the highest percentage in Tundayme (12.70%) and the lowest in Paquisha parish (5.20%). For the male population, Pachicutza parish has the highest rate (7.90%) and the lowest rate is in Paquisha parish (2.50%).

There are currently four educational establishments in the area. The 10 de Noviembre Millennium School in Los Encuentros is the only school that offers all grades from kindergarten to high-school graduation. In the area of study, men have an average of 7.8 years of school enrolment, and women have an average of 7.6 years. At the parish level, only 0.4% of the population has a post-graduate degree; in rural locations there is no one with this level of education, bringing the average for the area down to 0.01%.

Health

The current health status of the population was surveyed for the recurrence of illnesses over the month prior to the social baseline survey in April 2015. The data revealed that the population is in relatively good health; in all of the locations in the parish, the answer to the question of having been ill recently was predominantly “no”. It was also evident that the most common recurring illnesses are related to respiratory illnesses such as colds, flu, strep throat, bronchitis and minor gastrointestinal problems. Other less-frequent illnesses included headaches, backaches, dermatitis and urinary tract infections. These illnesses are typical in rural sectors of the Amazon. According to the data compiled, there is no evidence of serious or catastrophic illnesses in the area.

The mortality rate was studied for the year prior to the survey. During that year, the parish registered 24 deaths, predominantly in the parish seat. The causes were illness, accidents and old age. Illnesses causing death included unclassified death (unregistered causes), pneumonia, strokes and cancer (cancer cases were minimal).

The health infrastructure available consists of a basic hospital (in Yantzaza) which has an embedded health unit (i.e. a health clinic that operates as a centre for external consultations), two medical clinics (El Pincho, El Zarza), and two health centres (Paquisha, Los Encuentros). On average less than 1.00% of the population in the study area go to a private hospital, and only 1.62% receive medical attention from a private doctor.

Public Utilities

For populations in the Amazon there are several obstacles to providing public utilities. The most widely-provided utility is public power supply and the least-provided utility is sewage collection and treatment. The coverage and access to water supply, waste removal, public power supply and conventional and cell phones was evaluated.

For many locations in the parish, houses are connected to the local public water network. On average, 72.88% of the homes in the locations around the parish are connected; water from natural water flows supplies water to 27.08% of the households.

On a community level, the main human waste elimination system is septic tanks (47.72% average), and to a lesser degree connection to the sewage system (14.77% average). Sewage connection is limited to urban areas and immediate surroundings.

Electrical supply coverage in the area is at 82.44% on average.

The removal of solid waste through municipal garbage collection is 33.88%.

On average 16.96% of the population has direct access to fixed telephone lines. The use of cell phones is higher, with 81.46% of the population using cell phone services, even though the coverage is inadequate. Internet service is very limited, only 6.85% of the population has internet access.

Perception

Current Environmental Situation

Approximately 56.40% of the surveyed populace stated that they currently do not feel any kind of environmental contamination; 41.74% stated that they did feel contamination. Of those who perceived environmental contamination, 49.97%

identified air as the contaminated medium, 77.37% identified water, and 44.06% identified soil. The main cause of air contamination was related to the presence of garbage and fuel emissions; water contamination was associated with waste water and garbage in water bodies; and soil contamination was associated with garbage and the use of agro-chemicals.

Mining Industry

A total of 84.31% of the population considered that the mining industry is beneficial to the community; however at the same time 52.56% considered that it can be harmful to the community. This ambivalence was also noted in the general opinion about mineral exploitation in the area; 51.76% of the population were in favour, 48.34% were not in favour. It is evident that although the population is in favour of mining, there is a high percentage of people who prefer not to have mining activities in the area.

Lundin Gold

On average 81.84% of the population stated that they knew about Lundin Gold and its activities, 39.45% said that the company is doing studies in the area, and 56.86% knew that the company was in the exploration phase. A total of 79.69% supported Lundin Gold working in the area versus 10.49% that did not; 1.91% of the population was indifferent to the presence of the company, and 7.92% did not respond to the question.

20.1.4 Archeological Baseline

An archeological baseline has been compiled using results from preliminary investigations in the study area and specific investigations undertaken by Aurelian since 2008.

Archeological Investigations

Studies have included or applied the following methodologies:

- Archeological prospecting: Performed systematically with rigorous methodology. This research method uses intrusive prospecting, with shovel tests and diggings. Non-intrusive prospecting was also used making observation tours of areas with possible ancestral uses, as well as tours of local water bodies, swamps and flat surfaces. This methodology was the first step applied as part of the archeological research and determined that the specific areas of interest have archeological sensitivity that goes from zero to medium
- Archeological rescue: This methodology is also known as archeological excavation, and is applied when archeological prospecting gives positive results; i.e. when there is evidence of archeological remains. This work was also done systematically, using excavation, registering archeological evidence, recovering the evidence, and recovering samples and information, thus providing more elements for interpretation. When archeological evidence was identified excavation work

was extended horizontally to find continuation of the archeological evidence or remains

- Archeological monitoring: This work is divided into two stages: field work and desktop or laboratory work. Field work consists of direct observation and is done at the same time as earthworks, in order to determine whether there is any archeological material. When material is identified, it is immediately recovered, and earthworks can continue. In the laboratory, material and information from the field is processed and analyzed along with data from bibliographic sources. Aurelian performs earthworks only in areas where the company has obtained a National Cultural Patrimony Institute (INPC) Authorization, and always carries out archeological monitoring throughout the activity, as established in the Environmental Management Plan for each activity.

Based on the previous diagnostics and the areas where the Project will be built, research areas were divided into 12 sectors (Figure 20-2).

To date prospecting has been carried out in Sectors 1, 2 and 3 (2010); Sector 5 (2011); Sector 6 (2012); and Sector 7 (2015). Prospecting in Sectors 9, 10, 11 and 12 is currently underway. Although Sectors 4 and 8 are being investigated, they are outside the operational area for the Project.

There are three sectors that are considered to have medium sensitivity, one sector with low sensitivity, and two with no sensitivity (Table 20-3).

20.2 Waste Rock Storage Facilities

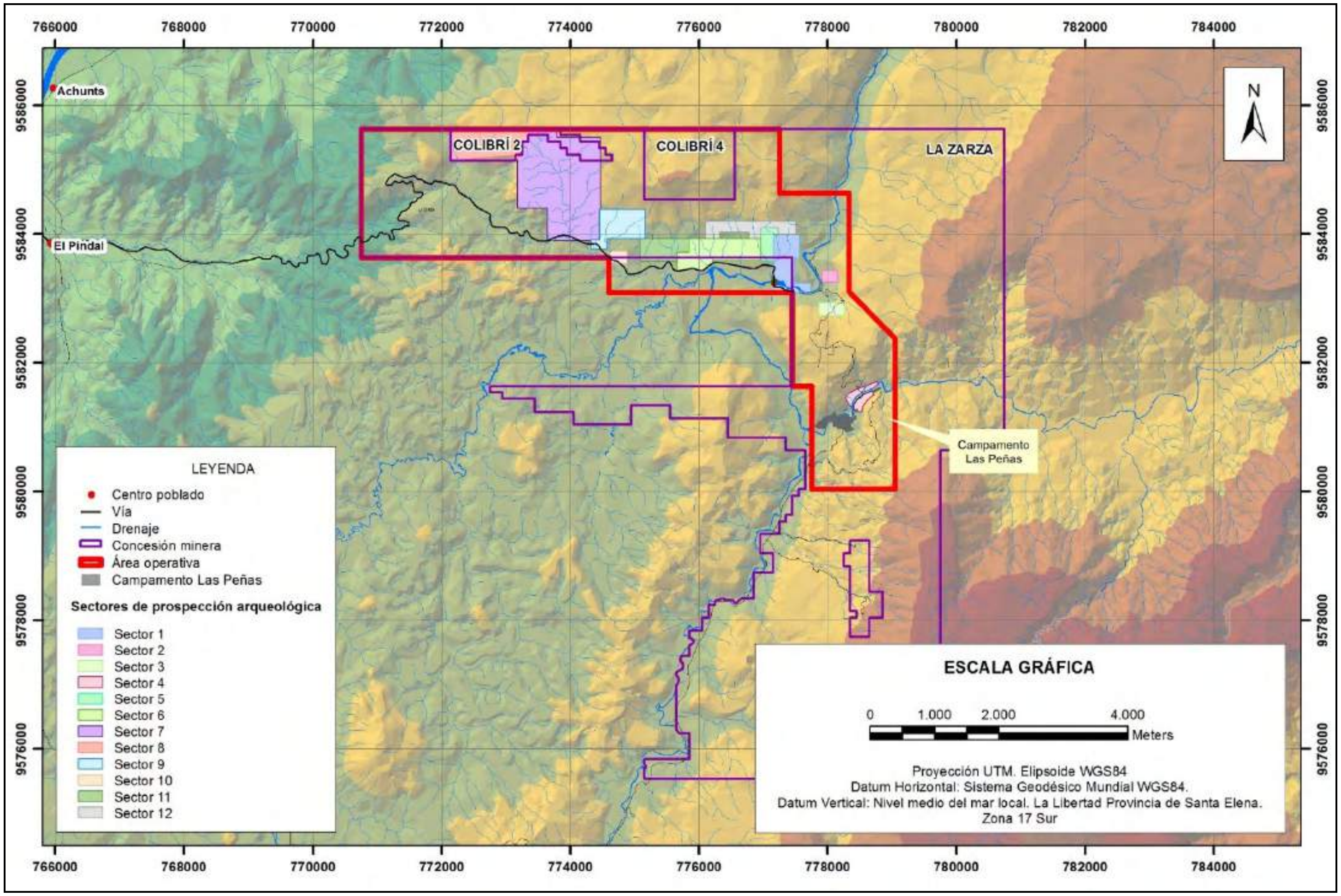
The excavation of the portals, declines, stope access tunnels and ventilation shafts will generate waste rock from a range of lithologies.

Waste from the declines will start coming to surface in 2017. The waste rock storage facility (WRSF) area has been designed to accommodate both “bad” (geotechnical properties are such that the material must be permanently stockpiled) and “good” (geotechnical properties are such that the material can be used in construction and backfill) waste.

Approximately 2 Mt of waste material will be generated. Of this, about 1.29 Mt (64%) will be returned underground as part of the backfill management strategy. The remaining material (approximately 0.74 Mt) will need to be permanently stored on surface.

The material left permanently in the WRSF will be mainly 55% intrusive andesite and 24% phreatomagmatic breccia. The allocated WRSF area, to be located south of the process plant, is sufficient to accommodate this material.

Figure 20-2: Location of Archeological Sites



Note: Figure courtesy Lundin Gold, 2016. Centro poblado = village or settlement; via = road; drenaje = drainage; concesión minera = mining concession, area operative = proposed operations area; campamento Las Peñas = current Las Peñas exploration camp; sectores de prospeccóon arqueológica = sections investigated as part of the archaeological baseline survey; escala gráfica = scale.

Table 20-2: Archaeological Sensitivity

Sector	Sensitivity	Observations
Sector 1	Medium	Archaeological site in Sector 1; four areas of archaeological interest (stone, ceramic, fragments and vegetal carbon) were recovered, dating from the sixth to seventh centuries.
Sector 2	None	—
Sector 3	None	—
Sector 5	Low	The definition of the archaeological areas of interest was based on the five areas researched through excavations.
Sector 6	Medium	Areas of archaeological interest were established. Cultural materials were identified in trenches; with a low density of ceramic and lithic fragments
Sector 7	Medium	Eleven areas of archaeological interest were found in this sector; additionally, carved rocks and stones were recorded.

Source: Aurelian Ecuador S.A. November 2015. Prepared by: Entrix, November 2015.

During operations all water runoff from the WRSF will be captured and treated.

Preliminary design for the long-term storage and management of the PAG material includes capping with a compacted saprolite layer to restrict water inflow, and covering with growth media for plant establishment. Further studies will be undertaken to confirm (via contaminant transport modelling) that this management strategy is effective in controlling ARD in the long term.

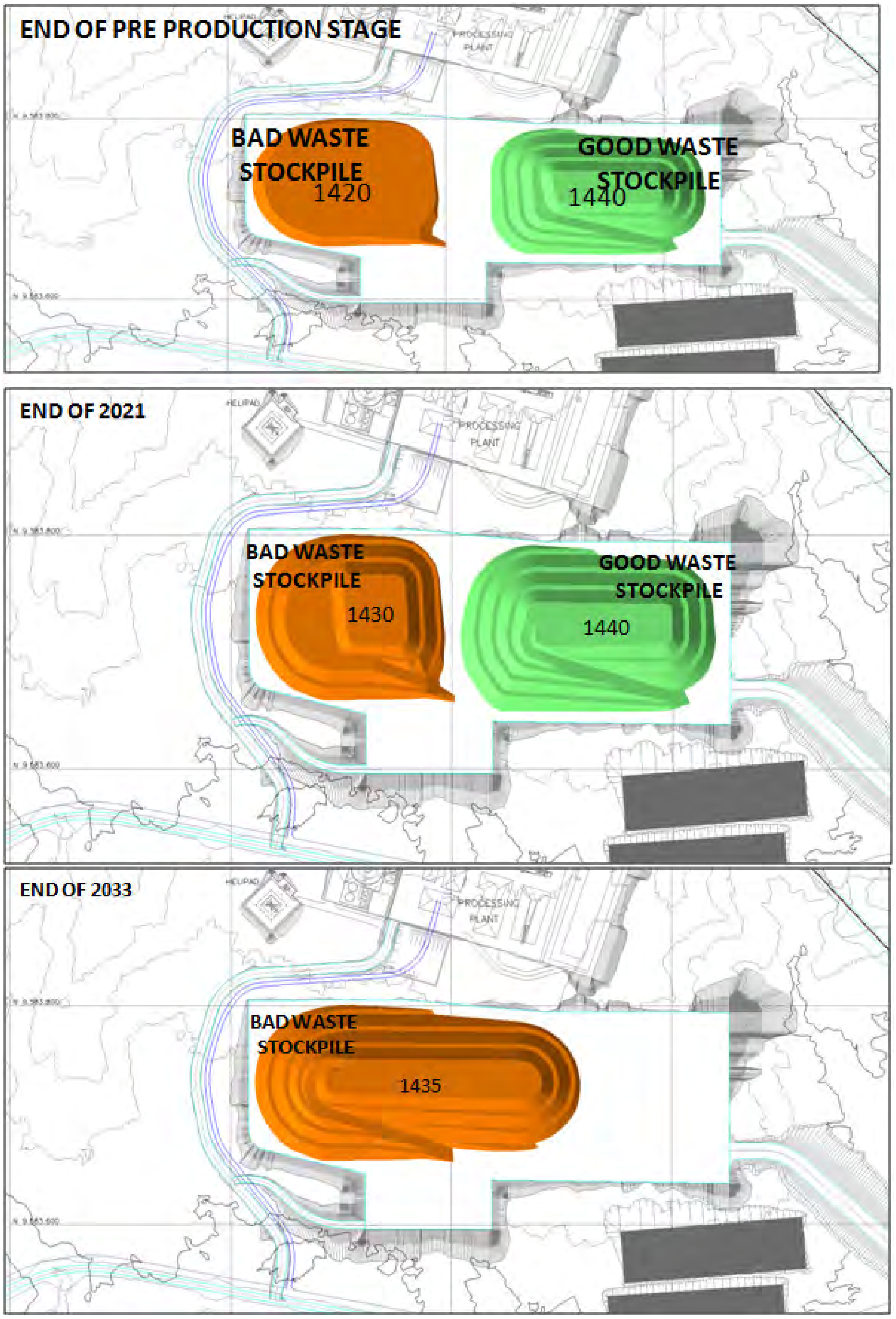
The facility design assumes slopes of 3H:1V with 5 m lifts.

A location plan is included in Figure 20-3.

A second WRSF will be located adjacent the Hollín Borrow Pit. This will store rock that is not suitable for use in construction or backfill. This waste rock facility is expected to contain PAG, and similar management techniques will be employed to control ARD to that described above for the main waste rock facility.

There is an existing WRSF, known as the South Waste Dump, which contains material that was excavated during the development by Kinross of the southern exploration decline. There is no plan to re-activate this facility for production; however, the WRSF may be partially re-opened to receive waste generated from training activities in the southern exploration decline.

Figure 20-3: End of Period Waste Rock Storage Facility Views



Note: Figure prepared by SRK, 2016. The WRSF area has been designed to accommodate both “bad” (geotechnical properties are such that the material must be permanently stockpiled) and “good” (geotechnical properties are such that the material can be used in construction and backfill) waste.

20.3 Ore and Low Grade Stockpiles

There will be three types of material stockpiled based on grade:

- High-grade (>7 g/t Au) – Almost never stockpiled
- Medium-grade (4.7 g/t Au to 7 g/t Au) – Maximum 30,000 t
- Low-grade (2.7 g/t Au to 4.7 g/t Au) – Maximum 170,000 t late in the mine life (Year 2033).

Hence, three long-term stockpiles are required to be built and maintained for the LOM. The area allocated for these stockpiles is close to the crushing station at the process plant on the surface.

As the stockpiles are intended to be temporary storages, the following design criteria are considered acceptable:

- 4.5 m berm
- 10 m ramp at 10% grade
- 22° overall slope angles
- 5 m lifts.

The initial total stockpile tonnes prior to commissioning of the process plant will be approximately 50,000 t. Where possible, a first-on-first-off strategy will be pursued in order to minimize oxidation and acid generation.

Material stored on the stockpile pad is expected to contain 2.6–3.8% sulphide, depending on the lithology and is PAG. The residence time of the stored material will generally exceed four months, which is the period required for the more reactive rock to start generating acid. Drainage will therefore contain elevated concentrations of sulphates and metals, and low pH.

These stockpiles will be depleted upon closure, and rehabilitated to ensure water running off these areas is neutral and the areas are stable.

Surface water drainage from the storage area will be diverted to holding ponds. At closure, any remaining reactive material in the stockpile will be removed and transferred to the waste rock pad.

A location plan is included in Figure 20-4.

20.4 Tailings Storage Facility

20.4.1 Proposed Location

The facility will be located in the uppermost portion of the valley, to minimize the catchment area and to maximize the separation distance from the Zarza River downstream. A layout plan of the proposed facility is included as Figure 20-5.

20.4.2 Site Conditions and Investigations

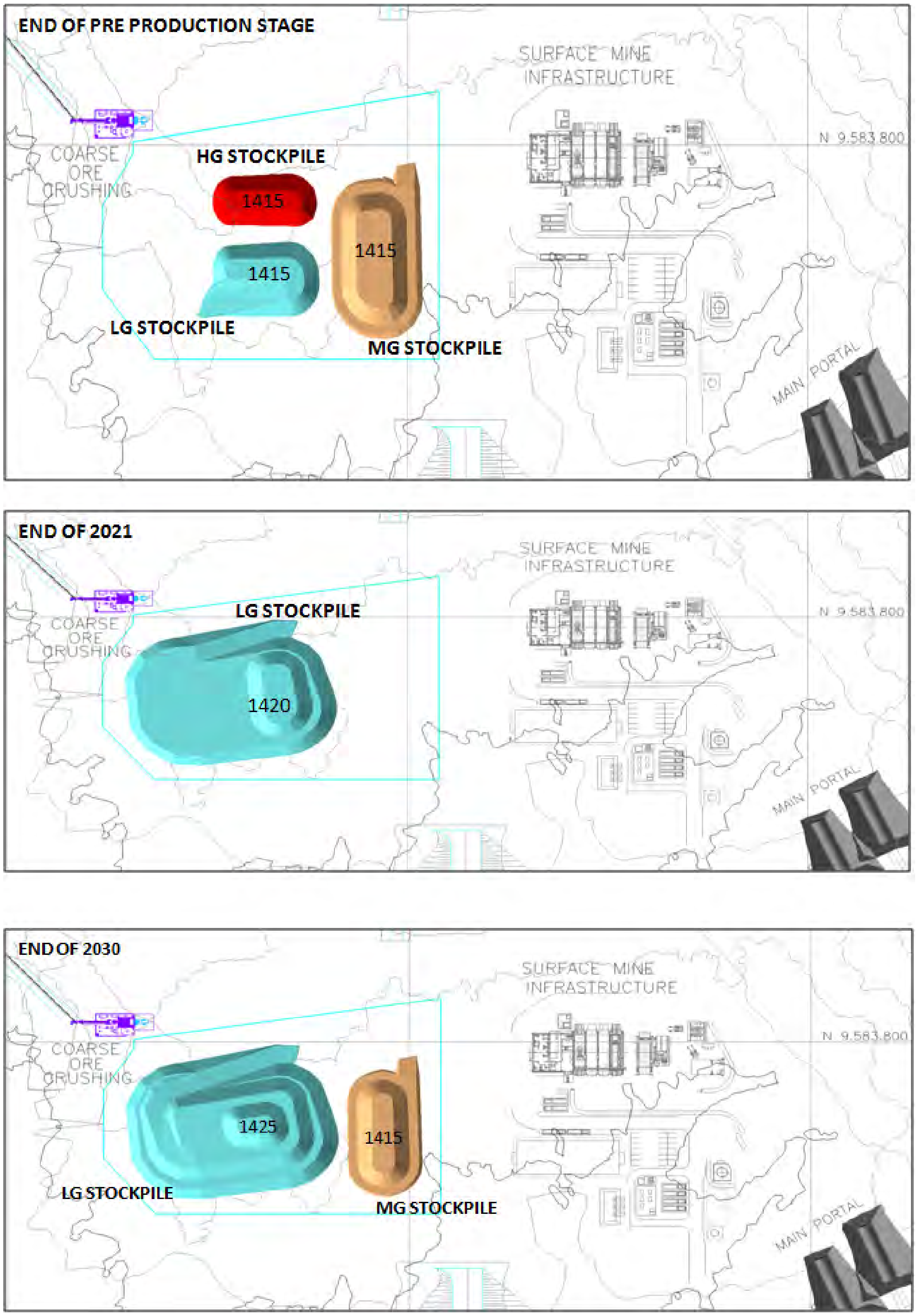
Field programs at the proposed TSF site were completed by KCB between 2009 and 2015, and included site mapping; deep drilling to bedrock, coring, logging and disturbed sampling within the dam footprint area; shallow test holes; visual inspections of exposures of the Hollín Formation upstream of the TSF site and surface flow water measurements in the main creek of the valley. The following were performed on selected drill holes: insitu permeability tests (packer and falling head), insitu density and strength interpretation using standard penetration tests (SPTs); deep and shallow stand-pipe well installations; and infiltration tests. Key findings included:

- The near surface materials have relatively low permeability with values less than 6×10^{-7} m/s
- Foundation materials show increasing trends of density, strength and stiffness with depth; foundation materials show a trend of decreasing fines content with depth
- Groundwater follows the topography and flows from the west and east sides of the valley catchment
- The predominant mineral in the saprolite samples is kaolinite with small portions of illite; no swelling minerals were identified.

20.4.3 Tailings Characteristics

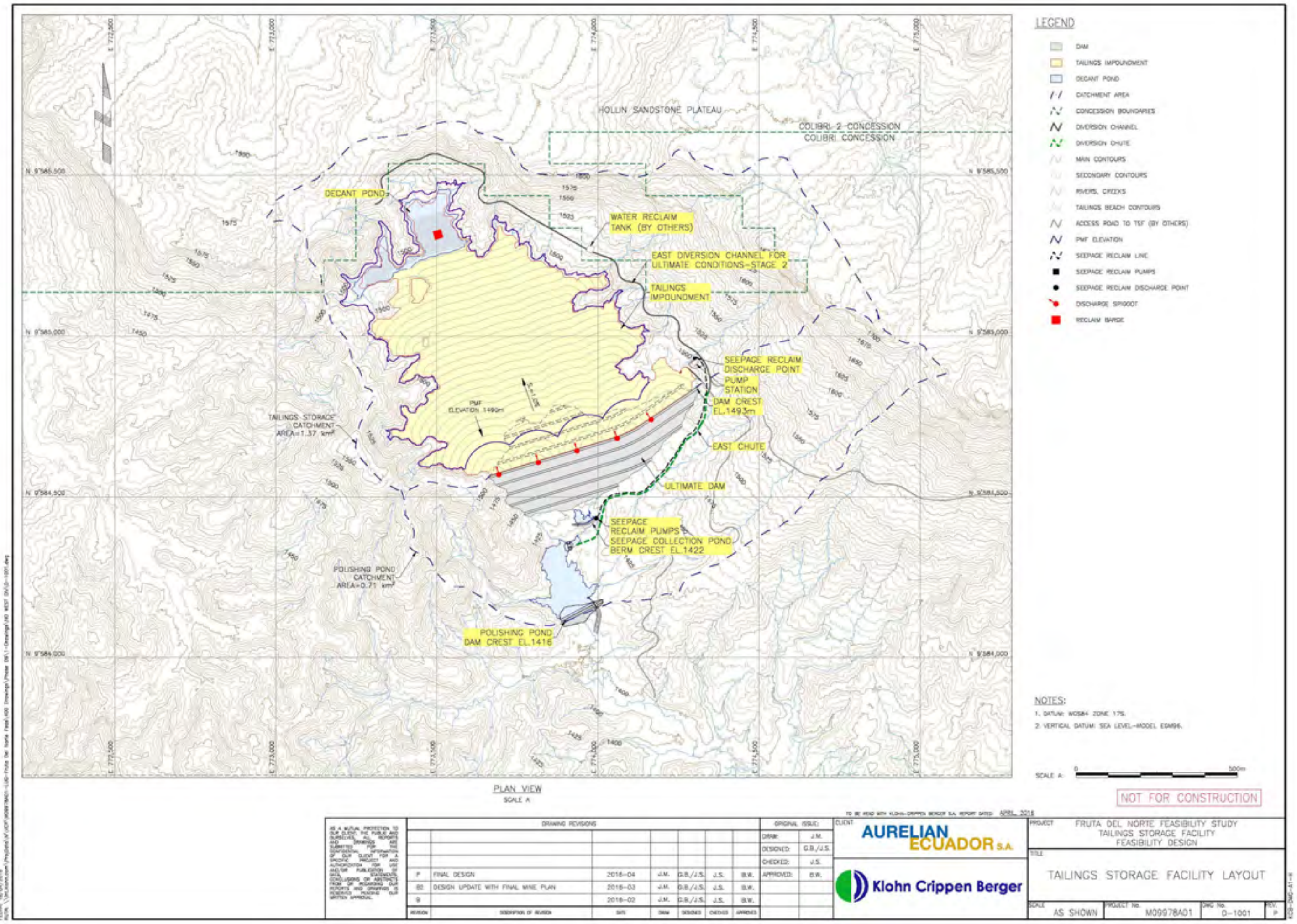
A geotechnical characterization assessment of the GFL tailings was completed by KCB. The results of this assessment were based on a sample obtained from an early metallurgical selection program using an existing LOM sample and are considered to be appropriate for a feasibility-level study. When samples from subsequent metallurgical programs become available and if observed to be finer grained than the ones used for the current design, more testing may be required. Tailings characteristics are summarized in Table 20-3.

Figure 20-4: End of Period Stockpile Views



Note: Figure prepared by SRK, 2016. Note: LG = low grade, MG = medium grade, HG = high grade.

Figure 20-5: Proposed TSF Layout



Note: Figure prepared by Klohn Crippen Berger, 2016.

Table 20-3: Tailings Characteristics

Area	Finding
Material sizing	Fine-grained materials with a fines content (passing sieve No. 200) of 97.6%.
Unified Soil Classification System (USCS) classification	Classified as low plasticity clayey silts (CL-ML) with a clay-sized particles content of 13%.
Mineralogy	Quartz and muscovite; quartz content ranges between 63% and 72% and the muscovite content ranges between 18% and 21%.
Total suspended solids	Decreases in tailings decant water by 72% after 24 hours of deposition
Settling results	Initial solids density of 63.7% and an initial dry density of 1.06 t/m^3
Compression tests	Conducted on GFL slurried tailings under loads ranging from approximately 1.4 kPa to 3,200 kPa in a standard oedometer. Results indicated reasonably good compression behaviour with an estimated average compressibility index (Cc) of 0.16, and voids ratio ranging between 1.06 and 0.62 for the stress range expected in the TSF. The estimated voids ratio is within the range of values estimated from KCB's experience with other conventional tailings facilities for gold mines.
Void ratio	A long term deposited void ratio of 1.0, equivalent to a deposited dry density of 1.35 t/m^3 can be expected. This value is based on KCB's experience at similar gold tailings sites.
One-dimensional finite difference consolidation modelling	Tailings exhibit a fast consolidation behaviour. The solids content will increase to a density of 75% by the end of the second year of deposition. On-going deposition will provide a minor increase in the solids content, up to 78% solids at the end of operations. These densities are equivalent to average, estimated void ratios of 0.9 and 0.77; for the 2 years and end of deposition scenarios, respectively. The models predict higher densities than those assumed and used for design; however, the use of lower densities than predicted by the models is prudent at this design stage.
Triaxial permeability tests	Tailings are low permeability materials with low hydraulic conductivity values in the order of 2 x 10-8 m/s, even under very low stress levels.
FSConsol consolidation models	Deposited average hydraulic conductivities range between 1.5 x 10-8 m/s at the bottom of the basin and 4 x 10-8 m/s near the surface.

Geochemical testwork on the tailings has characterized them as non-acid generating. There may be elevated levels of total suspended solids (TSS) requiring settlement, and there will be elevated levels of cyanide deposited in the tailings. For the purposes of the TSF design, it was assumed that cyanide will report to the TSF at approximately 1 ppm, and natural degradation from sunlight is predicted to reduce this level to 0.4 ppm. With rainfall and other inflows, the cyanide level in the decant water is expected to be below the discharge limit of 0.1 ppm. Additional modelling and data will be required to confirm that the discharge criteria will be met.

20.4.4 Design Basis

The key design assumptions are summarized in Table 20-4.

Table 20-4: TSF Key Design Assumptions

Item	Value/Comment	
Total available tailings (Mt)	14.76	
Tailings Required for Underground Paste Fill (Mt)	2.61	
Tailings storage requirement at TSF (Mt)	12.15	
Mine life (years)	14	
Detox tailings water CN concentration (ppm)	1.0	
Disposal technology	Conventional slurry tailings deposition at 55% solids density	
Additional storage requirements	• Sludge from water treatment plant (WTP) produced at 4 m^3/h • Sediments removed from ponds located at surface mine infrastructure area will be delivered to the TSF at 8 m^3/h • Storage of up to 15,000 m^3 of excess water from WTP, when operating capacity of plant is exceeded during extreme events	
Design guidelines	Dam Safety Guidelines - Canadian Dam Association (CDA)	
Dam consequence classification	Extreme	
Inflow design flood (IDF)	Probable maximum flood (PMF)	
Design earthquake	1:10 000 years or Maximum Credible Earthquake (MCE)	
Stability criteria	$FoS_{min} \geq 1.5$	End of construction and staged construction conditions
	$FoS_{min} \geq 1.5$	Long term static conditions
	$FoS_{min} \geq 1.0$	Long term pseudo static conditions
	$FoS_{min} \geq 1.2$	Rapid drawdown (For polishing pond)
Runoff control	Runoff diverted with lined diversion channels designed for the 1:100 year storm	
Operating decant pond volume (m^3)	100,000 m^3 For water clarification and floating of reclaim barge	
Water reclaim	• Water reclaim with pumping barge on decant pond • 100% of tailings transport water reclaimed to the plant • Excess water volumes reclaimed to WTP for treatment prior to discharge	
IDF management	• During construction and operations: 100% storage and zero release • At closure: routed and discharged through closure spillway	
Freeboard (m)	2 m above IDF elevation (minimum)	

20.4.5 Design

A starter dam will be initially constructed to elevation 1,462 m to store start-up water for the mill, to create sufficient storage for the tailings in the first year of operation, and to safely contain the probable maximum flood (PMF). The TSF dam will be raised continuously, using the downstream raise method, throughout the service life, until reaching the ultimate elevation at 1,493 masl. Each dam raise will be completed at least one year before the maximum tailings pond elevation required each year; currently dam raises are contemplated at Years 0, 2, 5, 10 and 14 (ultimate).

During operations, the tailings dam crest must be at least 2 m above the predicted intensity–duration–frequency (IDF) storm for the following year. At start-up, the freeboard exceeds this value to account for uncertainties in hydrology and tailings properties during operations.

The tailings dam will be an earth-and-rock-fill structure constructed with a maximum dam height of 63 m measured at the dam centre line. The ultimate dam will have a crest width of 6 m and a length of 700 m at final grade. The dam slopes will be benched for access with inter-bench slopes of 2H:1V, and an overall slope of about 2.4H:1V. As a contingency, the starter dam will have 2.5H:1V inter-bench slopes resulting in an overall slope of 3H:1V.

The body of the dam will be zoned with random sandstone rock-fill forming the downstream shell and compacted saprolite forming a bedding layer for the upstream geosynthetic liner. Fine and coarse filter zones will separate the rock-fill and soil bedding layer to prevent piping. To restrict seepage, the tailings dam and the first 200 m of the tailings basin upstream of the dam toe may be lined with a geomembrane. The liner will be laid directly onto the basin soils, which will be graded, re-compacted and smoothed prior to liner placement. The net hydraulic conductivity of the liner system will be further reduced by the low permeability of the tailings deposited on top of the liner.

It is expected that any contaminant loadings in potential seepage will be attenuated by geochemical interactions with the saprolite soils and any seepage reporting to the seepage collection pond downstream of the TSF will be captured and reclaimed to the decant pond. As a contingency measure, groundwater pump-back wells are included in the design along the dam toe to reclaim any seepage flowing into the transition zone and fractured bedrock. The need for these pressure relief wells needs to be confirmed as part of the recommended hydrogeological study.

The first stage of diversion channels will be constructed to convey non-contact runoff water around the TSF during start up and for the initial five years. New diversion channels will be constructed at a higher elevation for surface water control from five years to the ultimate condition. The diversion channels are designed to convey the 100 year return period peak runoff flows, and will be lined with a reinforced matt to prevent erosion.

Flow from the diversion channels will be routed into a polishing pond downstream of the tailings dam to facilitate settling of suspended solids prior to discharge to the river downstream. The polishing pond will also be used to capture any sediment runoff generated from TSF construction activities.

20.4.6 Operating Philosophy

A total of 12.15 Mt of tailings will be pumped to the TSF at 55% solids over the mine life. The tailings will be discharged into a drop box located near the east abutment of the TSF. Tailings will then be distributed via spigotting.

The sludge produced from the treatment of contact water from the mine at the WTP will be delivered at a rate of 4 m^3/h and stored in the TSF. Sediments removed from ponds located in the mine infrastructure area will also be stored in the TSF and will be delivered at a rate of 8 m^3/h. Sludge will be discharged sub-aqueously into the decant pond. A dedicated pipeline and pumping system for sludge delivery to the TSF will be provided.

Water will be reclaimed to the process plant by a floating pump barge positioned in the tailings pond at the north end of the TSF. The normal operating volume of the reclaim pond will be 100,000 m^3, or as necessary to achieve adequate clarification of the pond water for re-use in the process plant. The design assumes that excess water volumes will be reclaimed from the start of operations and no water will be accumulated and stored in the early stages.

The water balance calculations show that the average annual excess (surplus) water increases as the production rate and the surface area of the TSF grow over time. The average monthly total excess water rates range from 60 L/s to 94 L/s during the operating period. This water will be reclaimed to the process plant as required; the excess will be delivered to water treatment plants for release to the environment.

20.5 Hollín Borrow Pit

Lundin Gold will need to exploit a borrow pit to provide granular materials for construction and mine backfill, from construction through to mine closure.

The Hollín Borrow Pit will provide the following granular products:

- Rock fill for the TSF and seepage pond walls
- Fine and coarse filter material for the TSF and seepage pond walls
- Structural backfill for site preparation and platforms
- Road base for internal roads
- Coarse gravel, fine gravel and sand for concrete and shotcrete
- Paste plant aggregate for paste backfill for the mine stopes
- CRF aggregate for cemented rock backfill for the mine stopes.

The Hollín Borrow Pit will be exploited as an open pit mine with material extracted by ripping and by using explosives. Rock fill material (ROM) will be delivered direct to the TSF and seepage pond walls; all other materials will be processed through an aggregate plant, which will use screening and crushing/screening to produce the required products (Figure 20-6).

The final Hollín Borrow Pit location for the purposes of the 2016 FS is on top of a plateau that is located northwest of the FDN deposit. The plateau extends south to north, presenting a somewhat irregular surface with a sub-horizontal inclination to the north.

The mining of the Hollín Borrow Pit is expected to be done in sedimentary rocks, mainly siliceous sandstones belonging to the Hollín Formation, but due to the quantity of material required, will extend through the Hollín Formation sandstones into the intrusive rocks of the Zamora Batholith.

20.6 Waste Management

The waste management centre (WMC) was sized to receive waste during operations and manage the waste temporarily until final disposal by an authorized contractor. The WMC was designed to handle waste from one month of operations.

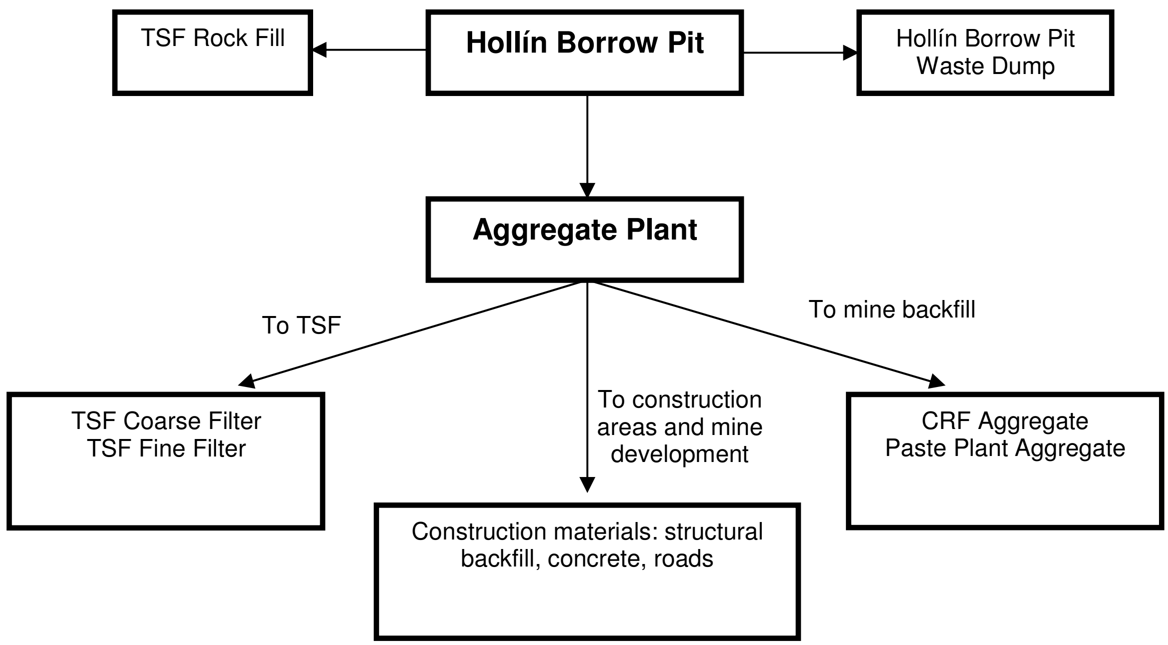
Waste management is based on the following criteria:

- Solid waste management as per the following hierarchy: replace, reduce, reutilize, recycle, eliminate
- Toxicity of chemical waste and disposed reagents will be reduced
- Waste management facilities will have covered stations to separate and temporarily store different types of solid waste
- Generation of hazardous solid waste will be minimized.

The waste management area was designed considering location, capacity, construction considerations and environmental, health and safety criteria. Contention measures were included to prevent spills of hazardous waste.

Hazardous waste will be generated mainly in the industrial processes. Waste is classified as hazardous if it has one or more of the following characteristics (Environmental Regulation AM No. 26): corrosive; reactive; explosive; toxic; flammable; or biologically infectious.

Figure 20-6: Hollín Borrow Pit Block Diagram



Note: Figure prepared by Lundin Gold, 2016.

Lundin Gold is registered with the Ecuadorian Authority as a hazardous waste generator. Before operations start, a hazardous waste-type update study will be carried out, as required by the local law, and the registration will be updated. Contractor taking part in the processing, transport and disposal of waste will have the required technical and environmental permits.

Organic waste materials will be recovered insitu. Organic waste will be removed from the WMC to avoid generation of organic acids and biogas, which require special treatment.

Domestic garbage from the camp and other permanent facilities, such as kitchenettes and offices, will be disposed of in the Project landfill. The landfill will be divided in two modules; one to be used during construction and the other to be used during operations. Garbage will be covered daily with granular material. In general, the cells will receive only domestic and similar garbage; no other type of waste will be deposited. Leachate will be managed through a drainage and collection system for final disposal and treatment at the camp sewage treatment facility.

20.7 Water Management

20.7.1 Hydrological Setting

Climate

The following key items were noted following a literature search and collection of site-specific climate data:

- The annual average temperature is 17.9°C; the maximum temperatures are typically recorded in November and the lowest temperatures in July
- The annual relative humidity is 89.9%, and varies depending on the season, being lower during August–November and higher during December–July
- Wind speeds vary from 0.7–1 m/s, averaging 0.9 m/s at the proposed mine site
- The annual average rainfall for the FDN site is 3,414 mm, with minimum and maximum values of 3,081 mm and 3,652 mm respectively; 73% of the annual rainfall occurs between December–July and 27% falls in the dryer months from August to November
- The 24 hour probable maximum precipitation (PMP) was estimated to be 400 mm using the statistical method developed by the World Meteorological Organization (WMO No. 1045, 2009). Long-duration storms at the FDN site typically consist of several precipitation events separated by short periods of little or no rainfall. The maximum rainfall for long-duration storms (seven days and 30 days) was developed from analyses of historical storms. A 100-year return event over a seven day period was estimated at 335 mm, and 648 mm was estimated for a 30 day period.
- The pan evaporation was estimated at 1,103 mm.

Stream Flow Rates

During the site investigations for the hydrology assessment, flows were measured in local streams around the proposed plant site and TSF areas.

The Machinaza River flood estimates included a hydrologic model (HEC-HMS) and a hydraulic model (HEC-RAS). The outflow of the HEC-HMS model was used as the inflow for the HEC-RAS model. The hydrologic model was run for the following single storm event 24 hour precipitation values: 134.8 mm (100 years), 142.0 mm (200 years), and 151.5 mm (500 years). The 10 year–seven day (dry condition) flows were also estimated.

20.7.2 Groundwater

At a large scale, groundwater flows northwards, following the Machinaza River. At the Project scale, groundwater generally flows towards the Machinaza River, roughly following the overall surface water drainage pattern. In the immediate vicinity of the underground mining area, further local scale variability exists; there have been observations of artesian conditions and the shallow aquifer has a relatively flat-sloping gradient.

In the vicinity of the proposed sites for the process plant and waste dumps, the presence of saprolite at ground surface may act to locally reduce groundwater recharge. In general, the majority of groundwater flow is expected to occur relatively close to ground surface.

The hydrostratigraphic system at the FDN Project is defined based on data collected during field programs in 2009 (Golder Associates), 2011 (Itasca Denver Inc.) and 2015 (SRK, KCB and Aurelian). Data collection is continuing in the areas of the planned mine, process facilities, TSF and Hollín Borrow Pit.

The hydrostratigraphic system consists of multiple lithology-based units, overprinted by geological structures and, in the area of the proposed underground mine, alteration. The bedrock groundwater system is generally considered to be fracture-controlled, with the majority of water occurring in fractures that overprint the different lithologies. The individual lithologies themselves are not considered to have significant primary porosity, or permeability. The lithologies hosting the deposit are expected to have lower permeability than shallower geological formations.

Geological structures overprinting all lithologies can have a highly variable effect on groundwater flow; acting as barriers or conduits for water flow; or having no discernible difference from the host lithology. Alteration affects many of the lithologies, at least in close proximity to the deposit, overprinting and changing host mineralogical assemblages. This in turn can have an effect on groundwater flow, in the form of higher percentages of clay that reduce the hydraulic conductivity.

Interpretations of hydraulic properties were based on over 100 hydraulic tests completed across the site. Rock tests were completed primarily in the area of the proposed underground mine. Rock properties are considered to be representative of the entire Project area, although local variability exists.

Groundwater chemistry data are available for the area of the proposed underground mine and were used to support estimations of water quality that will be discharged from the mining operation. These data were collected as part of hydrogeological investigations completed in 2010 and 2015. Data were collected using best practices for groundwater collection, including purging prior to sample collection; acid preservation of metal samples soon after sample collection, and chain of custody procedures.

Groundwater in the mine area will be actively managed to address mining requirements; inflows to the mine will be reduced or controlled using dewatering wells and dedicated underground infrastructure (i.e. drainage galleries).

Seven dewatering wells are planned. The wells are generally located along a north to south alignment, between the underground mine and the Machinaza River. At least one well will be completed in the area directly over the mine.

Dewatering wells will be located in deepest portions of the Suárez Formation, the formation considered to be most significant in terms of conveying water to the underground mine. Wells will also be drilled into the Mishualli Formation, which underlies the Suárez Formation. Wells are typically designed for pumping rates of about 60 m^3/h (17 L/sec).

Water pumped from the dewatering wells is considered neutral water. This water will be pumped to surface to a contingency settling pond and then discharged to the Machinaza River. In the mine, water will enter via workings and flow into the drainage galleries. All water entering the underground mine will be collected in underground sumps and pumped to the water treatment plant.

A groundwater monitoring program will include monitoring of groundwater chemistry and levels at monitoring wells located across the site. Monitoring well locations will be finalized as mine plans are finalized, construction begins, and access limitations are reduced. Monitoring wells that are not currently in place will be installed as soon as possible after construction starts and sampling will commence as soon as practicable.

20.7.3 Geochemistry

The materials expected to be generated during mining were identified and the geochemical characteristics were determined. Predictions of geochemistry source terms for the mine facilities were used as inputs to the site-wide water quality model to determine if material generated from mine operations might create an unintended environmental impact.

The geochemistry source terms include chemical loads generated by the Hollín Borrow Pit; borrow material crushing area; TSF; mine WRSF, and stockpile areas; borrow pit WRSF; and the mine.

Based on an appropriate geochemical characterization program, most of the waste rock from the development and operation of the borrow pit is classified as PAG, and could leach acidic solutions, metals, and sulphates. The fine-grained formations of the Hollín Formation exposed in the Hollín Borrow Pit and stored in the borrow pit WRSF will have some sulphide content. Drainage from the borrow pit and borrow pit WRSF may be acidic. More detailed studies and modelling are required to characterize the lithologies that would be stored in the borrow pit WRSF. At closure, the WRSF would be covered and groundwater monitoring will be conducted.

Cemented tailings are not acid generating if Hollín Formation sandstone is used as the aggregate. Aggregate from porphyry rock used in cemented tailings has an uncertain ARD potential. Sandstone from the Hollín borrow pit generally has some ARD potential but the potential volume of acid production is limited by the low sulphide content (<0.1%). Tailings are not expected to be acid generating.

20.7.4 Surface Water Quality Assessment

Water quality data are available from June 2008 until June 2015. Surface water quality has been monitored at least 49 monitoring sites located within the Project vicinity. The monitoring sites cover the Blanco, Machinaza and Zarza River basins including the main channels, tributaries, and small streams (upstream and downstream from possible sources of contamination). Parameters which exceeded the Ecuadorian reference limits (Irrigation and Livestock) in the Machinaza River included pH, fecal coliform counts, and metal concentrations (aluminium, iron, cobalt, hexavalent chromium, lead, manganese, zinc and selenium). A number of monitoring sites were sampled repeatedly to provide indications of temporal (seasonal) and spatial variability.

20.7.5 Water Types Requiring Management

Water types that will require management are summarized in Table 20-5.

There are four major management infrastructure types for each water type:

- Diversion works: To divert non-contact storm water, and prevent it from being contaminated by the site during construction and operations. Includes riprap interception works, lined channels, creek riprap discharge works, and slope drainage systems for mass earthworks
- Contact water works: To manage affected and unaffected water during the construction and operations phases. Comprises sumps, water management ponds, chutes (steep slope conduits), energy dissipaters, water treatment plants, pumping systems and emergency discharge works to natural water courses
- Neutral water works: To deal with groundwater from the dewatering wells above the deposit. Consists of a pumping system, a water management pond and a discharge to the Machinaza River
- Secondary and minor drainage networks to be located within the facilities for non-contact and contact water, including small sumps, downspouts, and minor collecting pipes. These works have not been designed to a feasibility level.

Table 20-5: Water Types Requiring Water Management

Water Type	Definition	Examples
Non-contact	Water (either runoff from precipitation or flowing in natural streams) whose quality is not impacted by the project infrastructure and activities.	Off-site rainfall and runoff Runoff diverted by contour channels before reaching a specific area of the Project Runoff collected on site without impact on its quality (for example runoff generated on roofs and kept isolated by means of downspouts and pipes, subsequently returned to a natural stream).
Contact	Water that has come in contact with infrastructure, facilities, materials and equipment, or Project activities that might have had its physical, biological, and/or chemical quality impacted. Depending on the water quality impact, contact water can be categorized as "affected" or "unaffected".	
	Unaffected: Water that is likely to have had a sediment load increase but not subject to chemical/biological impact requiring treatment other than total suspended solids (TSS) removal in order to meet water quality regulations. Requires TSS removal only, prior to discharge to a natural water course; no water treatment plant is required.	Runoff contacting construction areas where earthworks and deforestation activities may increase the sediment load. Runoff contacting Project site areas during operations but not impacted by the mining activity itself.
	Affected : Must be sent to a water management pond and a water treatment plant (WTP) prior to being discharged to the environment.	Underground mine dewatering. Water contacting tailings. Runoff from the crushing area stockpiles, mine waste dump, mine surface infrastructure, process plant, borrow pit, aggregate plant and landfill.
Neutral	Groundwater collected above the orebody at the underground mine. Requires TSS removal and/or primary treatment only (depending on the quality parameters) prior to being discharged into the Machinaza River.	Water from dewatering wells.

20.7.6 Water Infrastructure Design

The hydrological and hydraulic design criteria for water management are as indicated in Table 20-6.

Table 20-6: Hydrological and Hydraulic Design Criteria

Item	Unit	Value
10 year 24 h precipitation	mm	110.3
25 year 24 h precipitation	mm	119.4
200 year 24 h precipitation	mm	142.0
Runoff coefficient	—	0.8
Return frequency for sedimentation	years	10
Return frequency for flood routing:		
Temporary works	years	25
Permanent works - canals	years	200
Permanent works - SWMP	years	variable
Diversion works at TSF	years	1,000
Polishing Pond at TSF	years	200
Starter Tailings Dam	years	1,000
Diversion/contact channel slope	m/m	0.005
Diversion/contact channel lining	—	HDPE membrane
Diversion/contact channel roughness coefficient	—	0.013
Minimum channel freeboard	m	0.4
Channel side slopes	m/m	1.0
Maximum chute water depth/diameter ratio	m/m	0.5
Chute material	—	HDPE pipe
Chute roughness coefficient	—	0.012
SWMP dead storage depth	m	1.5
SWMP particle settling velocity	m/s	0.00002
SWMP freeboard	m	0.5

Water Discharge Quality Criteria

There are two governing regulatory water quality criteria for effluent discharges from any industrial facility in Ecuador:

- Effluent discharge limits
- Instream flow water quality standards.

The effluent discharge limit set by the Ecuadorian Government is the “end-of-pipe” regulatory requirement that must met under any and all conditions. The instream flow quality standard is based on the water use in the vicinity of a project, and is used to calculate the levels of chemical constituents that may be discharged to the receiving water body. The primary water use in the vicinity of the Project for the Machinaza River is irrigation and livestock.

All water discharges into natural water courses, from water management ponds, water treatment plants or diversion channels, must comply with the Ecuadorian Government water quality criteria requirements. These values apply for construction and operating phases of FDN and make a distinction between discharge into the Machinaza River and discharges to other streams (based on the background water quality).

Water Treatment Plants

Six WTPs are planned:

- Two potable water treatment plants: one will be located at the camp site and the other at the process plant
- A sewage treatment plant will be located at the camp site. The process plant sewage will be managed using septic tanks
- The main effluent water treatment plant (MWTP) will be located at the process plant site and will treat most of the affected contact water from the site
- The Hollín Borrow Pit water treatment plant will be located close to the aggregate plant and will treat affected contact water from the borrow pit area
- An existing compact plant at the site will be moved to the mine portal area and used during the first year of mine dewatering.

The water treatment plants have been sized based on the wettest month in a one-in-100 wet year.

The main effluent water treatment plant will be located at the process plant site and will treat most of the affected contact water from the site. The main characteristics of this plant are:

- It will treat affected contact water from the mine dewatering process, runoff from the process plant and mine surface infrastructure platforms, and runoff from the ROM stockpile, mine waste dump and CRF plant areas
- The TSF decant will be mixed with the discharge from this plant
- The high-density sludge (HDS) system will include low sulphate and high sulphate reactors, a clarifier and an ultrafiltration unit (UF)
- It will have a sludge line to pump sludge to the TSF
- The operating capacity will be of the order of 850 m^3/h (including the TSF decant). Through optimization, it was identified that the flows from the TSF and process plant could meet discharge criteria without full treatment. This could result in the capacity of the MWTP being reduced by approximately 40%.

It is expected that mine dewatering during the mine development stage will generate the first affected contact flows requiring treatment, hence the MWTP must be operational during the construction phase. A contingency plan was developed for the site in case of failure of the MWTP.

The Hollín Borrow Pit water treatment plant will be located close to the aggregate plant and will treat affected contact water from the borrow pit area. The operating capacity will be approximately 180 m^3/h. The feed chemistry for the Hollín Borrow Pit water treatment plant has not yet been finalized. However, it is assumed that it will use the same process as the MWTP, but on a smaller scale.

The existing compact plant uses dissolved air flotation (DAF) technology, capable of removal of nitrates, nitrites, TSS, oil and grease, and has an operating capacity of 79 m^3/h.

Water Management Ponds

The water management ponds have been designed as rectangular shapes, with an approximate relationship length/width of 4 to 5:1 in order to prevent short circuits. The ponds will be lined with HDPE membrane in order to prevent seepage. Side slopes of 3:1 (H:V) have been used to ensure stability.

Emergency works are not expected to operate during the LOM for affected contact water ponds. Unaffected water management pond emergency spillways are expected to operate, once or twice during the life of the mine. Emergency works include gravitational structures (spillways), and for the mine waste dump pond an emergency line to the sludge tank of the MWTP.

At this stage of engineering it is planned to remove sediment using perforated (slotted) pipes placed at the base of each pond. Around 33,000 m^3 of sediment is expected annually. Estimating four operations per day per truck, three vacuum trucks operating year round will be required for the pond sediment removal process. During the operations phase, sediments collected from the ponds will be transported either to the sludge tank located at the MWTP or directly to the TSF basin. During the next stage of engineering an alternative to this disposal will be considered based on field data.

During the early stages of construction neither the MWTP nor the TSF will be available; removed sediment will be placed close to the ponds on the upper side allowing seepage to flow back into the pond.

Diversion Channels

Diversion channels will divert non-contact water around the site. These works will consist of diversion channels with interception structures at the upstream section and discharge structures at the downstream end.

Trapezoidal diversion channels were designed with lining and 0.5% minimum longitudinal slope in order to prevent sediment settling. No bottom width will be narrower than 0.6 m to facilitate construction, and channel side slopes will be 1:1 (H:V). A 1 m wide service/maintenance strip will be kept alongside the channel to allow access during the construction and operation phases.

Interception structures will consist of a riprap revetment at the beginning of the diversion channel and at the downstream end of the creek to be diverted, and riprap protection at any sharp bends or changes in gradient. Discharge structures will consist of a riprap revetment located at the downstream end of the channel and at the receiving creek bed.

20.7.7 Facilities Requiring Water Management

The areas that will require water management include the following.

Underground Mine

Mine access will be via a twin decline system, located in an excavated area of a hillside on the right bank of the Machinaza River, at the east edge of the Project site in order to maximize overburden and minimize groundwater inflows. The declines will pass under the river, and the expected pillar will be approximately 150 m. Inflows of as much as approximately 14,000 m^3/day are expected. Groundwater in the mine will be managed by a combination of deep pumping wells, drainage galleries and sumps located inside the mine. Five to seven pumping wells are envisaged, deep enough to reach the bottom level of the mine, and will be located between the Machinaza River and the deposit. Drainage galleries will be excavated at several elevations located to the west of the deposit. From these galleries and drilling stations located at the east side of the mine, drilling will be carried out to capture and collect the groundwater before it reaches the mine.

Mining operations will produce neutral water (water which will be intercepted before reaching the mine working areas), and affected contact water (water which will be collected in the mine).

Mine Waste Rock Storage Facilities

Waste rock from the main decline, the internal ramps and accesses, ventilation shafts and other infrastructure is expected to be NAG. However, mining development west of the East Fault is predicted to produce waste rock containing sulphides, and is considered to be PAG. NAG waste rock produced during the construction phase could be used as construction material (geomechanical properties permitting). No PAG waste rock will be used as construction material. Drainage from the WRSF is defined as affected contact water, until the facility is capped and stabilized.

Monitoring/modelling of groundwater and the closed WRSF will be included in the final closure plan.

Stockpiles

Based on the results from geochemical analysis and field data, rock meeting cut-off grade is typically PAG, which could result in drainage with pH values in the range of 2 to 3, but sometimes less than 2. Arsenic, cobalt, copper, magnesium and zinc have been detected under acidic leaching conditions, in concentrations ranging from 1 mg/L to 100 mg/L. Under non-acidic leaching conditions, concentrations of metals were lower. Drainage from the stockpiles is classified as affected contact water. More detailed assessment and associated modelling is required to verify this. However, if TSS levels exceed the discharge limits, it is possible to manage the issue through the combination of increasing the retention period in the pond to allow for settling or/and use of flocculants and coagulant if required.

Water from the TSF pond will be used as make-up water for the process plant. TSF pond water in excess of the make-up water requirement will be sent to the water treatment plant or discharged to the environment, depending on the water quality. It is possible that water from the TSF will only exceed in total cyanide (TSS will be controlled in the TSF), in which case this water could bypass the main water treatment plant and mix with effluent downstream prior to discharge to the environment. It is also proposed that a turbidity meter be installed in the TSF decant pond and the contingency plan is that water exceeding the TSS parameter will be returned to the decant pond.

Topsoil Storage and Stockpile of Excavation Surplus

Stockpiling of topsoil and excavation surplus (defined as material with geotechnical characteristics which mean that it is not suitable for construction uses and has to be removed from the areas where construction is taking place) will be done west of the process plant. This will require deforestation. It has been estimated that a total topsoil stockpile area of the order of 12 ha to 15 ha will be required with a maximum stockpile height of 2.5 m, in order to prevent compaction. This will have a gentle slope which will minimize erosion. The excavation surplus stockpile is expected to require an area of 15 ha to 17 ha. The 10 m layers in this stockpile will be compacted to minimize the footprint. Drainage from this area is defined as unaffected water.

Hollín Borrow Pit

The Hollín Borrow Pit in the final stage of development will cover an area of 15 ha with elevations ranging from 1,675 masl to 1,710 masl. Drainage from the borrow pit is defined as affected contact water.

The water quality properties for this area are still under assessment. Currently it is considered that the water will be collected and treated in the Hollín Borrow Pit water treatment plant.

Hollín Borrow Pit Waste Rock Facility

The facility will include the top layer from the borrow pit, rejected material (out of specification), hillside cut material for the crushing area, and cut material from road construction. The volumes have been estimated considering 33% bulking. Drainage from the borrow pit waste dump is defined as affected contact water.

A more detailed design for the system to drain the water from the borrow pit and borrow pit waste dump will be carried out in the next stage of the Project. An allowance for this system has been included in the capital expenditures in Section 21.

Aggregate Plant

In order to meet the material specifications for the project, the aggregate plant will crush, screen and wash material from the borrow pit. Drainage from the aggregate plant is defined as affected contact water.

Paste Plant

The paste plant will provide the backfill material required for mine stability during operations. This will be the main supplier of cemented backfill for the mine. It will be located on the east area of the site, on the right bank of the Machinaza River.

The tailings required for paste will be supplied from the process plant. The water content will be reduced to 25% by a thickener and three filters. Make-up water for the paste plant and the process plant will be provided from the thickener. The underground distribution system will be rinsed before and after paste filling operations. Up to 25 m^3 of water will be required during this rinsing process; this water will be diverted to the underground dewatering pumps.

Surface drainage from the paste plant area is defined as contact unaffected water.

Accommodation Camp

Drainage from the camp is defined as contact unaffected water.

Waste Management Area

The waste management area will include a compost area, a landfill and a temporary waste storage area. The landfill will be located west of the excavation surplus stockpile. The runoff from the landfill will be sent to a management pond. The leachate will be collected in a sump, pumped out by a truck as needed and sent for treatment.

20.7.8 Water Balance Model

A water balance model was constructed that tracks water from the sources, through collection and conveyance systems, usage, storage, treatment and discharge to the environment.

The water management plan (WMP) simulated in the water balance model is based on the following design criteria.

- Ponds:
  - Sedimentation: 1 in 10 year flood
  - Normal operation: 1 in 100 year wet year, wettest month of the year (May)
  - Normal operation of the pumping system: 1 in 10 year flood
  - Containment of inflow with the emergency pumping system: 1 in 500 year, 24 hour flood.
- Main water treatment plant (MWTP):
  - 1 in 100 year wet year, wettest month.

The following are the main assumptions for the water balance model:

- Precipitation on the process plant surface area is considered contact water
- A runoff coefficient of 0.8 (provided by KCB) has been assumed for non-contact and contact water (camp, waste storage dump, mine surface infrastructure, process plant area, TSF beach, TSF area)
- Water retained in tailings, paste backfill and cement backfill is assumed to be outflow from the system
- The surplus water from the TSF is considered to be contact water and will be used as make-up water for the process plant; any excess will be sent to the MWTP
- Zero losses are assumed in the camp; all domestic non-potable water used will be treated and discharged to the environment
- During operations non-contact water will be routed through non-contact ditches/infrastructure before being discharged to natural creeks or rivers. Non-contact water flows are not included in the water balance, but were evaluated as part of the drainage system design.

The FDN water balance is a positive water balance with significantly more water coming into the system than required for operations. Excess water on the site will be managed and treated as required prior to discharge to the environment.

The results of the water balance model demonstrate that the proposed water management plan at the site is feasible. The water balance model evaluates the system on a daily and monthly basis over the life of the mine (2021 to 2031), as well as for an average year, dry year and wet year. The model also assesses on an hourly basis the operation of the ponds and pumping system to the MWTP during a one-in-24 hour design storm.

20.7.9 Surface Water Quality Model

The water quality model focused on parameters of concern identified from the surface water quality assessment (aluminium, arsenic, copper, cobalt, cyanide (CN), iron, magnesium, potassium, manganese, lead, selenium and zinc).

The water quality model is based on the water balance model flowsheet. Except for cyanide (which will be subject to natural degradation in the pond) the water quality model is based on conservative mass balance calculations of chemical loading and water balance model results.

The main water quality model inputs are:

- Background water quality
- Geochemical source terms
- Process plant effluent.

The parameters modelled were selected based a combination of factors including:

- Availability of the parameter in all the source terms
- Total metals were not provided in the source terms for the TSF, Hollín borrow pit, waste rock dump and underground mine, in these cases total metals concentrations were assumed to be dissolved metals concentrations
- The majority of the general parameters were not available for the source terms for the TSF, Hollín borrow pit, waste rock dump and underground mine with the exception of sulphate
- Total dissolved solids (TDS) for the TSF, Hollín borrow pit, waste rock dump and underground mine were not available; the TDS concentration was assumed to be the sum of the metal cations (dissolved metals) in the source term.

The main results from the water quality model are the excess flows which require treatment prior to discharge to the environment. The flows which required treatment prior to discharge are:

- Underground mine dewatering flow. Parameters from underground mine dewatering which exceed discharge limits that will require treatment include As and Fe; these chemical parameters will be treated to meet the discharge limits
- TSF supernatant. Sulphate, TDS and metal parameter concentrations from the TSF will be below discharge limits
- Waste rock dump and stockpile runoff. Parameters which exceed discharge limits include TDS, sulphate, aluminium, arsenic, cobalt, copper, iron and manganese; these chemical parameters will be treated to meet the discharge limits. The waste rock dump and stockpile runoff is the most impacted water stream on site, in terms of the number of water quality parameters requiring treatment and the concentrations
- Process plant and mine infrastructure area runoff. Hexavalent chromium (Cr^{6+}) is the only parameter which exceeds the discharge limit and will require treatment from the process plant contact pond

These flows will all be routed to the MWTP for treatment prior to discharge to the environment. There may be scope during detailed studies to optimize the WMP, and have the TSF discharge bypass the MWTP.

The results of the water quality model demonstrate that the proposed WMP for the site is feasible and will meet regulatory requirements for discharge to the receiving water bodies. The primary water use designated for the Machinaza River, due to existing water quality issues, is irrigation and livestock. The model demonstrates that the water quality at the discharge point will meet the irrigation and livestock water quality criteria for the Machinaza River, which is the primary receiving environment.

The preferable water source for make-up water for the process plant is the TSF supernatant which meets the water quality requirement for use. In an emergency, the underground mine dewatering flow can be used. This meets the minimum requirement for water use in the process plant.

20.7.10 Water Management Plan

Unaffected Contact Water

It is expected that most of the contact water that will be generated during construction will be unaffected, mainly related to site preparation works (clearing and grubbing, deforestation, topsoil removal, earthworks and civil works). Unaffected contact water requires TSS removal before being discharged into the environment. To accomplish this, several water management ponds that will be used in the operations phase will be installed early in construction. The construction contractors must ensure that TSS removal meets water quality criteria and a monitoring program will be instituted to demonstrate this.

Affected Contact Water Management

The MWTP will treat the largest volume of affected contact water and the worst quality water, because it will receive runoff from the waste dump and the ROM stockpile, both with ARD potential. During normal operations, pond water will be pumped to the MWTP. If there is an extreme hydrological event, two of the ponds will have an emergency pumping system to convey excess water to the sludge tank and from there to the TSF (where there is storage capacity). For extreme events flow from the TSF to the MWTP may be stopped to ensure that the MWTP operates within its design flow range. After the rainfall event the water stored in the TSF will be pumped back to the MWTP and normal operations water management will be restored.

20.8 Closure Plan

Closure planning has been undertaken to a conceptual level, and will be continually updated throughout the Project life. The conceptual Closure Plan has been developed in accordance with Article 125 of the Environmental Regulations for Mining Activities (RAAM) and Title X of the Mining Safety Regulations.

The conceptual Closure Plan defines the closure objectives and goals. The main principle will be passive or preventive care, which requires minimal management and maintenance after closure. The closure activities cover closure aspects related to environmental factors such as soil, air and water which are directly related to the community health and safety. Aspects related to economic and cultural dynamics of the communities have not been considered in the plan.

The definitive closure will be done in accordance with Art. 124 of RAAM, which requires that the definitive Closure Plan must be presented two years prior to cessation of operations.

A Social Impact Management Plan (SIMP) will be developed and implemented to manage socioeconomic impacts of the Project (both positive and negative) throughout all phases of the Project, including closure.

20.8.1 Closure Objectives and Activities

Lundin Gold´s closure objectives include the following four goals:

- Protect the health and safety of communities
- Prevent, minimize and mitigate any adverse environmental impacts
- Restore disturbed areas to the condition in which they can be self-sustaining, making possible the land use established in the Closure Plan

- Ensure the chemical, physical and hydrological stability of the waste storage areas in the long term, including the TSF and other mine facilities.

The expected closure activities that will be undertaken include:

- Dismantling of buildings, equipment, machinery, services and other surface and underground infrastructure
- Chemical and physical stabilization of the waste rock dumps and ore stockpiles, and the tailings storage area
- Rehabilitation of surfaces that have been affected by earthmoving and storage of ore and waste rock by reshaping and revegetation
- Remediation of contaminated soil
- Closure of access points and filling in of underground openings
- Monitoring.

Table 20-7 summarizes the conceptual closure activities.

20.8.2 Closure Considerations

Specific, conservative criteria that have been applied to ensure that the closure objectives can be met without requiring active treatments in the long term include:

- The TSF design has been cataloged as a very high risk infrastructure in accordance with the Canadian Dam Association (CDA) and takes into account the maximum probable seismic event (recurrence every 10,000 years) and a storm event with recurrence every 10,000 years over 30 continuous days
- PAG material cannot be used for erosion control works; such as rip-rap, sediment containing walls, or for building the TSF or access roads
- The storage areas for PAG material will be conservatively designed in order to maximize environmental protection. This includes using peak concentrations for storage of flooded rock or infiltration water from tailings. For ore and PAG stockpile areas, stable flows of metal leach/acid rock drainage (ML/ARD) discharges will be used in the design
- Studies will be completed to establish the mix for paste backfill to allow a high percentage of tailings in the mine backfill
- Piping used to transport hazardous materials will be built on the surface (not buried)

Table 20-7: Closure Activities

Area	Planned Work
Underground Mine	The underground mine will be flooded and access (declines and shafts) will be closed to prevent metals leaching (ML), ARD and access. Stopes will be backfilled during operation, and the only voids will be the main drifts, the access ramps and ventilation shafts. Ventilation shafts will be sealed using a concrete plug and profiling with fill. Access to the two portals will be sealed off by demolishing the portals and building a berm using waste rock, completely covering the area. In order to minimize the potential for ML/ARD generation inside the mine, internal and external underground water pumping will be stopped, allowing the mine to flood until reaching the natural phreatic level. This will minimize contact between oxygen and PAG material exposed inside the workings. Sealing off the portals and access points to the underground mine will impede the flow of water out of the mine. It is expected that the underground water level will be lower than the portal ground level, and therefore, no surface drainage is anticipated from the mine. It is assumed that management of effluents from the underground mine and active treatment will not be required in the post-closure phase. The portal areas will be reshaped and rehabilitated. Any soil that has been contaminated with hydrocarbons or other material will be removed from the site and replaced with clean soil. Compacted soils will be ripped to a depth of 0.4 m and aerated to facilitate revegetation with native plant species. Disturbed areas around the ventilation shafts will also be reshaped and rehabilitated. Provided that there is sufficient topsoil available, this will be used for the final cover because it has a higher organic content. Erosion control measures will be incorporated into reshaping to minimize effects on surface water bodies.
WRSFs	In the WRSF areas, no slopes will be reduced for closure, the final slopes that result from the operation will be left, allowing gradual sliding of the dump materials. The material that falls will accumulate at the toe of the slope. As a result, the slope angle will reduce over time, typically to between 22° and 23°, and the toe will advance to around 60% of the deposit height (height of the lower bench). The area at the toe of the WRSFs receiving fallen material will be restricted by building a berm at a distance from the toe equivalent to 60% of the WRSF height. The berms will be built using mine waste with a minimum height of 1.5 m and a slope of no less than 33° or 1.5:1.0 (H:V). These designs will be reviewed during the detail design phase to confirm the effectiveness. Designs may be optimized based on the materials available and the final geometry of the WRSFs. PAG material in waste rock dumps will be covered with the material to be determined by pilot tests using different types of covers to improve the revegetation process and minimize contact between water and PAG material. If ML/ARD is found, active water treatment systems will be operated until after the closure stage, after that passive systems will be installed to neutralize the pH (e.g. open limestone channels, anoxic drains in limestone). Perimeter and bench top ditches will be built to capture water that may come into contact with the waste rock and minimize the possibility of infiltration and accumulation of water in the WRSFs. WRSFs will be covered; the type of cover, materials and geometry will be defined following pilot tests during operations.
Hollín Borrow Pit	It is estimated that the total Hollín Borrow Pit area will be between 20 ha and 27 ha with depths between 40 m and 70 m at the end of the Project life. Mobile equipment (drills, trucks, backhoes) and stationary equipment (pumping equipment, electricity generators) will be removed from the area. During operations the last cut on each bench will use a lesser slope with a higher safety factor. Fences and signs will be placed to prevent access to areas where mining operations have taken place. No work will be done on slopes or benches for water management and erosion control; these processes will allow material to accumulate on the benches and will support vegetation. The natural conditions in the borrow pit area are topsoil consisting of peat or organic material directly deposited on the siliceous sandstone; therefore, an intermediate layer of soil is not required for revegetation. Erosion control and drainage channels will be installed in the borrow pit floor area. This water will be returned to the nearest water resource by gravity flow. The borrow pit is a non-mineralized area; hence ML/ARD is not expected to be generated. The borrow pit is expected to be geochemically stable over the course of the years and will not require specific closure measures for this factor.

Area	Planned Work
TSF	All of the elements of the reclaim water pumping system from the polishing pond will be removed once the pond water quality has chemically stabilized. It is estimated that this chemical stabilization will take approximately two years. One of the major activities of active closure will be the ongoing operation of the water treatment plant and detox system until the cyanide concentration in the tailings decant pond reaches trace levels. The TSF operating facilities and services that are not required for post-closure stage will be dismantled. The possibility of demolishing the settling pond wall will be analyzed in future studies. In order to do this, it will be necessary to clean out any remaining sediment from the pond. This reclaimed sediment will be deposited in the TSF. Rock fill material from the settling pond wall will be used to reinforce the toe of the tailings dam to increase stability during closure. Geotechnical monitoring devices such as inclinometers used to identify possible differential movements and piezometers used to analyze pore pressure will remain in operation for the required five-year monitoring period. These will be installed during construction. The design will be verified during the closure period to take into account the actual conditions of the tailings (density, consolidation of dry tailings) to verify that the design estimates are accurate. Tailings from the GFL process produced during the first five years of mining activities will have a sulphur content of around 0.32% and a low ML/ARD potential (NP/AP of 2.9 and CO_3 NP/AP of 1.4). The net acid generation tests at pH of 4.5 and 7 were both 0 kg H_2SO_4/t. Dynamic wet cell testing (HCT) is underway to determine how the tailings will behave over time. Cyanide destruction tests indicated that the cyanide concentration in the tailings sent to the TSF will be <1 ppm. Residual CN_{WAD} in tailings was only 0.02 mg/L. There will be two zones once the tailings storage facility is closed, a dry zone where the tailings beaches have been established, and a wet zone made up of a pond. The wet area will be monitored during the post-closure stage. It has been estimated that the water that filters through the tailings dam wall will comply with discharge parameters. Chemical analyses will be done during operations to confirm the absorption capacity of the saprolite under the dam wall. The cyanide destruction plant will remain in operation to treat any filtrations that are outside the permitted discharge limits until the water in the polishing pond is chemically stabilized. Cyanide will naturally decay in the tailings areas through ultraviolet action. Also fresh rain water that enters the polishing pond will provide natural dilution. Therefore, it is expected that two years after operations cease, the cyanide concentrations in the water will be at trace levels. A closure spillway will be dug in the east abutment of the TSF with capacity to divert the probable maximum flow. The channel lining will be decided based on the materials available at the time of closure. The pond formed during operations will be kept away from the tailings dam wall by controlling the formation and the consolidation of the dry tailings areas. Water from this pond will be treated in the water treatment plant and the cyanide destruction plant until chemical stability is achieved. The perimeter deviation channels will remain and will continue to function in the post-closure stage to minimize the amount of water that comes into contact with the tailings. After removing sediment from the TSF settling pond and confirming that the water meets discharge criteria, the pond wall will be demolished and the water will drain into the environment. The infiltration and polishing ponds will be maintained until the final closure stage. Monitoring wells and return pumping wells downstream and around the TSF will be kept in place until the Environmental Authority has approved the definitive Closure Plan.
Buildings	All buildings and surface and underground infrastructure will be dismantled, the debris will be classified and sent to final disposition or for recycling. All concrete structures will be demolished to a depth of 0.4 m below ground level and the debris will be sent to the mine waste dump, borrow pit waste dump or the borrow pit areas. Compacted soils will be ripped to a depth of 0.4 m below ground level to achieve suitable conditions for revegetation and rehabilitation.
Revegetation and Rehabilitation	Rehabilitated areas will be revegetated. Native seedlings grown in the plant nurseries will be used. Fertilizers will be avoided where possible. Areas with slopes of more than 50% will be covered with jute screens to minimize erosion. Rehabilitation of aquatic habitats will focus on reshaping the diverted natural drainage systems, rehabilitation of river-bank areas, and reconstruction of micro-habitats. Water courses will be stabilized to minimize erosion and micro-habitats will be reconstructed using boulders, trunks and other elements.

- Low-grade ore will be processed at the end of the Project life. This means that the last tailings deposited in the tailings dam will have a lower ML/ARD potential, and therefore provide a better surface for rehabilitation and revegetation for the TSF and lower probability of generating ML/ARD
- Passive protection systems have been designed downstream of the TSF, including seepage ponds and a settling and polishing pond, which will mitigate potential residual impacts on surface water
- The sediment ponds and water treatment plants will be kept in operation until the last stages of definitive closure to ensure appropriate surface water management
- Opportunities will be identified to reduce environmental liabilities during the operating phase, in order to reduce the financial guarantee. These are progressive closure methods
- A period of between two and three years has been established in the design criteria for the active closure, this includes dismantling surface and underground facilities, covering the TSF with organic soil, chemical stabilization of contact water stored in the TSF, reshaping the waste rock storage area, and rehabilitation and revegetation of affected areas
- The monitoring plan which by law must last five years after the end of operations, will cover both the active and passive closure stages. Monitoring will begin once the primary rehabilitation activities have been completed. This monitoring will ensure that chemical, physical and hydrological stability have been achieved and that the natural forest growth processes in the closed, rehabilitated and revegetated areas can continue.

20.8.3 Closure Costs

Lundin Gold has allocated closure costs of $28.8 million in the conceptual Closure Plan. A number of areas that will impact the final closure estimate will be determined during the construction and operations phases, and prior to the final closure plan that is required to be lodged prior to cessation of operations. These areas include confirmation of the necessary quantity of NAG and/or neutralizing material that will available to effectively manage the PAG waste rock in the long term, confirmation of the flows and quality of water requiring treatment post-operations, predictions of the number of years water treatment would likely be required once rehabilitation activities have been completed, evaluation of the water capture and treatment that may be required for the Hollín Borrow Pit and associated WRSF, confirmation of the cyanide destruction process, and consideration of the most applicable cover and rehabilitation materials for the various areas. Lundin Gold will also review opportunities to conduct progressive reclamation. There is an expectation that the final closure costs may be higher than the allocation in the conceptual closure plan when the applicable data have been reviewed and incorporated.

20.8.4 Risk Assessment

A conceptual analysis of the risks associated with closure of the FDN Project was undertaken. Risks were classified as being either high, moderate or low. Table 20-8 summarizes the risks that were ranked as "high", and the proposed management measures to mitigate those risks.

20.9 Permitting

20.9.1 EIS History

In June 2003 Aurelian presented an EIS to the Undersecretary of Environmental Protection (part of the Ministry of Energy and Mines) for the Advanced Exploration Phase for the La Zarza mining concession. This was approved in March 2004. In October 2011 Aurelian presented an EIS to the authorities for the beneficiation, smelting, and refining stages of the FDN Project, and obtained the Environmental Licence in 2013. Following Lundin Gold's acquisition of the Project, the FDN Project was further optimized, resulting in changes to the beneficiation phase and underground mine access points. Consequently the EIS required updating.

Lundin Gold presented a letter to the MOE with the technical update to the EIS, which was approved in April 2015.

During 2015, a number of changes to the mining concessions were applied for and approved through the Ministry of Mines. This resulted in an additional Update to the EIS being required.

The Terms of Reference for the Update to the FDN Mining Project EIS, with operative areas in the La Zarza, Colibri 2 and Colibri 4 concessions were approved in April 2016, which allowed for the immediate submission of the draft amended EIA to the MOE. The community and public participation process has commenced, and the current forecast is for approval of this EIS is in the fourth quarter, 2016.

Concurrently to the Update to the FDN Mining Project EIS being prepared, Lundin Gold has been preparing an associated EIA document for the Project, in accordance with the International Finance Corporation (IFC) Performance Standards. This EIA will be used for corporate purposes, and will provide a platform for environmental and social risks of the Project to be assessed and managed to leading practice standards. This EIA is still being developed.

Table 20-8: High Risks Identified in the Conceptual Analysis for Closure of FDN Project

Aspect	Consequence	Closure Techniques and Options
Community Health and Safety	Injuries or fatalities due to unauthorized entry into underground areas	Access to underground workings will be closed by filling the ramps and installing concrete plugs at the portals, as well as at all access points (escape routes, ventilation shafts) Warnings and prohibited access signs will be installed. A perimeter fence will be installed, especially around the plugged accesses.
Fauna	Deaths of species cataloged as EN or VU by the IUCN by getting into underground areas and not being able to escape.	Close the access points (as described above).
Community Health and Safety	Access to disturbed areas by residents can increase the risk of accidents involving locals and informal miners.	Install fences and signs to discourage residents from going into the borrow pit area. Especially those areas where there is a greater risk of major landslides. The quarry access roads will be closed.
ML/ARD (Geochemical Stability)	Impacts on the surface and underground water. Underground water does not have an identified use, surface water is mainly used for livestock in the El Remolino and Machinaza Alto areas, which are located more than 20 km downstream of the Project.	Carry out geochemical characterization of the waste rock stockpiles during the project operation and separate PAG and non-PAG materials. The waste rock stockpile areas will be installed on a previously compacted layer of soil with a low permeability coefficient (saprolite). The most appropriate coverage option for waste rock stockpile areas will be investigated. The cover type will be determined based on pilot tests, and will depend on the final geometry of the stockpiles, availability of materials, and the ML/ARD potential. The acid water generated by waste dumps will be collected and sent to the treatment plant until values permitted under Ecuadorian legislation are reached. Sources of sealant materials will be investigated. Water from PAG material will be collected and held in ponds at the waste rock stockpile area until the chemical make-up is determined. Underground water monitoring wells will be installed to identify any impacts on underground water and identify potential leachate plumes. Monitor surface water.
Physical Slope Stability/Erosion	If there is instability, impacts may be generated on surface drainage, such as plugging, which could increase risks downstream.	Profile the berms in the waste rock stockpiles and increase the slope safety coefficient for closure. Create cover using suitable material, which will be determined in future project stages. Build internal crown ditches on each berm of the waste rock stockpile and external channels. If the waste rock stockpiles can be eroded by nearby water courses, controls will be applied to prevent subsidence of the stockpile foundations.
Surface Water Drainage	Surface runoff and storm water drainage may increase both the chemical and physical instability; the latter due to an increase in erosion.	Build and maintain erosion control and surface water management works, including crown ditches on each berm and reduction of slopes. Build and maintain perimeter diversion channels around the waste rock stockpile areas.
Landscape	The visual impact caused by the WRF	Profile the waste rock stockpile berms to achieve a natural shape,

Aspect	Consequence	Closure Techniques and Options
	will be both obvious and high impact. Rapid information dissemination through images uploaded to social networks and other internet sources. The stockpile areas are not visible from tourist areas or villages, and therefore significant impact is not expected. Negative reaction from residents in the AII is not expected.	cover them with impermeable soil from construction and then with previously stored topsoil. Exposed tailings surfaces will be revegetated with native species that grow quickly and allow natural forest growth. Begin the revegetation process using native pioneer species from the plant nursery. Install erosion controls to prevent the topsoil from being washed out; specifically designed for the slope established in the final Closure Plan. Monitor the revegetation until conditions for natural forest succession are achieved.
Compacting	Soil will have been compacted in areas where heavy machinery operates, and due to the weight of structures creating unfavourable conditions for proliferation of seedlings.	Deep ripping of soils in construction areas to re-establish low compaction conditions that are suitable for the growth of native species.
Erosion	Uncontrolled erosion of the cover of the TSF would allow infiltrations and water will become contaminated.	Install erosion control on the surface of the TSF cover and do revegetation.
Structural Stability	In the event of a structural collapse, a movement could be created in the tailings mass, which could contain low levels of cyanide and heavy metals, directly impacting the Machinaza River.	The tailings dam design considers safety coefficients that are appropriate for the maximum probable earthquake event, and it is built with rock (not soil or cycloned tailings) which provides higher resistance and eliminates the possibility of instability due to liquefaction. Install geotechnical monitoring devices to identify dam behavioural patterns.
Acid Rock Drainage	Impacts on surface and underground water, generation of acid soils, generation of metal leaching that can affect water fauna, especially, and fauna that use the TSF pond as drinking water, also has a negative effect on the revegetation process of the coverage.	The tailings areas will be covered. The cover will be determined based on the geochemical characteristics of the tailings and pilot cover tests to establish the most appropriate cover to minimize reactions between the tailings, oxygen and water. This cover will be designed to minimize the possibility of generating ML/ARD. Once the TSF is closed, there will be two zones, a dry zone, and a wet zone made up of the decant pond. It is probable that the water that filters through the tailings dam will comply with the discharge parameters. To check this, chemical analyses will be done during operations to determine the absorption capacity of the saprolite under the dam wall.

20.9.2 Required Permits

The key permits that will be required to support construction and operations are summarized in Table 20-9.

Table 20-9: Key Permits

Permit/Requirement	Entity/Organization/Authority	Estimated Time to Obtain	Prerequisite
Permits to be Obtained Before Construction			
Certificate of Validity of Mining Rights	Arcom	15 days	
Environmental Licence for the macro project (mining and ore treatment)	MOE	10 months	Certificate of Validity of Mining Rights
Permit for water use for human consumption	Senagua with opinion by ARCA	10 months	
Water deviation permit for TSF	Senagua with opinion by ARCA	10 months	
Authorization to install and operate a process plant	Ministry of Mining	45 days	
Authorization to install and operate a TSF	Ministry of Mining	45 days	
Permit or authorization to extract aggregate and stone material	GAD Yantzaza	1 month	Environmental Licence
Authorization from the municipality of the district in the area of influence to deposit non-hazardous solid waste in the landfill	GAD Yantzaza	NA	NA
Fuel purchase permit	ARCH	5 days	
Permit of Approval of Drawings and Specifications for Camps	Arcom / Ministry of Mining; Ministry of Labour	ND	
Permit from the Fire Department for project facilities	Yantzaza Fire Department	1 month	
Project Construction Permit	GAD Yantzaza	45 days	Certificate of No Debts with the Municipality Municipal Business Licence (Patente) Project Property Base Line Form Permit from the Fire Department for Project facilities
Qualification for the initial request for heliport installation	DGAC	15 days	
Mining Easement	Arcom	4 months	
Registration as a Hazardous Waste Generator	MOE	8 months	Registration as a Hazardous Waste Generator (current)

Permit/Requirement	Entity/Organization/Authority	Estimated Time to Obtain	Prerequisite
Registration as an Explosives Consumer	Joint Chiefs of Staff of the Armed Forces	48 days	Certificate of Feasibility (for using explosives as part of the Project) Explosive Magazine Construction Permit
Transportation Permit for Explosives (Waybills)	Joint Chiefs of Staff of the Armed Forces	1 day	Registration as an “Explosives Consumer”
Permits to be Obtained During Construction			
Phytosanitary Certificate of Operation for Centres Producing Plant Material for Propagation (Plant Nursery Operation Permit)	Magap	30 days	
Operation Permit from the Fire Department (verification)	Yantzaza Fire Department	15 days	Permit from the Fire Department for project civil works
Dining Hall Operation Permit	Arcsa	5 days	Operation Permit from the Fire Department (verification)
Medical Clinic Operation Permit	Arcsa	1 month	Operation Permit from the Fire Department (verification)
Wood transport waybills	MOE	1 day	NA
Permits to be Obtained Before Operations			
Qualification for Importation, Possession and Use of Substances Controlled by Seted	Seted	15 days	
Transportation permit for controlled substances (waybills)	Seted	1 day	Qualification for importation, possession and use of controlled substances
Certificate of Registration of Hazardous Chemical Substances	MOE	2 months	Environmental Licence for the macro-project (mining and ore treatment)

Notes: ARCA: Administración de RegulaciónRegional para y Control de Agua or Regional Water Regulation and Control Administration; ARCH: Administración Regional de Regulación ypara Control de Hidrocarbonos or Regional Hydrocarbon Regulation and Control Administration; ARCOM: Agencia de Regulación y Control de la Mineria or Mining Regulation and Control Agency; DGAC: Dirección General de Aviación Civil or General Directorate of Civil Aviation; ARCSA: La Agencia Nacional de Regulación, Control y Vigilancia Sanitaria or National Agency for Sanitary Regulation and Control; GAD Yantzaza: Gobierno Autónomo Descentralizado Yantzaza, or decentralized autonomous government of Yantzaza; Magap: Ministerio de Agricultura, Ganadería, Acuacultura y Pesca or Ministry of Agriculture, Livestock and Fisheries; MOE: Ministerio del Medio Ambiente or Ministry of Environment; Senagua: Secretaría Nacional del Agua or Water Commission; Seted: Secretariat Técnico de Drogas or Drugs Technical Secretariat. NA = not applicable; ND = not determined.

Permits Required in Support of Construction

Before construction of permanent facilities starts, approximately 16 permits are required (not counting prerequisites); the most important is the updated Environmental Licence. This licence includes mining, beneficiation, smelting and refining and is issued by the MOE. This complete process can take up to 10 months. Lundin Gold advised that this process is well advanced, with the approval of the EIA expected in early fourth-quarter 2016. However, even though the Project has an Environmental Licence, it cannot use water from natural sources without a permit for human consumption and/or industrial use issued by Secretaría Nacional del Agua (Senagua). This permit process can also take up to 10 months. Lundin Gold advised that this process is also well advanced, and the licences are projected to be issued in February 2017, well ahead of requirements.

When these two processes are complete, Lundin Gold can request an “Authorization to Install a Process Plant” and an “Authorization to Build a Tailings Storage Facility”, which are issued by the Ministry of Mines.

The permits for explosives are included in the 16 permits required before construction. Registration as an explosives consumer takes 15 days; however, before this registration is approved, there are six prerequisite permits. These prerequisite permits take at least 25 days to obtain, and are related to the explosives magazine. When Lundin Gold is registered as an explosives consumer, the company can then obtain a permit for explosives transportation.

Six permits are required during construction which are related to secondary and/or complementary project services, such as wood transport waybills, dining hall operation, medical clinic operation and authorization for use of the heliport. These permits are issued by sectorial authorities, and the process can take up to one month. The local fire department must also inspect the Project facilities during construction.

Lundin Gold already has permits for general operations for the plant nursery (“Phytosanitary Certificate”), and generation and management of solid waste (“Registration as a Hazardous Waste Generator”). Wood transport waybills must be obtained each time wood is transported.

Permits Required Prior to Operations

Before operations start, three permits are required, and relate to import, transport and use of various chemicals. For general operations, Lundin Gold must obtain a “Qualification for Importation, Possession and Use of Substances Catalogued and Subject to Audit” from the Secretaría Técnica de Drogas (SETED), and a “Certificate of Registration of Hazardous Chemical Substances” from the MOE.

Transportation permit for controlled substances (waybills) must be obtained each time controlled substances are transported.

Permits Required during Operations

During operations, Lundin Gold must validly maintain Environmental Management Plan (EMP) performance bonds, permits for water use (human consumption), the fuel purchase permit, the fire department permit, the municipal business licence (patente), driving licences for each driver, vehicle registrations, the plant nursery operations permit, the dining hall and medical clinic operations permits, registration as an explosives consumer and the registrations for use of hazardous and controlled substances. Lundin Gold must also obtain waybills for transportation of explosives, wood and controlled substances when required.

A plan has been prepared to obtain the permits and registrations required for each Project phase, based on the order of application for each permit and the processes to be followed for each permit to meet the requirements of the issuing authority. The requirements to fulfill legal and regulatory issues after the permits have been issued have also been identified.

20.9.3 Environmental Design Criteria

The environmental design criteria (EDC), updated through October, 2015, are based on Ecuadorian law, quality criteria and regulations, as well as international standards such as those issued by the IFC, the World Bank, the World Health Organization (WHO), the International Cyanide Management Code (ICMC), the International Network on Acid Prevention (INAP), and the International Council of Mining and Metals (ICMM).

The EDC will be used throughout the various Project design phases to ensure the FDN Project meets the required and recommended regulations, guidelines and practices. Key areas that must incorporate the EDC that have been identified include:

- Environmental air quality: underground mine, waste rock facilities, TSF, process plant, infrastructure and services, storm water and sedimentation ponds
- Gas emissions: underground mine, waste rock facilities, TSF, process plant, infrastructure and services
- Water quality: underground mine, waste rock facilities, TSF, process plant, infrastructure and services, storm water and sedimentation ponds
- Noise: underground mine, TSF, process plant, infrastructure and services
- Vibration: underground mine, TSF, process plant, infrastructure and services

- Light and lighting: underground mine, TSF, process plant, infrastructure and services
- Non-ionizing radiation: underground mine, process plant, and infrastructure and services.

The EDC includes notations as to the key regulations and guidelines that must be incorporated, such as:

- Strategies for managing hazardous materials
- Strategies for managing hazardous industrial waste
- Strategies for managing non-hazardous domestic waste
- Strategies for handling acid rock drainage
- General recommendations for managing highways and roads
- Strategies for handling biodiversity
- Strategies for handling archaeological and cultural heritage aspects.

20.10 Considerations of Social and Community Impacts

The Project's indirect influence is expected to extend to some neighbouring communities, including the parish of Los Encuentros and two communities from neighbouring parishes. Los Encuentros is a rural parish located in Yantzaza county, characterized by the existence of one main population centre (the parish seat and home of the parish government) where the population has consolidated. There are also several scattered population centres, known as communities, neighbourhoods and sectors.

Although there have been some cultural sites recorded in the study area, the Project is not expected to impact any cultural heritage, and strict archaeological protocols are in place in consultation with the INPC.

Lundin Gold, and Kinross before them, have undertaken perception surveys on an annual basis to measure the attitudes and perceptions of the local communities to:

- Existing environmental baseline
- The mining industry (including artisanal and large scale)
- Lundin Gold as a company.

The results of these surveys have shown that although perceptions of artisanal mining are low, the community is very supportive of the Project, and the primary concern is access to employment. There is currently no large-scale mining in Ecuador.

A community relations program has been defined based on the Community Development Support Program (PADC, Plan de Apoyo a Desarrollo de la Communidad) which seeks to implement corporate responsibility strategies, to maintain a social licence with the communities, and to comply with socio-environmental legislation applicable to Aurelian's operations. The PADC is based on the principles of community participation, sustainable development and human development.

20.11 Comments on Section 20

20.11.1 Baseline Studies

Baseline studies have been undertaken to support the FDN Environmental Impact Study (EIS), and subsequent updates to the FDN EIS. Abiota, biota, social and archeological baseline studies were performed.

20.11.2 Waste Rock Facilities and Stockpiles

A total of 0.74 Mt of waste rock material will need to be permanently stored on surface. The remaining waste generated from mining activities will be used for construction purposes or returned underground as backfill.

Three ore stockpiles, segregated by gold grade, will be developed, and all ore on the stockpiles will be fed to the mill by the time of mine closure.

20.11.3 Tailings Storage Facility

The TSF has been sited so as to minimize the catchment area and to maximize the separation distance from the Zarza River downstream. The tailings dam will be an earth and rock fill structure. A total of 12.15 Mt of GFL tailings will be pumped to the TSF at 55% solids over the mine life. Water will be reclaimed to the process plant as required; the excess will be delivered to WTPs for release to the environment.

20.11.4 Hollín Borrow Pit

A borrow pit is required to provide granular materials for construction and mine backfill, from construction through to mine closure. The Hollín Borrow Pit will be exploited as an open pit mine, and will primarily exploit Hollín Formation sandstones, but due to the quantity of material required, will extend through the Hollín Formation sandstones into the intrusive rocks of the Zamora Batholith.

20.11.5 Water Management

Groundwater in the mine area will be actively managed to address mining requirements. Dewatering wells are planned, and a groundwater monitoring program will be put in place.

Three types of water requiring management have been categorized, non-contact, contact and neutral waters. Six WTPs are planned.

A water balance model was constructed that tracks water from the sources, through collection and conveyance systems, usage, storage, treatment and discharge to the environment. The FDN water balance is a positive water balance with significantly more water coming into the system than required for operations. Excess water on the site will be managed and treated as required prior to discharge to the environment.

20.11.6 Closure Plan

Closure planning has been undertaken to a conceptual level, and will be updated throughout the Project life.

20.11.7 Environmental Considerations

Geochemical characterization of the waste rock has indicated that there is strong potential for this material to be PAG. Further work should be undertaken to confirm that the necessary quantity of NAG and/or neutralizing material is available to effectively manage the PAG waste rock in the long term.

Water quality modelling should be extended to cover the likely closure scenario to confirm the flows and quality of water requiring treatment post-operations, and to predict the number of years water treatment would likely be required once rehabilitation activities have been completed.

Further work should be undertaken to understand the quality of water from the Hollín Borrow Pit and associated waste rock facility, to ascertain the water capture and treatment required for these areas during operations and into closure.

The cyanide destruction process after treatment is reliant on a combination of natural degradation/dilution. One of the major activities of active closure will be the ongoing operation of the water treatment plant and detox system until the cyanide concentration in the tailings decant pond reach trace levels.

20.11.8 Permitting

Aurelian currently holds some of the permits that will be required for the mining exploitation phase (such as the Environmental Licence for the Underground Exploitation of Metallic Minerals, which was granted by the Ministry of the Environment on 3 January 2013). This licence will allow certain Early Works, such as the portal and waste rock facility construction, to start prior to the approval of the updated FDN Project EIS, which is currently in the approval process.

Before construction of permanent facilities starts approximately 16 permits are required; the most important is the updated Environmental Licence. This licence includes mining, beneficiation, smelting and refining and is issued by MOE. This process can take up to 10 months. Although Lundin Gold has a valid Environmental Licence, the company cannot use water from natural sources without a permit for human consumption and/or industrial use. This permit can also take up to 10 months to be issued by Senagua. When these two processes are complete Lundin Gold can request an “Authorization to Install a Process Plant” and an “Authorization to Build a Tailings Storage Facility”, both of which are issued by the Ministry of Mines.

The other permits required before construction starts are for Project facilities; these permits are issued by local and sectorial authorities. Lundin Gold has acquired some of the necessary permits.

A number of permits must remain current in support of operations.

20.11.9 Social Considerations

The Project’s indirect influence is expected to extend to some neighbouring communities, including the parish of Los Encuentros and two communities from neighbouring parishes.

Although there have been some cultural sites recorded in the study area, the Project is not expected to impact any cultural heritage.

The PADC is based on the principles of community participation, sustainable development and human development.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

21.1.1 Basis of Estimate

The methodology used in the development of this estimate and the level of engineering definition result in the estimate having an accuracy of ±10% to ±15% including the contingency based on the 80% confidence level. Costs were built in Q1 2016 US dollars.

The capital cost estimate is the product of the engineering developed during the 2016 FS, with inputs as summarized in Table 21-1:

- KCB designed and quantified the TSF for the Project, and the unit prices and labour costs were provided by Amec Foster Wheeler
- Lundin Gold provided the Owner's costs and the costs for the electrical transmission line, which were integrated into the overall estimate prepared by Amec Foster Wheeler
- NCL provided a mine schedule and plan, a backfill schedule and plan, a summary level capital cost estimate. Specifically NCL issued a "ready to work" equipment list with costs and cost details for mining development and complementary facilities. NCL also provided the man hours required for mining activities for the period covered by the capital costs (i.e. prior to start-up)
- P&C provided a basis of estimate and capital cost estimate for the paste plant, initial underground paste distribution installations, and CRF.

The Project schedule from detail engineering to start-up was also used in the estimate preparation.

The cost estimate was divided into the following elements:

- Capital costs:
  - Direct costs: costs for productive works and permanent infrastructure. Includes productive infrastructure, services and equipment required for the extractive process
  - Indirect costs: costs needed to support the construction of the facilities included in the direct costs. Includes engineering, procurement and construction management (EPCM) services, EPCM temporary facilities (infrastructure) and construction management, construction camp and associated services, capital spare parts, freight and logistics

Table 21-1: Capital Cost Estimate Responsibilities by Firm

Cost Area	Entity Responsible
Surface mine infrastructure; coarse ore crushing; ore conveying; coarse ore storage; ore reclaim; grinding/milling/classification; flotation, regrinding and filtration; cyanide leaching; refining; lime and reagents; process utilities and services; tailings disposal; reclaim water; on-site mobile equipment; on-site bulk storage; on-site services/utilities; on-site control system, communications and security; on-site power supply and transmission; water management; off-site roads/water diversions; off-site facilities; off-site services/utilities; off-site power supply and transmission; airstrip; aggregate borrow pit; aggregate plant; EPCM services; temporary facilities; construction supplies; spare parts/first fills/commissioning; health/safety/security/environment; freight/logistics/taxes/utilities; other services	Amec Foster Wheeler
TSF	KCB
Owner's costs; studies (Owner's costs), electrical transmission line	Lundin Gold
Underground mine development; underground mining equipment; dewatering; ventilation and services; underground mine infrastructure; underground material handling	NCL
Backfill plant, CRF.	P&C

  - Owner's costs: costs associated with Lundin Gold's Project management, geological studies, support infrastructure, safety and environmental, community relations, administration and finance, human resources and others
  - Contingency: includes variations in quantities, differences between estimated and actual equipment and material prices, labour costs and site-specific conditions. Also accounts for variation resulting from uncertainties that are clarified during detail engineering, when basic engineering designs and specifications are finalized

- Sustaining and closure costs:

  - Capital expenditures after the start of operations include costs for the tailings dam wall growth, mine and other equipment replacement, the paste fill plant, and closure costs. These costs are included in the financial analysis in Section 22 in the year in which they are incurred. The capital cost estimate includes construction activity costs to Q1 2020. Costs after this are classified as sustaining capital.

The implementation phase is planned to start on 1 July 2017. Any construction before this date is considered Early Works (work plan capital) and is not included in the capital cost estimate.

The following items were excluded from the capital cost estimate:

- Sunk costs incurred to date, including studies and Early Works
- Taxes (included in the financial model)
- Geotechnical anomalies (must be considered as risk)

- Pre-operations testing and start-up beyond C4 certificate
- Operating costs
- Changes to design criteria
- Work stoppages
- Scope changes or an accelerated schedule
- Changes in national law
- Changes in national duties
- Hydrological issues
- Environmental issues
- Hazardous waste issues
- Closure costs (included in sustaining capital estimate)
- The cost of the mobile fleet required to support plant operations (these are included in the operating costs).

21.1.2 Direct Costs

Within Amec Foster Wheeler's estimate scope, direct costs are supported by information provided by engineering disciplines, such as equipment lists and material take-offs (MTOs). Prices were provided by the Project procurement group. Minor items, not quoted, were estimated using current information from Amec Foster Wheeler's database.

The costs for the main mechanical equipment, platework items, piping unit rates, the main electrical equipment and materials, and instrumentation prices were obtained from commercial bid evaluations.

Process equipment was classified as critical, main and secondary equipment. Critical equipment costs were based on firm quotations; main electrical and mechanical equipment and some secondary equipment costs were based on budget quotations; costs for other equipment were based on the current Amec Foster Wheeler database.

The concrete price was calculated from quotations based on third party supply of an on-site batch plant.

The work and material quantities were developed by engineering based on the drawings and documents when the take-offs were done.

Mass earthworks and structural earthworks, concrete and structural steel, geomembrane, piping and cables were based on the PDMS model. For the infrastructure (buildings) the estimate was based on the building surface areas and quotations for buildings using the architectural design and associated engineering. Instrumentation was based on the design development at the time the take-offs were done.

Construction and assembly costs were prepared taking into account the following:

- Contracting structure
- Quotations from specialized companies
- Current Ecuadorian contracts (quotes)
- Site conditions.

Amec Foster Wheeler's procedure is to prepare a set of factors that reflect the country-specific standards plus the impact of local project conditions. Based on these elements a productivity factor table is prepared reflecting the expected project efficiency against the standards. Once the factor was obtained, the final productivities were checked using benchmarking based on information from local contractors.

The labour rate includes both the direct and indirect cost of labour. Rates were determined for a typical crew for this type of project. Man hours were estimated using an efficiency index for the crews. These were taken from Amec Foster Wheeler standard working hours, which are based on typical mining projects and vendor information.

The cost of the construction equipment, estimated as dollars per hour of direct labour include purchase costs, depreciation, insurance, fuel, oil, lubricants, maintenance, service and repair. Construction equipment for earthworks was estimated as dollars per machine hour. The equipment operator hourly costs are part of this rate; they are not included in the labour crews.

Direct costs total US$426.1 million.

21.1.3 Indirect Costs

Indirect costs include all costs needed to carry out the engineering procurement, construction/engineering procurement, and construction management (EPC/EPCM) services for the Project. These costs were calculated by Amec Foster Wheeler's estimating group, using a spreadsheet. The main costs in this category are EPCM services, construction camp, catering, temporary facilities, third-party services, spare parts, freight, and customs. This estimate includes Owner's indirect costs calculated by Lundin Gold (see Section 21.1.4). The construction contractors' indirect costs are included in the construction costs.

The indirect costs were calculated using various sources of information, including the Construction Execution Plan and information provided by Lundin Gold.

Indirect costs, excluding Owner's costs, total US$126.1 million. Including Owner's costs, the total indirect capital cost estimate is US$175.4 million.

21.1.4 Owner's Costs

The following items are part of the Owner's costs:

- Environmental management and mitigation
- Security
- Pre-investment costs
- Personnel training
- Administration, financial and human resources costs (for Lundin Gold scope)
- Community relations.

Owner's costs total US$49.3 million.

21.1.5 Contingency

A range analysis was developed, based on Amec Foster Wheeler's probabilistic model with participation from Lundin Gold. All the cost elements were included in this model. A percentile was used as the basis for calculating the contingency.

Costs were grouped according to the source and a minimum and maximum percentage of variation was assigned to each group:

- Quantity take-offs
- Design status
- Market conditions
- Unit installation man hours
- Labour productivity
- Labour cost per hour
- Sub-contract costs
- Equipment and material costs
- Construction equipment costs.

The model was based on the Monte Carlo method and simulated the probability distribution curve of the overall estimated cost. The input was a set of elements that grouped information by category, the variation range of the cost components, design level, pricing conditions, number of construction and assembly hours, productivity and labour cost, sub-contracts, materials and construction equipment costs. The model was operated using the @Risk simulation program.

The contingency was calculated for the Project as a whole as requested by Lundin Gold, i.e. the contingency was calculated on Amec Foster Wheeler scope costs and costs provided by Lundin Gold including Owner's costs, and costs provided by NCL (mine) and KCB (tailings).

The total amount calculated for the contingency is US$67.3 million.

21.1.6 Sustaining Capital

Sustaining capital costs were estimated using the same estimation basis as outlined for the direct costs.

The sustaining capital is for planned future capital works for the Project, mainly the tailings dam wall raises, paste backfill plant, and mine equipment replacement. Closure costs are also included in this category.

The estimated sustaining capital and closure costs total US$291.9 million. The portion of the sustaining capital cost estimate that relates to closure costs is estimated at US$28.8 million.

21.1.7 Capital Cost Summary

The initial capital cost for the Implementation Phase is estimated to be US$668.7 million. The sustaining capital is estimated at US$291.9 million. The total capital construction costs are therefore US$960.6 million.

Table 21-2 shows the initial capital cost summary by area, Table 21-3 shows the summary of the total cost of the Project, and Table 21-4 shows the sustaining costs by year.

Table 21-2: Initial Capital Cost Summary by Area

Description	Amount (US$ M)	% of Total
Underground mine	120.5	18.0
Ore handling	7.5	1.1
Process plant	74.3	11.1
Tailings/ reclaim water facilities	30.8	4.6
On-site infrastructure	121.4	18.2
Off-site infrastructure	71.2	10.6
Aggregate borrow pit	0.4	0.1
Indirect costs	126.1	18.9
Owners costs	49.3	7.4
Contingency	67.3	10.1
Total	***668.7***	***100.0***

Note: Totals may not sum due to rounding

Table 21-3: Summary of Project Total Cost

Description	Total (US$ M)
Total initial capital costs	668.7
Total sustaining capital	291.9
Project Total Cost	***960.6***

Note: Totals may not sum due to rounding

Table 21-4: Sustaining Capital Costs by Year Summary

Description	Total (US$ M)
2020	99.3
2021	16.1
2022	10.6
2023	12.8
2024	23.9
2025	21.5
2026	10.4
2027	13.1
2028	15.9
2029	17.6
2030	19.6
2031	2.4

Description	Total (US$ M)
2032–2035	0
2036	28.8
Total Sustaining Capital	***291.9***

Note: Totals may not sum due to rounding

21.2 Operating Cost Estimates

21.2.1 Basis of Estimate

The operating cost estimate was based on Q1 2016 assumptions. The estimate has an accuracy of ±10%. All operating cost estimates are in US$ and do not include any foreign exchange calculations.

Mining, process and tailings management are generally itemized in detail, however, G&A items such as training are calculated estimates, or have been included as an allowance. Many items of the operating cost estimate are based on budget quotations, allowances are based on in-house data.

The operating cost estimate is inclusive of site costs during the construction and operational period as described below:

- Upon the completion of C4 commissioning, this occurs when the C4 certificate is signed-off (see Section 24 for a discussion of the C1–C4 certificates)
- Operation of the plant to final Project close-out (LOM) including waste management facilities.

The following were excluded from operating cost estimate:

- Costs up to and including C4 commissioning are excluded from operating costs and are included in the capital cost estimate.

The overall estimate combined inputs from Amec Foster Wheeler, KCB, Lundin Gold, NCL, and P&C as summarized in Table 21-5. General rates used in the estimate are summarized in Table 21-6.

Costs were based on the mine schedule indicative tonnage per time period that was produced by NCL on 25 February, 2016.

Table 21-5: Operating Cost Estimate Responsibilities by Firm

Cost Area	Entity Responsible
Process plant; Hollín Borrow Pit and plant; on-site and off-site infrastructure	Amec Foster Wheeler
Tailings storage facility	KCB
Labour rates; Owners costs; general rates	Lundin Gold
Mining costs	NCL
Paste backfill	P&C

Table 21-6: General Rate Assumptions

Area	Unit	Value
Plant capacity	t/d	3,500
Days per year	days	365
Hours per day	hours	24
Years of operation	years	14
Power	US$/kWh	0.0883
LOM production	tonnes x 1,000	15,477
LOM gold grade	Au g/t	9.67
LOM silver grade	Ag g/t	12.74
Diesel premium (tax excluded)	US$/L	0.352
Diesel #2 (tax excluded)	US$/L	0.320
Gasoline (tax excluded)	US$/L	0.365
Fuel transport	US$/L	0.079
Value-added tax (IVA)	%	12

The following items were assumed:

- All equipment and materials will be new
- The labour rate build-up will be based on the statutory laws governing benefits to workers that were in effect at the time of the estimate.

The following items were specifically excluded from the operating cost estimate, unless identified by the Owner's team and included in the Owner's costs:

- Cost of financing and interest (e.g. spare parts)
- Costs due to extraordinary currency fluctuations (e.g. materials sourced from overseas)
- Changes in Ecuadorian law
- Other duties and taxes (except as identified)

- Any provision for force majeure events
- Pre start-up operations and maintenance training
- Transport and handling of concentrate and doré from the plant.

All regulatory requirements for operating issues are assumed consistent with those in place as of Q1 2016.

Additional assumptions included:

- No cost of commissioning assistance post C4 certificate issuance is included in the operating cost estimate
- Freight estimates are based on vendor supplied freight quotations or in-house data. Freight for reagents is included in the price of those commodities. Freight for steel consumables is included in the price of that material. Freight for spare parts is calculated as a percentage of equipment cost expected to be used annually
- No contingency is assumed
- No cost escalation (or de-escalation) is assumed.

21.2.2 Mine Operating Costs

Mining operating costs were developed from first principles, in a bottom-up unit cost approach:

- Modelling the mining cycle in detail, itemizing all materials and consumables used in each process. The tasks include drilling, blasting, mucking, bolting, utilities, backfill preparation and stope backfilling
- Locally-available, high-use commodities, including diesel fuel, cement, power and explosives, were priced from current Ecuadorian information. Other commodities in the cost models were estimated from NCL's current supply database
- Equipment operating costs were estimated by determining the lubricant consumption and hourly maintenance costs. Fuel and power expenses were estimated separately
- Tasks included in the development cost estimates included jumbo drilling, blasting, mucking, bolting and service installation. These costs were calculated per linear metre of development.

21.2.3 Process Operating Costs

The process plant operating costs are based on operations in the following areas:

- Coarse ore crushing
- Ore conveying
- Coarse ore storage
- Ore reclaim
- Grinding/milling/classification
- Flotation, regrinding and filtration
- Cyanide leaching and destruction
- Refining
- Lime and reagents
- Process utilities
- Tailings thickening.

The following were incorporated:

- Reagent quantities were based on the process design criteria and take into account testwork results, scale-up from laboratory to operations, and mass balance impacts. The cost per tonne was fixed, and the annual cost varied with the tonnage produced from the mine schedule
- Steel consumables for the process plant included grinding media for the SAG mill, ball mill and regrind mill and were calculated from first principles
- Process power loads, were based on estimated average demand. Estimated annual power usage was calculated from the production tonnage, and power costs were calculated using the rates provided for the Project
- Estimates of spare parts and maintenance supply costs area were provided by an external consultant with input from Amec Foster Wheeler. Costs included lubricants, parts, materials, steel liners, repair, external repairs, and inspections. Costs were also included for services provided to the plant by contractors and vendors.

The transport of the gold concentrate from the process plant to the port was excluded from the process area estimate but was accounted for in the financial model (refer to Section 22).

21.2.4 Infrastructure Operating Costs

On-site and Off-site Infrastructure

Power loads were based on estimated average demand. Estimated annual power usage was fixed and costs were calculated using the rates provided for the Project.

Annual maintenance costs for the on-site and off-site infrastructure were based on an estimate of 3% of the capital cost of mechanical equipment. In addition, estimates of spare parts and maintenance supply costs were provided by an external consultant with input from Amec Foster Wheeler. Costs include lubricants, parts, materials, repair, external repairs, and inspections.

The accommodation camp will require a sewage treatment plant and potable water treatment plant.

The camp size was based on the peak personnel head count.

Water Management

Water management costs include the operation and maintenance of the non-contact diversion systems, sedimentation ponds, WTPs, pipelines and pumping systems.

Power loads were based on estimated average demand from the water treatment equipment, and on the assumed power requirements.

Reagent quantities were based on the water treatment plant design including the estimated consumptions for the HDS, clarification and UF systems.

The Hollín Borrow Pit water treatment plant was scaled from the MWTP, with the scaling assumptions based on factoring the flowrates for each plant.

Tailings Storage Facility

The operating costs for the TSF included the distribution of tailings, in addition to TSF support facilities such as diversion channels, polishing ponds, seepage pumps, and geomembrane liners.

Dam raise construction activities will be completed by a contractor and are included in sustaining capital and thus excluded from the operating cost estimate.

Fuel consumption was estimated based on equipment usage for staff mobilization, dam raises, operation of the seepage reclaim pumping system, and the back-up diesel generator.

Power costs are included for the distribution of tailings and are estimated from the LOM dam raise construction curve.

Maintenance of the TSF area was estimated by sub-area and included regular tasks performed by the maintenance crew in addition to contractor services required to support larger activities. Tailings distribution, lighting systems, tailings pipeline relocation, and seepage reclaim system maintenance were assumed to be performed by the maintenance crew. Contractor service maintenance will be required for the tailings management area, diversion channels and polishing pond.

Shared Services

Areas that will be shared by various groups include the assay laboratory, maintenance team, surface mobile equipment, and the Hollín Borrow Pit and associated plant. These costs were calculated by area and back-allocated.

21.2.5 General and Administrative and Owner Operating Costs

General and administrative (G&A) costs for the operations phase were established from current knowledge of the site costs and the proposed operational structure. Costs were estimated by area, and include provisions for business sustainability, finance, environment and permitting, human resources, training, health, safety, and security, technology, supply chain, site administration, and general management and cover the Quito, Los Encuentros and FDN site offices.

In general, staff located in Los Encuentros and Quito will work regular hours from Monday to Friday. Site staff are planned to work 14 x 14 day shifts of 12 hours. Site managers and administrative staff will typically have a 5 x 2 day roster, with eight hour workdays. Staff will travel from Quito and Los Encuentros to the FDN site as necessary to support operations.

The labour costs were developed based on a head count and the total personnel associated with the operation. Labour costs were based on rates provided by Aurelian Ecuador S.A. and an operational labour force estimate. Labour costs were allocated by area and by location. The basis for the calculation of labour costs included base salaries, social benefits and taxes. In addition, travel, camp, and catering costs were incorporated into the labour cost estimate.

21.2.6 Operating Cost Summary

The overall life of mine operating cost estimate is US$118/t in Q1 2016, and includes base costs, non-recoverable taxes and leasing. The LOM undiscounted total is estimated to be US$1,828 million. Operating costs are estimated at US$414/oz Au including all site costs and excluding transportation of concentrate and doré, based on the production of 4.42 Moz of gold over the LOM.

A summary of the costs by area is provided in Table 21-8.

Table 21-7: Operating Cost Summary

Area	LOM Total US$ (million)	US$/t	US$/oz Au
Mining	934.4	60.30	211.5
Process	516.9	33.40	117.0
Surface infrastructure	142.8	9.20	32.3
G&A	234.2	15.10	53.0
Total	***1,828.3***	***118.00***	***413.8***

Note: Totals may not sum due to rounding

The mining cost is the greatest contributor to the overall operating cost followed in order by process, G&A and surface infrastructure costs.

21.3 Comments on Section 21

The initial capital cost, excluding Early Works, is estimated to be US$668.7 million. The sustaining capital is estimated to be US$291.9 million.

The overall life of mine operating cost estimate is US$118/t, and includes base costs, non-recoverable taxes and leasing. Operating costs are estimated at US$414/oz gold, including all site costs.

22.0 ECONOMIC ANALYSIS

22.1 Forward-looking Information

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here. Forward-looking statements in this Report include, but are not limited to, statements with respect to future gold and silver prices, the estimation of Mineral Reserves and Mineral Resources, the realization of Mineral Reserve estimates, unexpected variations in quantity of mineralized material, grade or recovery rates, geotechnical and hydrogeological factors, unexpected variations in geotechnical and hydrogeological assumptions used in mine designs including seismic events and water management during the construction, operations, closure, and post-closure periods, the timing and amount of estimated future production, costs of future production, capital expenditures, future operating costs, costs and timing of the development of new ore zones, success of exploration activities, permitting time lines and potential delays in the issuance of permits, currency exchange rate fluctuations, requirements for additional capital, failure of plant, equipment or processes to operate as anticipated, government regulation of mining operations, environmental, permitting and social risks, unrecognized environmental, permitting and social risks, closure costs and closure requirements, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

22.2 Methodology Used

The Project has been evaluated using a discounted cash flow (DCF) analysis. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including the three years of pre-production costs; operating costs; taxes; and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the mid-point of each period.

To reflect the time value of money, annual net cash flow (NCF) projections are discounted back to the Project valuation date using several discount rates. The discount rate appropriate to a specific project depends on many factors, including the type of commodity; and the level of project risks (e.g. market risk, technical risk and political risk). The discounted, present values of the cash flows are summed to arrive at the Project's net present value (NPV).

In addition to the NPV, the internal rate of return (IRR) and the payback period were also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project.

22.3 Financial Model Parameters

22.4 Basis of Analysis

The financial analysis was based on the Mineral Reserves presented in Section 15, the mine and process plan and assumptions detailed in Sections 16 and 17, the projected infrastructure requirements outlined in Section 18, the doré and concentrate marketing assumptions in Section 19, the permitting, social and environmental regime discussions in Section 20, and the capital and operating cost estimates detailed in Section 21.

22.4.1 Metal Pricing

For the purposes of the financial analysis, the gold and silver prices projected are presented in Table 22-1.

22.4.2 Transport Costs

Doré

A helipad will be built near the process plant area, and helicopter transport will be used to fly doré from site to an international commercial airport in Ecuador. The doré will then be delivered by air to the nearest commercial airport to the refinery by the security contractor. Most refineries will handle the logistics chain from the point of receipt at the mine site through delivery to the refinery.

Doré transport and insurance costs are expected to average US$3.69/oz of gold produced.

Concentrate

MIQ Logistics evaluated the concentrate transport options. For the purposes of the 2016 FS, it was assumed that concentrate would be trucked in a 20 ft container, capable of holding up to 24 t of concentrate, to the selected port of export. A review of the port facilities indicated that either Guayaquil or Bolivar could be used.

The transport cost for concentrate including cargo costs, extra volume surcharge, container prices, temporary storage, external warehousing, and maritime transport is expected to average US$174/wmt.

Table 22-1: Gold and Silver Price Projections

Metal Price Assumptions	Units	LOM
Gold	US$/oz	1,250
Silver	US$/oz	20

22.4.3 Working Capital

Working capital cash outflow and inflows are included in the model. The calculations are based on the assumptions that accounts payable will be paid within 30 days and accounts receivable will be received within 60 days.

The impact of the working capital on NPV 5% is approximately US$23 million.

22.4.4 Royalties

The Ecuadorian Government, as the owner of non-renewable natural resources, is entitled to receive royalties from mining companies that have exploitation agreements. The rate is 5% of the net sales less treatment costs and refining costs (TC/RCs) and transportation costs. Royalties are paid semi-annually in March and September of each year.

Lundin Gold agreed to an advance royalty payment of US$65 million, with US$25 million due upon execution of the Definitive Exploitation Agreement. The balance of the payment will be due in two equal payments on the first and second anniversaries of the execution of the Definitive Exploitation Agreement. The amount is deductible against the royalty payments and will be the lesser of:

- 50% of the royalties in the semester
- 20% of the amount of the anticipated royalties.

The first royalty prepayment of US$25 million is not included in the model because it falls before the valuation start date.

The Hollin Borrow Pit is subject to a royalty at a rate of 3% of the production costs.

A 1% net revenue royalty is payable to a third party on production from Aurelian's mining concessions, including the La Zarza concession (see discussion in Section 4).

22.4.5 Taxes

Applicable Taxes

The Project was evaluated considering the following taxes:

- Income tax

- Profit sharing
- Government royalties applied to mining
- Value-added tax (IVA in the Spanish acronym)
- Currency export tax (ISD in the Spanish acronym, an outflow tax)
- Windfall taxes and sovereign adjustment payment (government share)
- Other taxes
  - Business permit
  - Gross assets tax
  - Environmental conservation tax
  - Contribution to the Superintendent of Companies
  - Fodinfa

Companies domiciled in Ecuador are subject to tax on their worldwide income, at a rate of 22%. The income tax is paid annually in April of the following calendar year. The income tax basis is determined by the total taxable income less allowable deductions according to the tax law. All deductions and rates are based on currently-enacted legislation, and are subject to change in the future.

Profit sharing rate is 15% calculated on the taxable income. Workers linked to the mining activity will receive 3% and the remaining 12% will be paid to the Federal Government and the local governments (GAD).

Indirect Taxes

IVA at the general rate of 12% applies to the acquisition of goods and rendering of services in Ecuador with several exceptions established in the tax law. Importation of goods and services are levied with 12% IVA; as are copyrights, industrial property and related rights (royalties).

In Ecuador, exportation of goods and services are levied with 0% IVA, as well as, other goods and services specifically included in the tax law. Mining concessionaires are entitled to the refund of IVA paid as of 1 January 2018 which, taking into consideration the current law applicable for exporters, will be a maximum of 12% of the value of exports in the period.

All transfers of cash overseas are subject to a 5% ISD tax; some exceptions apply such as dividends that are paid abroad and importation of goods under special regimes. ISD is generated not only when foreign currency is actually sent from Ecuador overseas, but also in certain cases in which the law presumes foreign currency remittances.

Payments for the import of raw materials, supplies and capital goods contained on a list issued by the Tax Policy Committee generate a tax credit under the capital outflow tax. The ISD may be used for the payment of income tax or income tax advance or:

- Deductible expense in the current fiscal year in which the ISD payment was made
- Tax credit on the payment of income tax for the fiscal year or in the subsequent four years
- Reimbursement of the ISD in the fiscal year in which the payment was made, or in the subsequent four years.

Based on the base case metal prices used in the financial model presented in this Report, the Project will not trigger payment of either windfall taxes or the sovereign adjustment payment.

Other Taxes

The following additional taxes apply:

- Business permit: based on the equity of the company, the minimum payment is US$10 and the maximum US$25,000
- Gross assets tax: calculated by multiplying gross assets less current and contingent liabilities by 0.0015
- Environmental conservation: the concessionaire has to pay an annual fee per mining hectare each March, as follows:
  - During the initial exploration phase, an amount equivalent to 2.5% of a unified basic remuneration (the UBR is US$366 for 2016)
  - During the period of advanced exploration and economic evaluation an amount equivalent to 5% of the UBR
  - During the operations phase, an amount equivalent to 10% of the UBR
- Contribution to the Superintendent of Companies: the rate is 0.0071 to 0.0093 on real assets; the amount to be paid is calculated and supplied to the taxpayer by the Superintendent
- Fodinfa: this is an additional contribution on the import of goods for the Development Fund for Children. The taxable basis is the cost, insurance and freight of the importation, and the rate is 0.5%.

22.4.6 Historical Costs

An amount of US$430 million of historical costs is considered in the financial model. These historical costs provide a shield against taxes and profit-sharing expenses.

22.4.7 Financing

The model does not include any costs associated with financing.

22.4.8 Inflation

There is no adjustment for inflation in the financial model; all cash flows are based on 2016 US dollars.

22.5 Financial Results

Table 22-2 summarizes the financial results, with the base case NPV highlighted. The after-tax NPV at a 5% discount rate over the estimated mine life is US$676 million. The after-tax IRR is 15.7%. The after-tax payback of the initial capital investment is estimated to occur 4.5 years after the start of production. Table 22-3 shows the cashflow broken out on an annualized basis.

The LOM all-in sustaining cost (AISC) per ounce is US$623. Operating statistics are included as Table 22-4.

22.6 Sensitivity Analysis

A sensitivity analysis was performed on the base case NPV after taxes to examine the sensitivity to gold price, operating costs, capital costs and labour costs. The results of the sensitivity analysis are shown in Figure 22-1 for the after-tax scenario.

In the pre-tax and after-tax evaluations, the Project is most sensitive to changes in gold prices, less sensitive to changes in operating costs, and least sensitive to capital cost and labour cost changes.

The gold grade is not presented in the sensitivity graph because the impact of changes in the gold grade mirror the impact of changes in the gold price.

Table 22-2: Financial Analysis Summary (base case is highlighted)

Indicator	Units	LOM Value
Pre-Tax		
NPV 4%	US$ million	1,452
NPV 5%	US$ million	1,283
NPV 8%	US$ million	879
NPV 10%	US$ million	675
Payback period from start of production	Years	3.7
IRR	%	23.8
After Tax		
NPV 4%	US$ million	791
NPV 5%	US$ million	676
NPV 8%	US$ million	402
NPV 10%	US$ million	264
Payback period from start of production	Years	4.5
IRR	%	15.7

Table 22-3: Cash Flow on an Annualized Basis

	Units	LOM Value	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Metal prices																				
Gold	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Silver	US$/oz	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00	20.00
Mill feed																				
Mill feed	kt	15,490	—	—	—	595	1,205	1,212	1,161	1,250	1,270	1,278	1,278	1,281	1,258	1,278	1,278	977	171	—
Gold grade	g/t	9.67	—	—	—	8.47	8.65	10.89	11.97	10.36	10.37	9.76	9.39	9.39	9.34	9.21	9.01	8.07	10.00	—
Silver grade	g/t	12.74	—	—	—	9.22	10.49	13.64	12.41	12.64	13.24	12.93	11.75	12.59	15.51	14.23	12.94	11.86	12.71	—
Au recovered																				
In gravity	koz	914	—	—	—	35	67	83	103	83	81	75	76	72	58	62	64	43	11	—
In flotation	koz	3,096	—	—	—	99	212	270	268	263	271	260	245	250	257	254	244	168	35	—
In leach	koz	408	—	—	—	14	29	36	40	36	36	34	33	33	30	31	30	21	5	—
Total	*koz*	*4,418*	—	—	—	*149*	*308*	*390*	*411*	*382*	*389*	*368*	*354*	*355*	*345*	*346*	*339*	*232*	*50*	—
Ag recovered																				
In Gravity	koz	761	—	—	—	21	49	64	56	61	65	64	58	62	75	70	64	45	8	—
In Flotation	koz	4,224	—	—	—	113	266	350	293	333	357	354	318	346	434	400	359	253	46	—
In Leach	koz	192	—	—	—	7	14	17	19	17	17	16	16	15	13	14	14	10	2	—
Total	*koz*	*5,177*	—	—	—	*141*	*329*	*431*	*367*	*411*	*440*	*434*	*392*	*424*	*523*	*484*	*438*	*307*	*57*	—
NSR																				
Doré	'000 US$	1,669,490	—	—	—	62,463	120,943	150,990	180,771	150,570	148,085	136,749	137,175	132,730	111,986	117,219	119,482	81,170	19,156	—
Concentrate	'000 US$	3,631,205	—	—	—	116,571	246,622	313,614	312,079	306,179	319,078	305,724	287,388	293,787	302,589	299,712	289,185	197,524	41,152	—
Total	*'000 US$*	*5,300,695*	—	—	—	*179,035*	*367,565*	*464,604*	*492,850*	*456,749*	*467,163*	*442,473*	*424,563*	*426,517*	*414,575*	*416,931*	*408,668*	*278,694*	*60,308*	—
Mining	'000 US$	932,185	—	—	—	50,066	74,008	67,317	64,359	73,985	70,349	74,161	71,643	69,562	76,286	84,325	80,060	56,970	19,094	—
Operating Costs																				
Process	'000 US$	516,077	—	—	—	27,013	39,278	39,519	38,478	40,313	42,424	40,862	40,862	40,940	40,455	40,862	40,862	34,730	9,478	—
G&A	'000 US$	233,377	—	—	—	15,923	17,373	17,361	17,346	17,348	17,294	17,256	17,252	17,251	17,253	17,256	17,249	17,232	9,983	—
Surface costs	'000 US$	141,823	—	—	—	9,707	10,269	10,653	10,324	10,195	11,496	10,196	10,337	10,675	10,262	10,198	10,803	10,201	6,506	—
Total on site operating cost	*'000 US$*	*1,823,461*	—	—	—	*102,709*	*140,928*	*134,849*	*130,507*	*141,841*	*141,562*	*142,476*	*140,095*	*138,429*	*144,256*	*152,642*	*148,974*	*119,134*	*45,061*	—
Total transport and insurance doré	*'000 US$*	*4,879*	—	—	—	*322*	*348*	*362*	*372*	*362*	*361*	*357*	*356*	*355*	*349*	*350*	*350*	*332*	*303*	—
Total administration and transport concentrate	*'000 US$*	*133,133*	—	—	—	*3,997*	*10,075*	*13,463*	*11,894*	*12,731*	*10,932*	*10,185*	*10,719*	*10,148*	*11,204*	*9,990*	*9,008*	*7,280*	*1,505*	—
Total taxes and royalties	*'000 US$*	*914,155*	*12,354*	*34,889*	*41,183*	*16,400*	*(6,075)*	*15,976*	*68,293*	*55,504*	*92,691*	*88,533*	*88,380*	*89,012*	*81,137*	*85,017*	*81,092*	*47,049*	*22,719*	—
Capital Costs																				
Initial capital	'000 US$	668,733	77,582	286,959	264,696	39,496	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Sustaining capital	'000 US$	263,055	—	—	—	99,255	16,108	10,572	12,750	23,902	21,482	10,424	13,147	15,919	17,571	19,557	2,367	—	—	—
Closure costs	'000 US$	43,401	—	—	—	—	—	—	—	—	—	—	—	—	—	—	4,380	4,818	34,203	—
Total capital costs	*'000 US$*	*975,188*	*77,582*	*286,959*	*264,696*	*138,751*	*16,108*	*10,572*	*12,750*	*23,902*	*21,482*	*10,424*	*13,147*	*15,919*	*17,571*	*19,557*	*6,747*	*4,818*	*34,203*	*0*
Royalty advance	'000 US$	65,000	25,000	20,000	20,000	—	—	—	—	—	—	—	—	—	—	—	—	—	—	—
Working capital	'000 US$	0	—	—	—	(45,518)	(8,056)	(10,580)	(5,128)	6,934	(1,883)	4,072	2,792	(505)	2,528	202	976	18,770	29,334	6,061
Royalty and tax adjustment (see Note 1)	'000 US$		25,784																	

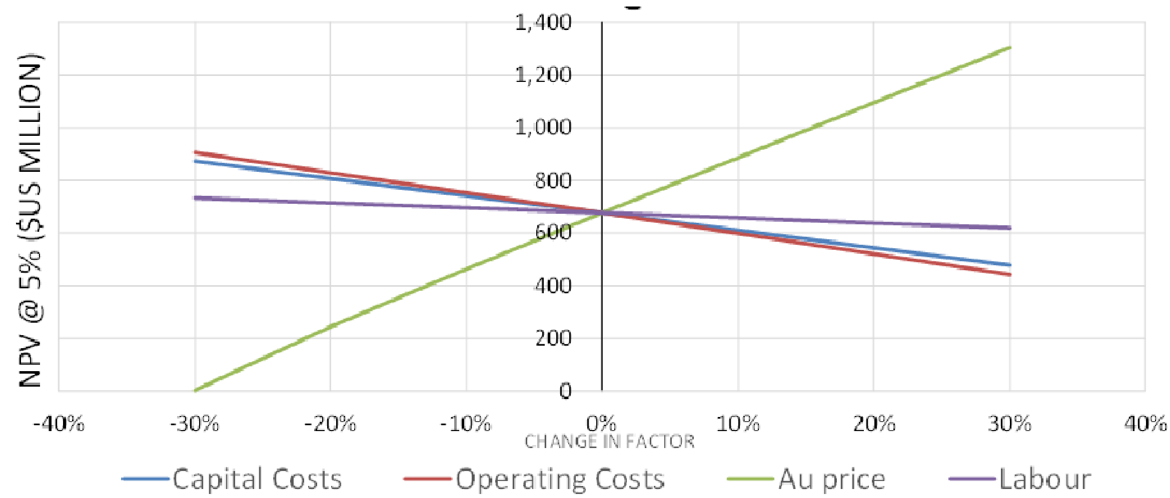
	Units	LOM Value	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034
Pre Tax																				
Cash flow	'000 US$	2,364,034	(77,582)	(286,959)	(264,696)	(112,261)	192,049	294,778	332,198	284,847	290,943	283,103	263,039	261,162	243,724	234,594	244,565	165,899	8,570	6,061
Cumulative cash flow	'000 US$		(77,582)	(364,541)	(629,237)	(741,498)	(549,448)	(254,670)	77,528	362,375	653,319	936,421	1,199,460	1,460,622	1,704,346	1,938,940	2,183,504	2,349,403	2,357,973	2,364,034
NPV 5%	'000 US$	1,283,273																		
Payback period from start of production	Years	3.7																		
IRR	%	23.8																		
After Tax																				
Cash flow	'000 US$	1,449,879	(89,152)	(341,848)	(325,878)	(128,661)	198,124	278,802	263,905	229,343	198,253	194,570	174,659	172,149	162,586	149,577	163,473	118,850	(14,149)	6,061
Cumulative cash flow	'000 US$		(89,152)	(431,000)	(756,878)	(885,539)	(687,415)	(408,613)	(144,708)	84,635	282,887	477,457	652,116	824,265	986,852	1,136,429	1,299,902	1,418,752	1,404,602	1,410,664
NPV 5%	'000 US$	676,454																		
Payback period from start of production	Years	4.5																		
IRR	%	15.7																		

Note 1: Represents taxes and royalties paid during Q1 and Q2 of 2017 prior to the start of valuation date

Table 22-4: Operating Statistics

	Units	Year 1	Year 2	Year 3	Avg. Y1–10	LOM
Metal Production						
Au recovered	koz	149	308	390	345	4,418
Ag recovered	koz	141	329	431	389	5,177
AISC Costs and Profit Margins per oz payable						
Au price	US$/oz	1,250	1,250	1,250	1,250	1,250
Cash cost sub-total (operating cost)	US$/oz	823.82	585.78	473.08	541.78	552.56
Sustaining and closure costs	US$/oz	701.12	63.77	35.86	102.92	70.87
AISC costs/oz Au payable	US$/oz	1,524.94	649.55	508.94	644.70	623.43
Operating Margin/oz Au payable	US$/oz	-274.94	600.45	741.06	585.52	626.57

Figure 22-1: After-Tax Sensitivity Analysis (NPV 5%)



Note: Figure prepared by Amec Foster Wheeler, 2016.

22.7 Comments on Section 22

Under the assumptions presented in this Report, the Project demonstrates positive economics. The after-tax NPV at a 5% discount rate over the estimated mine life is US$676 million. The after-tax IRR is 15.7%. The after-tax payback of the initial capital investment is estimated to occur 4.5 years after the start of production.

The LOM AISC per ounce is US$623

In the pre-tax and after-tax evaluations, the Project is most sensitive to changes in gold price, less sensitive to changes in operating costs, and least sensitive to capital cost and labour cost changes.

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Execution Plan

The proposed Project schedule is included as Figure 24-1.

24.1.1 Implementation Strategy

The implementation strategy for the Project is dictated by the duration of the construction of the twin declines, which will provide access to the deposit; the estimated duration for this construction is 34 months. It is possible to build all the surface facilities including the process plant and associated infrastructure during this period. Therefore, the construction of the mine access is the critical path and the Early Works (see recommendations in Section 26 for the proposed work program) to expedite the construction of the access are also critical. The objective of the Early Works is to build access and platforms for the start of construction of the portals and declines, and to provide support facilities. The Early Works have been given special attention in the execution plan because they will need to start very soon after approval of the 2016 FS, if the proposed Project schedule is to be met.

Planned Schedule

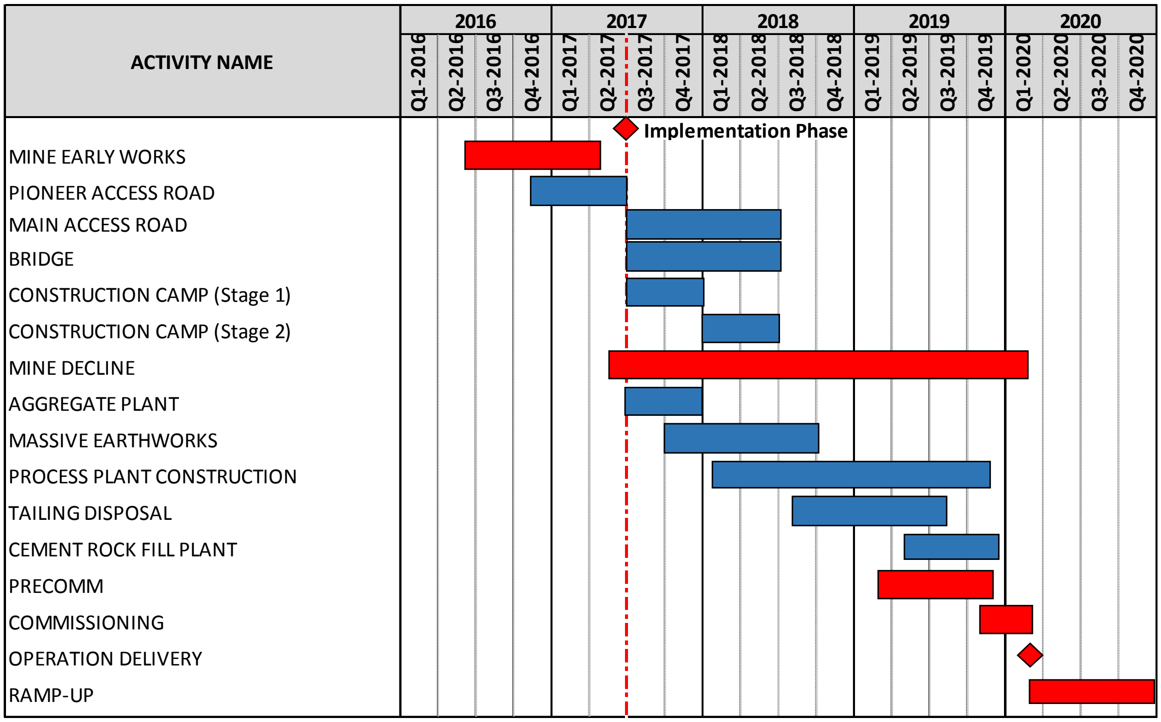
A Level 3 schedule was developed for the project considering the implementation strategy, key milestones mainly for the pre-operation of the mine, and external constraints such as environmental approvals and government regulations.

The Project schedule entails significant project activity durations, some of which may run concurrently, and includes a duration of 11 months for the engineering, procurement, contracting and preliminary construction of Early Works, 12 months for the construction of the access road and bridge over the Zamora River, 34 months for construction of the twin declines, six months to develop the aggregate borrow pit and plant, nine months for the mass earthworks and 20 months for the construction of the process plant and facilities.

Backfill Plant

The mining operation requires a CRF plant which will provide material for filling early stopes in the mine, and a paste backfill plant for filling later stopes when tailings are available. It is estimated that the paste backfill plant will only be required to be in operation after 2020.

Figure 24-1: Project Schedule



Note: Figure courtesy Lundin Gold, 2016.

Process Plant

The Project execution plan considers the parallel development of the mine and the construction of the process plant. The process plant will be pre-commissioned by section based on the construction completion. Six months has been allowed for pre-commissioning and includes the start to finish of all sections in the process plant. Sufficient time is allocated for corrections and for ensuring that the electro-mechanical operability of the plant is achieved prior to commissioning.

Commissioning will be staggered with pre-commissioning. It is planned to use low-grade ore from the low-grade stockpile for commissioning, which is expected to be completed in January 2020. Hand-over to the operations team is expected in early February 2020. The ramp-up to full production will start in February 2020 and continue to January 2021 when a throughput of 3,320 t/d will be achieved. The ramp-up period and monthly throughput is dictated by the mine plan. Throughput is expected to be consistent at 3,320 t/d for the next 37 months when the production will increase to 3,500 t/d as the mine develops.

Camps

A new camp will be built before the start of major works to house the construction staff. During the Early Works it is planned to expand the current Las Peñas camp and also to install a temporary camp at the portals. The camps will provide all the services necessary for accommodation, catering and recreation for the construction staff, including domestic and industrial water, power from diesel generators and sewage treatment.

Water Supply

Drinking water will be bottled water.

Construction water will be obtained from the accumulation ponds designed for the water management system. Each contractor will be responsible for extraction, transport, storage and distribution of the water from these ponds to the work site.

Power Supply

During the construction period the permanent power supply will not be available (the permitting and construction of the main supply line will be done in parallel with the construction of the mine and plant facilities). Therefore, it is planned to use diesel generators during the construction period. The permanent power supply is assumed to be available for the start of pre-commissioning of the plant in April 2019.

Implementation Team

Project implementation will be led by a team from Lundin Gold and a Prime Contractor who will be responsible for detailed engineering, procurement, contract administration, and construction management. The Project implementation strategy will include several types of contracting, which will be administered by the Prime Contractor:

- Engineering, procurement, and construction (EPC)
- Build, operate, and transfer (BoT)
- Construction contracts (CCs)
- Service contracts (SCs).

The Prime Contractor will be responsible for the integration of engineering by third parties into one single 3D model and for balance-of-plant-type engineering for a complete design. Design contracts will be awarded to specialist companies for areas such as the TSF and CRF plant.

Lundin Gold will institute an Operational Readiness program early in Project development to prepare and install the management team and systems, and to hire and train operating staff (see also Section 24.2).

The Lundin Gold team will supervise the overall site management during construction; however, it will be the Prime Contractor's responsibility to provide site functions and manage site services. The Operations team will be built up during construction and will work with the Prime Contractor's team during pre-commissioning. FDN Operations will take full control of the operation after commissioning, when load and performance testing is complete. FDN Operations will carry out warranty testing.

24.1.2 C1 to C5 Certification

The start-up of the facility will be a structured and planned process. The commissioning procedure will have four stages differentiated by the change of responsibility for the care, custody and control of the systems. This will be formalized by the use of the following certificates:

- C1: Construction completion: achieved when the C1 certificate is issued to indicate completion of the construction of discrete pieces of equipment within buildings, structures or systems. C1 equipment certification means that the particular piece of equipment has been installed following the design and is ready to be activated and tested as a component of the system
- C2: Pre-commissioning: achieved when the C2 certificate is issued to certify that testing has been completed on the equipment after being energized for operations. The C2 certificate applies only to equipment with a drive system and control and instrumentation devices. For C2 certification the equipment must be operative from the control system (using the operating software and program) at the operator's work station. The C2 certification is issued to confirm the communications and operability between the software and the equipment
- C3: Commissioning and water run testing: testing of the system operation is carried out as part of C3 certification. After C2 and C3 certification, the responsibilities for activities are transferred from the construction manager to the commissioning manager
- C4: Commissioning with load/performance testing: starts with the introduction of feed to the process plant. The systems requiring reagents are inspected and loaded during the final stage of the C3 commissioning. Commissioning will be

done at a multi-system level. The systems start-up sequence will follow the order of the plant flowsheets for normal operation. The C4 certification will occur when the performance milestones for production for all systems within the process plant are reached. At some stage during the C4 certification, the responsibility for activities will be transferred from the FDN commissioning manager to the FDN operations manager. The timing for this transfer will be determined by mutual agreement

- C5: Completion certificate/substantial completion: The signing of the C5 certificate confirms that the Project is operating to full production capacity and that the final delivery certificate is held by operations personnel. It indicates that the Project has been tested and Project specifications have been satisfied.

24.2 Workforce Considerations

24.2.1 Personnel Requirements

The direct and indirect personnel required to build the Project will consist mainly of local staff with the support of expatriates in key positions that require expertise in the execution of mining projects.

The peak of construction staff on site at any time is estimated to be about 1,370, with a total of approximately 2,040 employed.

24.2.2 Human Resources

The human resources (HR) approach is based on Lundin Gold's established policies and directives, Ecuador's laws and legislative environment, international best practice associated with labour and working conditions, and baseline social conditions within the communities closest to the mine site. The approach also aligns with the Project corporate social responsibility commitments. Information is subject to change as planning advances. Lundin Gold will continue to develop the Project HR policies and procedures after the feasibility stage.

24.2.3 Training

The Project will require approximately 800 employees for the mine and process plant operations (excluding the construction phase). This is broken down into approximately 70% skilled and semi-skilled workers and 30% managers, professionals and technical specialists. Mine operations will require about 95% skilled positions, the backfill paste plant operation will require about 89% skilled positions and the process plant operation will require about 51% skilled positions.

The training strategy for skilled and semi-skilled employees covers four general areas of training: foundation learning courses, on-site technical mining training programs, vendor-specific equipment training, and on-the-job training, using a modular system of programming and certification.

Large-scale mining is an emerging industry in Ecuador, with one large mine currently in construction and four projects in the planning stage. The FDN Project faces several significant challenges, including:

- The lack of modern large scale mining experience in Ecuador from which to draw expertise
- The remote location of the mine presents geographical challenges
- The availability of a skilled and/or adequately educated local work force in surrounding local communities is limited
- Expectations for employment of local people are high.

Due to limited experience in the local population, Lundin Gold anticipates that experienced expatriates will be required to initially fill management, professional and technical specialist positions. Expatriate positions will be transitioned to Ecuadorian employees as they acquire the necessary skills, competencies and experience.

Initially, Lundin Gold will develop the surface and underground mine infrastructure using skilled, experienced contract labour, a significant proportion of whom are expected to be expatriates. This contract labour will mainly be used to construct specialized pieces of infrastructure (e.g. declines, raise bores, process plant). During mine construction and operations, training of local labour will be a priority. Over time, there will be a gradual introduction of local labour and transition to the Owner’s team, phasing out the use of expatriate labour to the extent possible and practicable.

The primary gap between the available labour supply and the Project demand is for mining-specific occupations. Therefore, the employment strategy will focus on training for skilled and semi-skilled work in mine operations, paste backfill plant and process plant operations. Cultivating a work force for an industry currently in its infancy will require communication and outreach that focuses on building occupational and industry awareness so that individuals have sufficient information to make career decisions.

Lundin Gold recognizes the potential risks inherent in developing, recruiting and retaining a skilled work force for the Project. Management and the Human Resources Department will actively manage human resource-related risks through the proposed training programs, HR approach, and policies.

24.3 Risk Management

24.3.1 Risk Management Strategy

Lundin Gold's risk management process has been structured as follows:

- Definition of the risk management methodology and processes for evaluation and mitigation
- Training of the participants in the use of risk management methodology
- Establishing evaluation criteria for probability and impact of risks
- Risk evaluation and classification according to the exposure level
- Review sessions to validate the risk matrix and proposed mitigation actions
- Residual risk evaluation for the risks with greatest exposure
- Development of a Risk Management Plan to be implemented as appropriate.

Risk management will be a continuous process of monitoring and reviewing through all stages of the Project. The risk log will continue to be reviewed during the detailed engineering stage in risk workshops, and reports with updates will be issued periodically. Workshops will be held at the start of each phase, and periodically during each phase.

24.3.2 Risk Workshops

Two Project risk workshops were held, and quarterly updates were conducted by the risk managers. The risk structure breakdown was based on the main project phases and areas

Mitigation strategies were defined and assigned to the risks. Mitigation can either reduce the probability of the risk occurring and/or can reduce the impact of the risk if it occurs. Risks will be managed as follows:

- The risk manager will review the risk at regular intervals to indicate the effectiveness of mitigation and to validate or amend the classification if appropriate
- The FDN Project Manager will be responsible for the risk log which will be issued on a quarterly basis.

24.3.3 Risks Matrix

The top Project risks after mitigation are shown in Table 24-1.

Table 24-1: Risk Rating

Strategic Area	Project Aspect	Phase	Risk Event	Risk Ranking
Process Plant	Permits	Construction	Delays in receipt of permits for construction, in particular for the construction of the process plant, which would affect the EIA, the Project schedule, and the mine and process plan	High
General	Local conditions	Construction	Non-availability of sufficient qualified staff to support the construction efforts, which would initially affect the construction phase, but would also have a run-on effect on the Project schedule	
Human Resources	Local conditions	All	Labour disputes	Moderate
	Local conditions	Construction and operation	Non-compliance with some aspects of labour law	
Mine	Health and Safety	Operation	Rock instability due to hydrogeology and/or geotechnical conditions	
Safety	Health and Safety	Construction	Technical breaches by the contractors	

25.0 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this Report.

25.2 Mineral Tenure, Surface Rights, Royalties and Agreements

- Information from legal experts supports that the mining tenure held is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves
- Lundin Gold through Aurelian has acquired or is in the process of acquiring, the surface rights necessary to support Project construction. Lundin Gold holds sufficient surface rights to allow construction and development of the planned mining-related infrastructure. Additional surface rights, if needed for accesses such as easements, could be acquired through negotiation, or by direct request to the Ministry of Mining to impose easements over the required lands. The QP considers it a reasonable assumption that with continued negotiation, all of the necessary surface rights to support the infrastructure locations planned in the 2016 FS can be obtained
- Although Lundin Gold holds water rights in the Project area, the 2016 FS envisages that no overall water permit for industrial usage for mining activities will be requested, since the water that is proposed to be used will be from secondary, not primary, sources
- There are surface disturbances associated with artisanal workings within the Project area; there is an expectation that environmental contamination will be associated with these sites
- Lundin Gold and the Government of Ecuador have established the fiscal terms and conditions for the development of the FDN Project through the Definitive Exploitation Agreement for the Project
- Additional permits are required for Project development, the most important being execution of the Exploitation Agreement. Lundin Gold is in the process of preparing a Phase Change Application to the Government of Ecuador in respect of the concession which is host to the FDN Project, to advance the concession from economic evaluation to the exploitation phase. Lundin Gold has up to six months subsequent to the approval of the Phase Change Application to execute the Exploitation Agreement. Delays in the execution of the Exploitation Agreement could impact the schedule outlined in the Project execution plan

- To the extent known, Lundin Gold has advised Amec Foster Wheeler that there are no other significant factors and risks that may affect access, title, or the right or ability to perform work on the property that have not been discussed in this Report.

25.3 Geology and Mineralization

- The FDN deposit and many prospects that have been identified in close proximity to the deposit are classified as intermediate sulphidation-style epithermal systems
- Knowledge of the FDN deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization is sufficient to support Mineral Resource and Mineral Reserve estimation
- Geochemical sampling, geological mapping and geophysical surveys have identified a number of anomalies, a portion of which have been drill tested. Additional exploration potential remains within the Project area, and exploration along the Las Peñas fault zone remains the first priority in the region

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

- The quantity and quality of the lithological, geotechnical, collar and downhole survey data collected in the exploration and infill drill programs conducted during the Aurelian, Kinross and Lundin Gold campaigns are sufficient to support Mineral Resource and Mineral Reserve estimation
- No drilling occurred on the Project between 1 December 2015 and 25 April 2016. A new exploration drill program that commenced on 26 April 2016 is focused on key exploration targets outside the La Zarza concession, and is provisionally envisaged as consisting of 20–30 drill holes (7,500–10,500 m), depending on results. As of 24 May 2016, six core holes had been completed on exploration targets within the Princesa and Emperador 1 concessions, for a total of approximately 2,217 m. Assay results are pending
- Drill hole inclinations vary significantly (from -45° to -84°) and the mineralized zones have variable dips from moderate to steep westerly to steep easterly dips. Therefore, most drill holes intersect the mineralized zones at an angle, and the drill hole intercept widths reported for the Project are greater than true widths
- Sample security procedures met industry standards at the time the samples were collected. Current sample storage procedures and storage areas are consistent with industry standards

- Data verification has been extensively conducted by RPA, and no material issues have been identified by those programs. In addition, Aurelian, Kinross and Lundin Gold have regularly used various procedures to verify the quality of the data.
- The lithological and mineralization models have been diligently constructed, and have been prepared using industry-standard practices.
- Data collected have been sufficiently verified that they can support Mineral Resource and Mineral Reserve estimation and be used for mine planning purposes.

25.5 Metallurgical Testwork

- Much of the pre-2012 testwork is not relevant to the Project as a result of a flowsheet concept change. Testwork completed from 2012 to 2016 supports the plant design and metallurgical recovery estimates
- Acceptable gold recoveries were achieved using a GFL (gravity, flotation and leaching circuit) flowsheet. This has resulted in a reasonable overall gold recovery for the GFL option
- Using the MET1 variability results, recovery relationships for gold and silver were developed using the Au/S and Au/Ag ratios of the feed. The relationships developed were confirmed by the results of the MET1 and MET4 composite samples. These relationships were provided to the mine model to support estimation of the Mineral Resources and Mineral Reserves
- The gold recovered in the gravity circuit is important to the planned process, because the proportion of free gold is highly variable throughout the deposit. The gravity circuit will help to stabilize the flotation feed grade
- The two products of the plant, the concentrate and the doré, are saleable without major penalties. The levels of arsenic and mercury in the flotation concentrate are expected to be able to be maintained at acceptable levels.

25.6 Mineral Resource Estimates

- A total of 246 drill holes support the estimate. The most recent drill holes used to estimate Mineral Resources were drilled in 2012
- The estimates were prepared by RPA. 3D solid models of the lithology, degradation, faults and alteration were constructed; compositing, exploratory data analysis including variography; interpolation; statistical validation; and resource classification were completed. Validation of the resulting model was performed. The estimated elements in the model, using an OK estimator, are gold and silver.

Density data average values were assigned to the block model to convert volume to tonnes

- Mineral Resources have had reasonable prospects of eventual economic extraction considerations applied. Mineral Resources were reported at a block cut-off grade of 3.5 g/t Au. Silver was not included in the cut-off grade calculation due to its relatively low grade and small contribution to the value of the mineralization
- Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources have been estimated using standard practices for the industry, and conform to the 2014 CIM Definition Standards
- Factors which may affect the estimates include: metal price and exchange rate assumptions, changes to the assumptions used to generate the cut-off grade value, changes in local interpretations of mineralization geometry and continuity of mineralization zones, density and domain assignments, changes to design parameter assumptions that pertain to stope designs, changes to geotechnical, mining and metallurgical recovery assumptions, assumptions as to the continued ability to access the site, retain mineral and surface rights titles, obtain environmental and other regulatory permits, and obtain the social licence to operate.

25.7 Mineral Reserve Estimates

- The mine plan is based on Indicated Mineral Resources. The Inferred Mineral Resources grades were set to zero for the purposes of Mineral Reserve estimation
- The current Mineral Reserve estimates are based on the most current knowledge, permit status and feasibility-level engineering and operational constraints. Mineral Reserves have been estimated using standard practices for the industry, and conform to the 2014 CIM Definition Standards
- Factors that may affect the Mineral Reserves include: long-term commodity price assumptions, long-term exchange rate assumptions, and long-term consumables price assumptions. Other factors that can affect the estimates include changes to: Mineral Resources input parameters, constraining stope designs, cut-off grade assumptions, geotechnical and hydrogeological factors, metallurgical and mining recovery assumptions, and the ability to control unplanned dilution. Operations will require the grant of the exploitation licence.

25.8 Mining Plan

- The mine plan required consideration of the following key factors:

- The host rock for the deposit appears competent but the resource zone is less competent with a small portion in Poor rock (less than 10%). Geomechanically, the rock mass quality varies from Poor to Fair (RMR range 40 to 55) with the intact rock strength averaging 60 MPa. The deposit is also relatively close to surface (within 140 m of surface in some locations)
- Given the variable conditions likely to be encountered, a range of methods and/or support regimes was considered appropriate for FDN. The primary methods of extraction selected are TS in the better ground conditions and D&F in the more challenging areas
- Use of backfill to reduce the risk of geotechnical failure and maximize extraction
- Proximity of the Machinaza River and dewatering requirements

- The West, Central, and portions of the East Fault are significant fault structures that represent a risk to the stability of an open stoping method and subsequently these areas are considered suitable only for a limited man-entry mining method where conditions can be well controlled such as D&F. Not all structures could be modelled, and the influence of the secondary and tertiary level structures are not well understood in some areas
- Degradation of Suárez Formation conglomerate results in difficult mining conditions that can be mitigated through extraordinary ground support (full shotcrete lining and invert) which will be a high mining cost with slow advance rates
- Stress measurements are not currently available for FDN, and stress assumptions are based on SRK's evaluation of the structural geological setting and the World Stress Map
- The total groundwater inflow will not be large compared with many other mines around the world, and could be dealt with by in-mine pumping, but the combination of the water with poor ground conditions and the mining methods could have an influence on mining productivity. Groundwater inflow risks and potential effects will be managed in multiple ways, including cover and probe drilling, localized grouting, dewatering wells and underground drainage galleries
- SRK recommended TS where there is no Poor domain rock quality. The recommended dimensions for TS are: 12 m wide x 20 m long x 25 m high. Outside the long-hole stoping area, where the COG allows, the D&F mining method will be used. The dimensions provided in SRK's geotechnical recommendations, is a section 4 m wide x 4 m high. Hence D&F was designed with vertical cuts of 4 m each
- The crown pillar will be from the 1240 L (south area of the mine) to the 1270L (north area of the mine). Because of the instability risk associated mainly with the

rock quality, the mining method for these areas will be D&F. A sill pillar was included between the TS horizons 1080L and 1170L at 1155L, which allows for earlier production. The mining method for this sill pillar is TS with a stope height of 15 m (instead of 25 m in the regular stopes)

- The plant feed was estimated from production from the mine as follows:
  - Ore is taken from the mine to the HG, MG and LG stockpiles
  - The stockpiles are planned to be exploited in grade order, with the HG first, then MG, and lastly LG
- The paste plant has been designed for a nominal throughput of 70 m^3/h and will operate at an average utilization rate of approximately 60%. The main pour target strength of 300 kPa will be reached after 14 days with a plug pour target strength of 434 kPa after three days. The nominal design production rate of the CRF plant is 180 m^3/h. The CRF target strength of 3 MPa to 5 MPa will be reached after seven days
- The main ventilation system is designed to accommodate the initial ore production rate and the ramp-up to the required tonnes per day. The ventilation system proposed is a mechanical exhaust ventilation system (pull) where fresh air will enter by suction
- Material handling requirements have been minimized by relying on mobile equipment for transport instead of permanent infrastructure and facilities
- During operations it is planned that FDN will carry out all mining operations. Mine operations will use the same equipment for development for TS and for D&F. Drilling, support, loading and hauling equipment are the same for both methods. Different production equipment is required due to the different opening sizes
- Although specific equipment manufacturers and equipment models are described in this Report, no final equipment selection has been made. The equipment listed is indicative of the type and size of the equipment planned and was used as the basis for the capital cost estimate. Equipment selected is conventional within the industry.

25.9 Recovery Plan

- The flowsheet design is conventional, and assumes a gravity–float–leach approach.
- The grinding circuit is designed at the 85th percentile
- The flotation circuit is capable of recovering the gold associated in sulphides (pyrite). The flotation circuit is able to reduce spikes in sulphide gold grade and

provide a consistent feed to the CIL circuit. Typically, CIL circuits function best on a uniform feed, this can be provided by the combined gravity and the flotation circuits

- The concentrate production rate is expected to be 160 t/d at a feed rate of 3,320 t/d and 140 t/d at a feed rate of 3,500 t/d. The concentrate quality could vary from month to month based on ore variability, mine planning and sequencing as well as the geometallurgy
- The total gold expected to be produced as doré is 1,323 koz during the LOM. The precious metals portion is expected to contain approximately 70% gold and 30% silver
- Equipment proposed is conventional for the type of flowsheet.

25.10 Infrastructure

- A new main access road is planned to support construction and operations. The Project will have limited temporary construction roads that will need to be decommissioned after construction
- The main port for international cargo arrival will be Guayaquil. The Port of Bolivar may be used as an alternative
- On-site buildings will include a truckshop, mine office building with change house, process plant office, electrical room and control room, workshop, canteen, first aid station and fire station, laboratory and camp. The permanent camp facilities will be located adjacent to the main access control to the Project site. The camp facilities will be located at an altitude below 1,500 masl. The camp will accommodate approximately 830 people. The camp facilities will be constructed early in Project construction in order to accommodate construction personnel as the construction manpower grows. The temporary camp will be located alongside the permanent camp and will provide 1,184 beds in tents. It will be closed when operations start
- On-site non-process services such as the camp, greenhouse, sewage treatment plant and mobile equipment will support the operation. There will be fresh water, domestic water and process water systems and a fire detection and protection system. Mobile equipment for maintenance, operations services and transportation includes tractors and loaders for stockpile rehandling, mobile cranes, buses and utility vehicles
- Some support facilities will be located off the main Project site. For the purpose of the 2016 FS, the off-site facilities are considered to be conceptually located 12 km from the main Project site. These facilities include: guard house, electrical room, light vehicle shop, mechanical workshop, warehouse and laydown area, storage

shed, main office building, and canteen. The actual location of the off-site facilities will be determined during a future Project stage

- Lundin Gold is participating in a public infrastructure investment to reinforce the SND matrix in the area, and is contributing to the installation of a transmission line between Taday and Bomboiza. The overall Project power requirements are expected to be met via a 230 kV double circuit transmission line from the Bomboiza substation to the El Pindal substation, near Los Encuentros. This system will be a public transmission line and substation, owned and operated by CELEC EP Transelectric, with installation paid for by Lundin Gold. From the El Pindal substation, a single-circuit, 230 kV dedicated transmission line will be built to feed the Project

25.11 Environmental, Permitting and Social Considerations

25.11.1 Baseline Studies

- Baseline studies have been undertaken to support the FDN Environmental Impact Study (EIS), and subsequent updates to the FDN EIS. Abiota, biota, social and archeological baseline studies were performed.

25.11.2 Waste Rock Storage

- Approximately 2.03 Mt of waste material will be generated. Of this, approximately 1.29 Mt (64%) will be returned underground as part of the backfill management strategy. The remaining 0.74 Mt of material will need to be permanently stored on surface
- An area to the south of the process plant has been allocated to accommodate waste from the underground mine. The waste stockpile has been designed to accommodate both “bad” (must be permanently stockpiled) and “good” (can be used in construction and backfill) waste, and the allocated stockpile area is large enough to accommodate this material
- Geochemical characterization of the waste rock has indicated that there is strong PAG potential. Further work should be undertaken to confirm that the necessary quantity of NAG and/or neutralizing material is available to effectively manage the PAG waste rock in the long term.

25.11.3 Stockpiles

- Three ore stockpiles, segregated by gold grade, will be developed. The area allocated for these stockpiles is close to the crushing station at the process plant on the surface

- All ore on the stockpiles will be fed to the mill by the time of mine closure, and the areas will be rehabilitated at closure.

25.11.4 Tailings Storage Facility

- The facility will be located in the uppermost portion of the valley, to minimize the catchment area and to maximize the separation distance from the Zarza River downstream
- The TSF dam will be raised throughout the service life until reaching the ultimate elevation at 1,493 m. Currently dam raises are contemplated at Years 0, 2, 5, 10 and 14 (ultimate)
- A total of 12.15 Mt of GFL tailings will be pumped to the TSF at 55% solids over the mine life
- A geotechnical characterization assessment of the GFL tailings was completed by KCB. The results of this assessment were based on a sample obtained from an early metallurgical selection program using an existing LOM composite. The results from this program are considered adequate for feasibility study level. If samples from subsequent metallurgical programs become available and if tailings are finer grained than the samples used for the current design, more testing may be required
- Geochemical testwork on the tailings has characterized them as non-acid generating. There may be elevated levels of TSS requiring settlement, and there will be elevated levels of cyanide deposited in the tailings. Cyanide management after treatment will be via a combination of natural degradation from sunlight and dilution from rainfall and other inflows.

25.11.5 Hollín Borrow Pit

- Lundin Gold will need to exploit a borrow pit to provide granular materials for construction and mine backfill, from construction through to mine closure
- The Hollín Borrow Pit will be exploited as an open pit mine with material extracted by ripping and by using explosives
- Rock fill material will be delivered direct to the TSF and seepage pond walls; all other materials will be processed through an aggregate plant, which will use screening and crushing/screening to produce the required products
- Further work should be undertaken to understand the quality of water from the Hollin Borrow Pit and associated waste rock facility, to ascertain the water capture and treatment required from these areas during operations and into closure.

25.11.6 Water Management

- The development of a water management plan (WMP) is of paramount importance for all Project stages from construction and operations to closure
- Four main water types will require management: non-contact, unaffected contact, affected contact and neutral water types. Management includes water treatment plants and water-related infrastructure such as water management ponds, diversion channels, pipes and pumps
- Water quality discharge criteria for the site were established
- A water balance model was constructed and demonstrated that the proposed water management plan at the site is feasible
- A site-wide water quality model was built and demonstrated that the proposed WMP for the site is feasible and will meet regulatory requirements for discharge to the receiving water bodies
- Water quality modelling should be extended to cover the likely closure scenario to confirm the flows and quality of water requiring treatment post-operations, and to predict the number of years water treatment would likely be required once rehabilitation activities have been completed

25.11.7 Closure

- Closure planning has been undertaken to a conceptual level, and will be updated throughout the Project life. A comprehensive plan will be required two years prior to cessation of operations. Regulations require at least five years of post-closure monitoring
- The closure activities cover closure aspects related to environmental factors such as soil, air and water which are directly related to the community health and safety. Aspects related to economic and cultural dynamics of the communities have not been considered in the plan
- A SIMP will be developed and implemented to manage socioeconomic impacts of the Project (both positive and negative) throughout all phases of the Project, including closure.

25.11.8 Permitting

- As a result of changes to the Project description, an additional Update to the granted EIS is required

- Concurrently with the Update to the FDN Mining Project EIS being prepared, Lundin Gold is preparing the associated EIA document
- Permitting requirements were evaluated by project phase, including before construction (16 permits), during construction (six permits), and before operations (three permits)
- The EDC are used throughout the various Project design phases to ensure the FDN Project meets the required and recommended regulations, guidelines and practices.

25.11.9 Social Considerations

- The Project's indirect influence is expected to extend to some neighbouring communities, including the parish of Los Encuentros and two communities from neighbouring parishes
- The Project is not expected to impact any cultural heritage, and strict archaeological protocols are in place
- Although perceptions of artisanal mining are low, the community is very supportive of the Project, and the primary concern is access to employment. There is currently no large-scale mining in Ecuador
- A community relations program has been defined. The PADC is based on the principles of community participation, sustainable development and human development.

25.12 Markets and Contracts

- The Project will produce doré bars and gold–silver concentrate
- No contracts are currently in place for any production from the Project.

25.13 Capital Cost Estimates

- The initial Implementation Phase capital cost is estimated to be US$668.7 million. The sustaining and closure capital is estimated to be US$291.9 million. The total capital construction costs are therefore US$960.6 million.

25.14 Operating Cost Estimates

- The overall LOM operating cost estimate is US$118/t, and includes base costs, non-recoverable taxes, and leasing. Operating costs are estimated at US$414/oz Au, including all site costs.

25.15 Economic Analysis

- Under the assumptions presented in this Report, the Project demonstrates positive economics. The after-tax NPV at a 5% discount rate over the estimated mine life is US$676 million. The after-tax IRR is 15.7%. The after-tax payback of the initial capital investment is estimated to occur 4.5 years after the start of production
- The LOM AISC per ounce is US$623
- The Project is most sensitive to changes in gold price, less sensitive to changes in operating costs, and least sensitive to capital cost and labour cost changes.

25.16 Execution Plan

- The Project schedule entails significant project activity durations, some of which may run concurrently, and includes a duration of 11 months for the engineering, procurement, contracting, and preliminary construction of Early Works (see Section 26 for the recommended work program), 12 months for the construction of the access road and bridge over the Zamora River, 34 months for construction of the twin declines, six months to develop the aggregate borrow pit and plant, nine months for the mass earthworks and 20 months for the construction of the process plant and facilities.
- The implementation strategy for the Project is dictated by the construction of the twin declines. It is possible to build all the surface facilities including the process plant and associated infrastructure during this period
- The construction of the mine access is the critical path and the Early Works to expedite the construction of the access are also critical. The Early Works have been given special attention in the execution plan because they will need to start very soon after approval of the 2016 FS, if the proposed Project schedule is to be met.

25.17 Risks and Opportunities

- Risk registers were prepared in support of development of the Project Closure Plan and as part of overall risk assessments
- The risks rated as “high” included delays in the grant of the EIA or receipt of permits for construction that would affect the Project schedule for the development of the mine and/or process plant, and non-availability of sufficient qualified staff to support the construction efforts

- Moderate risks included labour disputes, issues arising because of non-compliance with local labour laws, technical breaches by contractors, and issues arising from unforeseen geotechnical and hydrogeological parameters.

25.18 Conclusions

Under the assumptions discussed in this Report, the Project returns a positive economic outcome. The decision to initiate construction activities rests with Lundin Gold management. Additional permit grants by the Government of Ecuador will also be required.

26.0 RECOMMENDATIONS

26.1 Introduction

A two-phase work program is proposed for the Project. Phase 1 is designed to provide information to support the Early Works and complete the construction phases associated with the Early Works. Phase 2 comprises additional data collection in specific technical disciplines and will support basic engineering and refinement of the capital cost estimates. The work phases can be carried out in parallel. As information from a discipline area becomes available, it will be reviewed to determine if any revisions to the existing Project assumptions, Early Works or Project schedule may be required.

The Phase 1 program is estimated at US$32.7 million; Phase 2 is provisionally estimated at approximately US$7.9–US$13.7 million.

26.2 Phase 1: Early Works

The main objective of the Early Works program is to provide the infrastructure, services and facilities to support the start of construction of the mine twin declines and to reduce the risk for the basic engineering capital expenditure estimate and remaining earthworks. The scheduled start date for the decline construction is 1 May, 2017 and the Early Works for gaining access to start work on the declines needs to be completed by 30 April, 2017.

The Early Works consist mainly of infrastructure, environmental and preliminary mining works such as the construction of access and on-site roads, platforms, water management infrastructure, the expansion of the existing Las Peñas camp, biotic rescue, archeological rescue, deforestation, survey, geotechnical drilling and tendering of the portal construction and mine contractor contracts. The Early Works are planned to start in June 2016 and will finish on 1 July, 2017, which is the start date proposed for the Implementation Phase.

The Early Works are split for estimation purposes between EPCM and basic engineering-related Early Works activities, and Early Works in support of mine access and implementation:

- EPCM services for Early Works and contracts: drawings; bids; contract awards; engineering support, construction management; early contracts to support Project implementation, including EPC for the access road; bridge over the Zamora River; construction camp: budget estimate approximately US$6 million
- Early Works in support of portal mine access construction: surveys; geotechnical; biotic rescue; storage areas for forest products and topsoil; contractor mobilization; expanding the exploration camp; laydown areas; road construction and upgrades;

mine portal platforms; water management infrastructure: budget estimate approximately US$26.7 million.

26.3 Phase 2: Additional Data Collection

26.3.1 Exploration

Ongoing exploration is planned, and is likely to include:

- Drill testing of known targets
- Follow up drilling on previous results, and identification of new targets
- Geophysics on new areas of interest for target definition based on prospecting activities
- Continued regional exploration to define new areas of interest

The program assumes:

- $200,000 to $300,000 allocation for geophysics
- $100,000 to $150,000 allocation for surface geochemical surveys
- $2.6–$3.2 million allocation for drilling assuming 8,000 m to 10,000 m, and assuming an all-in drilling cost of $320/m

The overall exploration budget to complete the recommendations is estimated at $2.9 million to $3.65 million.

26.3.2 Hydrogeology

Recommendations for future hydrogeology work include:

- Complete large-scale (minimum one month duration at maximum possible pumping rate) pumping tests at two of the proposed dewatering well locations
- Collect additional water samples from the pumping tests and confirm water quality for discharge to the Machinaza River
- Conduct hydraulic testing in any planned geotechnical drill holes
- Re-calibrate the groundwater numerical model based on results of the pumping tests and revise estimates of groundwater inflow
- Revise design of dewatering wells based on results of the pumping tests
- Optimize the underground groundwater management plan based on results of the re-calibrated groundwater model and any updates to the mine plan

- Continue monitoring existing groundwater instrumentation and collection of groundwater samples.

The program is estimated at US$1.5–US$2.5 million.

26.3.3 Geotechnical

Recommendations for future geotechnical work include:

- Complete early works drilling to complete investigations at the North Portal box-cut and decline, North Decline spiral area, and North and South ventilation shafts
- Further optimization of the global mine extraction sequence to reduce areas with longer duration temporary access excavation
- Implement the underground cover drilling program as the twin declines are advanced to verify geotechnical and hydrogeological conditions under the Machinaza River, at the base of the North and South shafts, workshop location, and east infrastructure areas
- Update ground support designs based on the results of underground cover drilling program
- Complete insitu stress testing during the early construction phases to verify assumptions made on the pre-mining stress field, and adjust design if required.

The program is estimated at US$2–US$3 million.

26.3.4 Process

Recommendations in support of further flowsheet development include:

- The application of centrifugal gravity concentration to either the flotation rougher concentrate or the cleaner scavenger tailings to improve the recovery of fine free gold. The recovered fine gold could be added to the intensive leach reactor to produce doré or the final flotation concentrate, resulting in a higher-grade product
- The foundation recommendations for the process plant are still at the feasibility stage because drilling has not been done in the deepest part of the excavation to confirm that the soil profile is as expected.

The program is estimated at US$0.5–US$1.5 million.

26.3.5 Tailings Management

Recommendations in support of TSF design and tailings management include:

- Hydrogeological study of the TSF area to judge whether high groundwater pressures can result from blinding of the upper elevations by the tailings deposit
- Establish monthly monitoring of all major streams in the tailings area
- Investigations along the upstream toe of the dam for the detailed design of the cut-off trench and upstream toe of the starter dam
- Review the potential for alternative geosynthetic filters for the dam
- Compaction test fill to establish the true compaction characteristics of the saprolite
- Investigation of the sorption capacity of the saprolite clays in the TSF basin
- Geotechnical investigations to confirm undrained shear strength of foundation materials with depth
- Preparation of lining system sequencing, construction drawings and specifications.

The program is estimated at US$0.5–1.5 million.

26.3.6 On-Site Infrastructure and Services

- Further field investigations should be executed in the next phase of the Project to confirm the geotechnical and hydrogeological conditions on the northern side of the process plant site and to investigate other on-site facilities including the paste plant, aggregate crushing plant, temporary camp sites, soil stripping stockpiles and Hollín Borrow Pit waste dump
- The layout and size of the paste plant presents an opportunity for improvement.

The program is estimated at US$0.5–1.5 million.

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