SENET

A DRA Global Group Company

EUROSUN MINING

NI 43-101 TECHNICAL REPORT ON THE ROVINA VALLEY PROJECT IN ROMANIA

Prepared for
EURO SUN MINING INC.

Prepared by
NEW SENET (PTY) LTD

Effective Date
14 April 2021

DEFINITIVE FEASIBILITY STUDY

SP0829 Rovina Valley NI 43-101 Technical Report Rev0

DATE AND SIGNATURE PAGE

This report titled "NI 43-101 Technical Report on the Rovina Valley Project in Romania" was prepared for Euro Sun Mining Inc. by New SENET (Pty) Ltd. The report is compliant with the Canadian National Instrument 43-101 (NI 43-101) and Form 43-101F, and was signed by the following Qualified Persons:

Nicholas Dempers
MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr Eng (RSA), Reg. No. 20150196, FSAIMM (RSA)
New SENET (Pty) Ltd
14 April 2021
Johannesburg, South Africa

David Alan Thompson
B-Tech, Pr Cert Eng, SACMA
Pr Eng (RSA), Reg. No. 201190010
DRA Projects (Pty) Ltd
14 April 2021
Johannesburg, South Africa

Sivanesan Subramani
BSc (Hons) Geology and Economic Geology
Caracle Creek International Consulting MinRes (Pty) Ltd
14 April 2021
Johannesburg South Africa

Robert Cross
MEng BASc
P.Eng. (ON): Reg. No. 100173823
P.Geo. (ON): Reg. No. 2845
Klohn Crippen Berger
14 April 2021
Toronto Canada

Carlos Diaz Cobos
MASc, BEng
P.Eng.(ON): 100191866
Klohn Crippen Berger
14 April 2021
Sudbury Canada

Andrew Hovey
BSc Earth Sciences (Hons)
RPGeo (AIG) Reg. No. 4202
Klohn Crippen Berger
14 April 2021
Brisbane Australia

Richard W. Lawrence

BSc (Mining Engineering)

PhD (Biohydrometallurgy)

P.Eng. (BC, Canada) Reg. No. 22564

Lawrence Consulting Ltd

14 April 2021

Mill Bay, Canada

Kevin Leahy

BSc (Hons) Geological Sciences

PhD. (Diamond Exploration)

CGeol (UK) Reg. No. 1005123

Environmental Resource Management Ltd

14 April 2021

Oxford, United Kingdom

The effective date of the NI 43-101 Technical Report is 14 April 2021, which is the cut-off date for all scientific and technical information included in the recently completed Rovina Valley Project Definitive Feasibility Study. The effective date of the mineral resource estimate in the Technical Report prepared by Caracle Creek International Consulting MinRes (Pty) Ltd is 14 April 2021.

CERTIFICATE OF QUALIFIED PERSON – NICHOLAS DEMPERS

I, Nicholas Dempers, do hereby certify that

1. I am a principal process engineer at New SENET (Pty) Ltd, Building 12, Greenstone Hill Office Park, Emerald Boulevard, Greenstone Hill, Greenstone 1609, Modderfontein, Gauteng, South Africa.
2. I am a reviewer of the report titled "NI 43-101 Technical Report on the Rovina Valley Project in Romania", prepared for Euro Sun Mining Inc., with an effective date of 14 April 2021.
3. I am a graduate of the University of Cape Town, with a BSc in Chemical Engineering. I also hold a MSc in Chemical Engineering from the University of Cape Town and a BCom from the University of South Africa.
4. I am a registered professional member of the Engineering Council of South Africa (Reg. No. 20150196), and I am a fellow of the Southern African Institute of Mining and Metallurgy.
5. I have practised my profession continuously since 2001. I have over 19 years' experience in the minerals industry. I have been involved in the process operation (production) and plant design, from conceptualisation to complete project execution, of more than 10 mineral process projects, as well as more than 12 process plant studies for major commodities including cobalt, copper, gold, uranium, rare earths, and platinum group metals (PGMs). I have assisted in or compiled National Instrument 43-101 (NI 43-101) Reports for various projects that have been listed on the TSX stock exchange.
6. I have read the definition of "qualified person" set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
7. I have not visited the Rovina Project Site.
8. I performed consulting services and reviewed files and data supplied by Euro Sun Mining Inc. between 28/04/2020 and 30/03/2021.
9. I am responsible for the preparation of Sections 13, 17, 18, 21 and 22 and contributed to Sections 1, 24, 25 and 26 of the Technical Report.
10. I have had no previous involvement with this project or any other project on this property.
11. I am independent of Euro Sun Mining Inc. as independence is described in Section 1.5 of NI 43-101. I do not have nor do I expect to receive a direct or indirect interest in the Mineral Properties of Euro Sun Mining Inc., and I do not beneficially own, directly or indirectly, any securities of Euro Sun Mining Inc. or any associate or affiliate of such company.
12. I have read NI 43-101 and Form 43-101F1, and the part of the Technical Report for which I am responsible has been prepared in compliance therewith.
13. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to ensure that the Technical Report is not misleading.

Signed at New SENET (Pty) Ltd, Johannesburg, South Africa on 14 April 2021

NICHOLAS DEMPERS

MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr Eng (RSA), Reg. No. 20150196, FSAIMM (RSA)

CERTIFICATE of QUALIFIED PERSON – DAVID ALAN THOMPSON

I, David Alan Thompson, B-Tech, Pr Cert Eng, SACMA do hereby certify that:

1. I am a principal mining engineer for DRA Projects (Pty) Ltd of 3 Inyanga Close, Sunninghill, Johannesburg, South Africa.
2. This certificate applies to the report titled “NI 43-101 Technical Report on the Rovina Valley Project in Romania”, prepared for Euro Sun Mining Inc., with an effective date of 14 April 2021.
3. I am a graduate of the University of Johannesburg with a Baccalaureus Technologies Degree in Mining Engineering. I have worked as a mining engineer for a total of 33 years and for 11 years since my B-Tech graduation.
4. I am a member of the Engineering Council of South Africa (No. 201190010) and a current member of the South African Colliery Managers’ Association (5066).
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6. I am co-ordinating author of the Technical Report, and co-author responsible specifically for Sections 1.6, 1.7, 1.8, 1.9, 1.10, 2.4, 15, 16, 21, 22, 25 and 26, unless subsections are specifically identified by another Qualified Person.
7. I visited the property from 09 to 13 November 2020, and I have inspected all the relevant areas of interest with regard to the project site and reviewed all the technical documentation available for the project to date.
8. I am independent of Euro Sun Mining Inc. applying all the tests in Section 1.5 of NI 43-101.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and Form 43-101F1; the sections of the Technical Report I am responsible for have been prepared in compliance with that Instrument and form.
11. As of the effective date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed at DRA Projects (Pty) Ltd, Johannesburg, South Africa on 14 April 2021

DAVID ALAN THOMPSON
B-Tech Mining, SACMA, ECSA 201190010

CERTIFICATE OF QUALIFIED PERSON – SIVANESAN SUBRAMANI

I, Sivanesan Subramani, do hereby certify that

1. I am a principal mineral resource geologist at Caracle Creek International Consulting MinRes (Pty) Ltd, 90 Beryl Avenue, Bramley North, Sandton, 2090, Gauteng, South Africa.
2. I am a co-author of the report titled “NI 43-101 Technical Report on the Rovina Valley Project in Romania”, prepared for Euro Sun Mining Inc., with an effective date of 14 April 2021.
3. I am a graduate of the University of KwaZulu Natal, with a BSc Honours in Geology and Economic Geology.
4. I am a registered professional member of the South African Council for Natural Scientific Professions (Reg. No. 400184/06). I am a member of the Geological Society of South Africa, and a member of the Geostatistical Association of Southern Africa.
5. I have practised my profession continuously since 1995. I have over 25 years' experience in the exploration and mining industry. I have been involved in mineral resource estimation and compilation of technical reports since 2005.
6. I have read the definition of “qualified person” set out in NI 43-101 and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101), and relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
7. I visited the Rovina Valley Project Site from 9 to 12 November 2020.
8. I am responsible for the preparation of Sections 4, 5, 6, 7, 8, 9, 10, 11, 12 and 14, and contributed to Sections 1 and 26 of the Technical Report.
9. I have had no previous involvement with this project or any other project on this property.
10. I am independent of Euro Sun Mining Inc. as independence is described in Section 1.5 of NI 43-101. I do not have nor do I expect to receive a direct or indirect interest in the Mineral Properties of Euro Sun Mining Inc., and I do not beneficially own, directly or indirectly, any securities of Euro Sun Mining Inc. or any associate or affiliate of such company.
11. I have read NI 43-101 and Form 43-101F1, and the part of the Technical Report for which I am responsible has been prepared in compliance therewith.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to ensure that the Technical Report is not misleading.

Signed at Caracle Creek International Consulting MinRes (Pty) Ltd, Johannesburg, South Africa on 14 April 2021

SIVANESAN SUBRAMANI
BSc Hons (Geology), Pri.Sci.Nat (400184/06)

CERTIFICATE OF QUALIFIED PERSON – ROBERT CROSS

I, Robert Cross, P.Eng., P.Geo., as an author of this report entitled "Technical Report on the Rovina Valley Project" prepared for Euro Sun and dated April 14, 2021, do hereby certify that:

1. I am a Geological Engineer and the Toronto Office Manager with Klohn Crippen Berger Ltd, located at 801 – 120 Adelaide Street West, Toronto, ON.
2. I am a graduate of the University of British Columbia in 2007 with a Bachelor of Applied Science in Geological Engineering, and a graduate of the University of British Columbia in 2009 with a Master of Engineering.
3. I am registered as a Professional Geoscientist in the Province of Ontario (Reg #2845) and as a Professional Engineer in the Province of Ontario (Reg. #100173823). I have worked as a geological engineer for a total of 14 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Geotechnical site characterization;
   - Geotechnical engineering of tailings and water dams, foundations, and rock support;
   - Design and construction projects, construction monitoring, field investigations and reviews for project sites in North America, South America, Central Asia and Africa.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Rovina Valley Project site from November 1 to 14, 2020.
6. I am responsible for Section 18.3 and I share responsibility with my co-authors for Sections 1, 24, 25, and 26 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Sections for which I am responsible in the Technical Report contains/contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed at Klohn Crippen Berger Ltd on 14 April 2021

ROBERT CROSS
P.Eng., P.Geo.

CERTIFICATE OF QUALIFIED PERSON – CARLOS A DIAZ COBOS

I, Carlos A. Diaz Cobos, P.Eng., as an author of this report entitled "Technical Report on the Rovina Valley Project" prepared for Euro Sun and dated April 14, 2021, do hereby certify that:

1. I am a Civil Engineer with Klohn Crippen Berger at Unit 101, 1361 Paris Street, Sudbury, ON, P3E 3B6.
2. I am a graduate of Universidad Pontificia Bolivariana, Bucaramanga, Colombia, in 2001 with a Bachelor of Engineering in Civil Engineering. I am also a graduate of University of Toronto, ON, Canada, in 2005, with a Master of Applied Science.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #100191866). I have worked as a Civil Engineer for a total of 11 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Review and report as a consultant on several operational and closed tailings facilities for due diligence and regulatory requirements.
   - Prefeasibility Study project work for mining projects in Romania and Mauritania.
   - Prefeasibility and Feasibility Studies for rehabilitation of dams and water management structures associated with tailings storage facilities around the world.
   - Engineering support during implementation and construction phases of projects.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Rovina Valley Project.
6. I am responsible for Section 18.4 and I share responsibility with my co-authors for Sections 1, 24, 25, and 26 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Sections for which I am responsible in the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed at Klohn Crippen Berger on 14 April 2021

Carlos A. Diaz

CARLOS A. DIAZ COBOS
P.Eng.

CERTIFICATE OF QUALIFIED PERSON – ANDREW HOVEY

I, Andrew Hovey, RP.Geo., as an author of this report entitled “Technical Report on the Rovina Valley Project” prepared for Euro Sun and dated April 14, 2021, do hereby certify that:

1. I am a Hydrogeologist with Klohn Crippen Berger at 1-154 Melbourne Street, South Brisbane, QLD, Australia, 4001.
2. I am a graduate of the University of Queensland, Brisbane, in 2001 with a Bachelor of Earth Sciences with Honours.
3. I am registered as a Professional Geoscientist in Australia with the Australian Institute of Geoscientists (Reg. #4202). I have worked as a Hydrogeologist for a total of 20 years since my graduation. My relevant experience for the purpose of the Technical Report is:
    - Sub-surface site investigation and data collation at greenfield mining project sites internationally to support hydrogeological studies and engineering design.
    - Prefeasibility Study project work for mining projects internationally.
    - Involvement in the prior Pre-Feasibility Study for the Rovina Valley project (as project manager and hydrogeologist).
    - Technical contributions to studies for groundwater impact assessment, pit dewatering/depressurisation design and mine waste residue storage.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I previously visited the Rovina Valley Project as part of the Pre-Feasibility Study, but not as part of the current DFS.
6. I share responsibility as co-author of Section 18.4 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Sections for which I am responsible in the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed at Klohn Crippen Berger on 14 April 2021

ANDREW HOVEY
RPGeo

CERTIFICATE OF QUALIFIED PERSON – KEVIN LEAHY

I, Kevin Leahy, BSc (Hons), PhD, CGeol, SiLC do hereby certify that

1. I am a geologist and Technical Director with Environmental Resources Management Ltd located at 2nd Floor, Exchequer Court, 33 St Mary Axe, London, UK. EC3A 8AA
2. This certificate applies to the technical report titled "NI 43-101 Technical Report on the Rovina Valley Project in Romania", prepared for Euro Sun Mining Inc., with an effective date 14 April 2021.
3. I am a graduate from University of Leeds, UK, where I obtained a degree in Geological Sciences in 1992, and a PhD on the subject of diamond exploration in 1996.
4. I am a registered Fellow of The Geological Society, Burlington House, Piccadilly, London, UK, and have been a Chartered Geologist there since 2005. I am also a Registered Suitably Qualified Person and Specialist in Land Condition in the UK Land Forum National Quality Mark Scheme since 2017.
5. My relevant work experience includes geological exploration, environmental impact assessment, mine audit and land contamination remediation and closure planning on numerous mines, processing plants and smelters over my 25-year career:

    - Geological exploration for minerals in South Africa, Equatorial Guinea, Sweden and Canada, as well as a structural geologist for numerous hydrocarbon exploration projects. In the last five years, I jointly developed a hydro-geochemical exploration tool, including several projects in Europe, the US and Australia.
    - Environmental Impact Assessment on dozens of EIA projects in Europe, Africa and Asia, on a variety of mineral targets and deposit types. My role in EIA projects is usually as topic lead on soils and geology, also contributing to surface and groundwater and early closure planning.
    - Mine audits on several sites in Europe, Asia and South America both for transaction due diligence and for compliance with environmental standards: internal, national and international.
    - Closure planning and land contamination projects on several mine, smelter and processing sites in Europe.
    - Participation in several NI 43-101 Technical Reports, including as a Qualified Person for a project in Serbia.

6. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and certify that, by reason of my education, affiliation with a professional association, and relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
7. I am independent of ESM, applying all the tests in Section 1.5 of NI 43-101. I do not have nor do I expect to receive a direct or indirect interest in the mineral properties of Euro Sun Mining Inc., and I do not beneficially own, directly or indirectly, any securities of Euro Sun Mining Inc. or any associate or affiliate of such company.
8. I am responsible for the preparation of Section 20. I am also responsible for the relevant portions of Sections 1, 25 to 27 of the Technical Report.
9. I have not personally visited the property that is the subject of the Technical Report due to Covid-19 restrictions. However, several members of my team visited the site for baseline surveys in 2020.

10. I have had no prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
12. As at 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 the portions of the Technical Report for which I am responsible not misleading.

Signed at Environmental Resource Management on 14 April 2021

KEVIN LEAHY
BSc (Hons), PhD, CGeol, SiLC

CERTIFICATE OF QUALIFIED PERSON – RICHARD W. LAWRENCE

I, Richard W. Lawrence, do hereby certify that

1. I am the Principal of Lawrence Consulting Ltd, 942 Lilmac Road, Mill Bay, British Columbia, Canada.
2. This certificate applies to the report titled “NI 43-101 Technical Report on the Rovina Valley Project in Romania”, prepared for Euro Sun Mining Inc., with an effective date of 14 April 2021.
3. I am a graduate of the University of Wales (Cardiff), with a BSc in Mining Engineering (1970) and a PhD in Mineral Processing/Biohydrometallurgy (1975).
4. I am a registered professional member of the Engineering and Geoscientists of British Columbia, Canada, Registration No. 22564. I have practised my profession continuously since 1975 and have over 40 years’ experience in the minerals industry, particularly in geochemical assessments of mine sites.
5. I have read the definition of “qualified person” set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association as defined in NI 43-101 and relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6. I visited the Rovina Valley Project Site in June 2011 as part of an earlier study. I have inspected all the areas of interest at the project site and have reviewed all the technical documentation available relevant to my involvement in the project to date.
7. I have managed and reported on several geochemical laboratory studies for the project and am responsible for the preparation of Section 18.3.3, and for contribution to Section 1.8 of the Technical Report.
8. I am independent of Euro Sun Mining Inc., as specified in Section 1.5 of NI 43-101. I do not have nor do I expect to receive a direct or indirect interest in the mineral properties of Euro Sun Mining Inc., and I do not beneficially own, directly or indirectly, any securities of Euro Sun Mining Inc. or any associate or affiliate of such company.
9. I have read NI 43-101 and Form 43-101F1, and the part of the Technical Report for which I am responsible has been prepared in compliance therewith.
10. As of the effective date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed at Lawrence Consulting Ltd on 14 April 2021

RICHARD W. LAWRENCE
BSc (Mining Engineering), PhD (Mineral Processing/Biohydrometallurgy), P.Eng. (Reg. No. 22564)

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Unit	Description
%	per cent
% m/m	percentage mass per mass
% w/w	percentage weight per weight
µg	microgram
µm	micrometre (micron)
µS	microsiemen
°C	degree Celsius
a	annum
cm	centimetre
d	day
dB	decibel
dmt	dry metric tonnes
g	gram
g/t	gram per tonne
Ga	billion years (10^9 years)
h	hour
ha	hectare
Hz	hertz
kg	kilogram
km	kilometre
km^2	square kilometre
koz	thousand troy ounces
kV	kilovolt
kVA	kilovolt ampere
kW	kilowatt
kWe	kilowatt
kWh	kilowatt hour
lb	troy pound
L	litre
m	metre
masl	metre above sea level
Mlb	million troy pounds
min	minute
mm	millimetre
Moz	million troy ounces
MPa	megapascal

Unit	Description
Mt	million tonnes
MW	megawatt
N	newton
Nm	newton metre
NTU	Nephelometric Turbidity Unit
oz	troy ounce
Pa	pascal
Pa s	pascal second
ppb	part per billion
ppm	part per million
s	second
s^{-1}	reciprocal second
t	metric tonne
V	volt

It is noted that, throughout the report, table columns might not add up due to rounding.

LIST OF ABBREVIATIONS

Abbreviation	Description
AACE	Association for the Advancement of Cost Engineering
AAS	Atomic absorption spectroscopy
ABA	Acid-base accounting
AGP	AGP Mining Consultants Inc.
Ai	Abrasion index
ARD	Acid rock drainage
BBWi	Bond ball work index
BFA	Bench face angle
BOQ	Bill of quantities
BRWi	Bond rod work index
C&I	Control and instrumentation
CAGR	Compound annual growth rate
CAPEX	Capital cost
CCIC MinRes	Caracle Creek International Consulting MinRes (Pty) Ltd
CCTV	Closed-circuit television
CDF	Co-disposal facility
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRM	Certified reference material
CWi	Crushability work index
DCF	Discounted cash flow
DFS	Definitive Feasibility Study
E&I	Electrical and instrumentation
E, C&I	Electrical, control and instrumentation
EC	European Commission
ECSA	Engineering Council of South Africa
EIA	Environmental Impact Assessment
EP	Equator Principle
EPC	Engineering, procurement and construction
EPCM	Engineering, procurement and construction management
ERM	Environmental Resource Management
ESG	Environmental, Social and Governance
ESIA	Environmental and Social Impact Assessment
ESM	Euro Sun Mining Inc.
ESMP	Environmental and Social Management Plan
EU	European Union
FCF	Free cash flow

Abbreviation	Description
FEL	Front-end loader
FoS	Factor of safety
G&A	General and administration
GIS	Geographic information system
GPS	Global positioning system
HAZOP	Hazard and operability
HDPE	High-density polyethylene
HR	Human resources
IBC	Intermediate bulk container
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma – Atomic Emission Spectroscopy
IEC	International Electrotechnical Commission
IFC	International Finance Corporation
I/O	Input/output schedules
IP	Induced Polarisation
IRR	Internal rate of return
ISO	International Organization for Standardization
IT	Information technology
IUCN	International Union for Conservation of Nature
KCB	Klohn Crippen Berger
LAN	Local area network
LBMA	London Bullion Market Association
LCT	Locked cycle test
LME	London Metal Exchange
LOM	Life of mine
LV	Low voltage
M&I	Measured and Indicated (resource)
MCC	Motor control centre
MCP	Mine Closure Plan
MEL	Mechanical equipment list
MIA	Mining industrial area
MIBC	Methyl isobutyl carbinol
ML	Metal leaching
MTO	Material take-off
MV	Medium voltage
NAMR	National Agency for Mineral Resources (Romania)
NGL	Natural ground level

Abbreviation	Description
NGO	Non-Government Organisation
NI 43-101	Canadian Securities Administrators' National Instrument (NI) 43-101
NPV	Net present value
OEM	Original equipment manufacturer
OK	Ordinary kriging
OMC	Orway Mineral Consultants
OPEX	Operating cost
ORP	Operational Readiness Plan
P&G	Preliminary and general
P&ID	Piping and instrumentation diagram
PAG	Potentially Acid Generating
PAS	Process automation system
PEA	Preliminary Economic Assessment
PLC	Programmable logic controller
PM	Particulate matter
PSD	Particle size distribution
PUZ	Plan Urbanistic Zonal
QA	Quality assurance
QC	Quality control
QP	Qualified person
R&R	Rest and relaxation
RFBP	Request for budget pricing
ROM	Run of mine
RVP	Rovina Valley Project
SAG	Semi-autogenous grinding
SAIMM	Southern African Institute of Mining and Metallurgy
SAMAX	SAMAX Romania SRL
SCADA	Supervisory control and data acquisition
SG	Specific gravity
SGS	SGS Lakefield
SHE	Safety Health Environmental
SMC	SAG Mill Comminution
SMPP	Structural, mechanical, plate work and piping
SWM	Surface Water Management
TSS	Total suspended solids
TSX	Toronto Stock Exchange
UCS	Uniaxial compressive strength

Abbreviation	Description
UPS	uninterruptible power supply
UTM	Universal Transverse Mercator
VGF	Vibrating grizzly feeder
VSD	Variable-speed drive
WHO	World Health Organisation
WMF	Waste management facility
XRF	X-ray fluorescence

1 SUMMARY

1.1 INTRODUCTION

This NI 43-101 Technical Report was compiled by NEW SENET (PTY) Ltd (SENET) for Euro Sun Mining Inc. (ESM) with contributions from the Qualified Persons as set out in Table 1.1 to support ESM's press release dated 01 March 2021 and to summarise the results of the definitive feasibility study (DFS) of the Rovina Valley Project (RVP).

Table 1.1: Qualified Persons and Their Contributions

Qualified Person	Company	Contribution
Nicholas Dempers	SENET (South Africa)	Metallurgical test work interpretation Processing plant and project infrastructure Economic evaluation Coordination and compilation of report
David Alan Thompson	DRA Projects (South Africa)	Mining, mineral reserves
Sivanesan Subramani	Caracle Creek International Consulting MinRes (CCIC MinRes) (South Africa)	Geology and mineral resources
Robert Cross	Klohn Crippen Berger (KCB) (Canada)	Waste storage facility Geotechnical
Carlos Diaz Cobos	KCB (Canada)	Hydrotechnical
Andrew Hovey	KCB (Canada)	Hydrogeology
Richard W. Lawrence	Lawrence Consulting Ltd	Geochemistry
Kevin Leahy	Environmental Resource Management (ERM) (United Kingdom)	Environmental Social assessment Permitting

The RVP is located in the Hunedoara County in Western Romania, approximately 300 km northwest of the capital city of Bucharest and 140 km east of the city of Timisoara.

The RVP consists of three deposits: Rovina to the north, Colnic centrally and the Ciresata deposit to the south. The March 2021 DFS only incorporates the Rovina and the Colnic deposits; the Ciresata deposit can be brought into the project for development later. The Rovina exploration licence is held by SAMAX Romania SRL (SAMAX) a Romanian registered company which is a wholly owned subsidiary of ESM. Since November 2018, ESM has possessed an exploitation permit and mining licence with a renewable 20-year validity.

The RVP is planned to be mined with a standard open-pit mining method using trucks and a hydraulic loader. The open-pit mining operation is anticipated to last approximately sixteen and a half years, during which the lower-grade material will be stockpiled on a pad close to the primary crusher location for treatment over another eighteen months.

Over the life of the project, it is expected that 133.4 Mt of ore will be mined. Of this ore, 119.4 Mt will be delivered to the processing facility, and 14 Mt low-grade ore will be stockpiled for future processing. A total of 246.7 Mt of material will be mined and placed on the waste facility, representing a life of mine stripping ratio of 1.85:1.

A total of 120,256 m of diamond core drilling was done between 2006 and 2012 through the porphyry deposits.

This report sets out the Mineral Resource Estimate, Mineral Reserve Estimate, production schedules, and the capital and operating expenditures over the life of the project. The report culminates in a full economic analysis of the project's value.

1.2 QUALIFICATIONS OF QUALIFIED PERSONS

The relevant sections of this NI 43-101 Technical Report were compiled by the consultants' Qualified Persons, as this term is defined in NI 43-101. The certificates of the Qualified Persons (QPs) follow the Date and Signature Page of this report. A summary of their qualifications and responsible sections, and whether they conducted site visits or not, is given in Table 1.2.

Table 1.2: Summary of the Qualifications and Responsibilities of the QPs

QP	Qualification	Company	Site Visit	Responsibility (Section of Report)
Nicholas Dempers	MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr Eng (RSA), Reg. No. 20150196, FSAIMM (RSA)	SENET	No	13, 17, 18, 21, 22 and parts of 1, 24, 25 and 26
David Alan Thompson	B-Tech (Mining Engineering)	DRA Projects	Yes	15 and 16 ,1, 21 and parts of 25 and 26
Sivanesan Subramani	BSc Honours (Geology and Economic Geology)	CCIC MinRes	Yes	4, 5, 6, 7, 8, 9, 10, 11, 12, 14, and parts of 1 and 26
Robert Cross	MEng, BASc P.Eng. (ON): Reg. No. 100173823 P.Geo. (ON): Reg. No. 2845	KCB	Yes	18.3, and parts of 1, 24, 25 and 26
Carlos Diaz Cobos	MASc, BEng P.Eng. (ON): 100191866	KCB	No	18.4, and parts of 26
Andrew Hovey	BSc Earth Sciences (Hons) RPGeo (AIG) Reg. No. 4202	KCB	No	Parts of 18.4
Richard W. Lawrence	BSc (Mining Engineering) PhD (Biohydrometallurgy) P.Eng. (BC, Canada) Reg. No. 22564	Lawrence Consulting Ltd	Yes	18.3.3, and contributions to 1.8
Kevin Leahy	BSc Honours (Geological Sciences) PhD (Diamond Exploration) CGeol (UK) Reg. No. 1005123	ERM	Yes	20 and contributions to 1, 24, 25 and 26

1.3 PROPERTY DESCRIPTION

Regionally, the RVP is in the Golden Quadrilateral Mining District of the South Apuseni Mountains in west-central Romania, approximately 300 km northwest of the city of Bucharest, and 140 km east–northeast of the city of Timisoara. Locally, the property is approximately 25 km north of the city of Deva, which is the administrative centre for the county, and 7 km east of the town of Brad for which mining has played an important role.

The Golden Quadrilateral has a long history of gold mining, which predates the Roman occupation. Results of modern exploration efforts have defined two other advanced-stage gold projects, Rosia Montana (Gabriel Resources) and Certej (Eldorado Gold). From the Rovina Licence, Rosia Montana is approximately 25 km northeast, and Certej is 17 km southeast.

1.4 OWNERSHIP OF THE PROPERTY

The Rovina property consists of one Exploitation (Mining) Licence (the Rovina Exploitation Licence, Number 18174/2015 for Cu-Au), covering an area of approximately 2,768 ha. ESM, through intermediary subsidiaries, owns 100 % of SAMAX, which in turn owns 100 % of the Rovina Exploitation Licence. The Rovina Exploitation Licence was ratified by the Romanian Government on the 9 November 2018 and is valid for 20 years, starting on 16 November 2018, and renewable for periods of five years. Upon any production, SAMAX must pay a 5 % to 6 % royalty for copper and gold, respectively, to the Romanian Government.

ESM does not hold any surface rights on their Rovina property. Romanian law does not vest surface rights with mineral rights and any proposed development requires the developer to either purchase the surface rights or enter into an appropriate agreement with the surface rights owners to have access to the property. According to Romanian Mining Law, upon conversion of their Exploration Licences to an Exploitation Licence, ESM has the right to legally acquire these rights through one of the following processes:

- Sale
- Land exchange
- Rental
- Expropriation, if in the public or national interest
- Application
- Association with an existing owner
- Other process allowed by law

Numerous private individual local landowners and the state forestry hold surface rights over the Rovina, Colnic, and Ciresata deposits. ESM has initiated a land acquisition programme with three phases:

1) Public information campaign
2) Surveying and registration of land parcels not officially registered with the local government cadastral map
3) Acquisition of surface rights using one of the methods listed above

This programme is being implemented by SAMAX through a Social-Community Relations Manager, Legal Team, and Survey Team. The information campaign is well advanced, informing landowners of the legal process of cadastral registration through public postings and public meetings held between 2012 and 2015. Owing to the large number of unregistered land parcels, SAMAX anticipates 1.5 years from the completion of the identification and surveying of all plots to complete all cadastral registrations prior to implementing a land acquisition strategy.

1.5 GEOLOGY AND MINERAL DEPOSIT

The Rovina, Colnic, and Ciresata porphyry deposits are the principal targets for exploration and mining within the RVP, with their locations defining a north-northeast trend. The Rovina porphyry is the northern-most deposit with the Colnic porphyry lying approximately 2.5 km south of the Rovina porphyry, and the Ciresata porphyry approximately 4.5 km south of the Colnic porphyry.

1.5.1 Geology

On a regional level, most of the mineral deposits in the Romanian region are located in the Carpathian Fold Belt; an arcuate orogenic belt which is part of a much larger belt extending westward into Austria and Switzerland and south into Serbia and Bulgaria. These belts developed during the late Cretaceous and Tertiary periods, following closure of the Tethys Ocean due to the collision of continental fragments of Gondwana with continental Europe and the related subduction of small, intervening oceanic basins. The development of the Carpathian Fold Belt was accompanied by widespread igneous activity, including a suite of late Cretaceous to early Eocene acidic to intermediate intrusive and extrusive rocks, known as “banatites”. These rocks are believed to have formed during the early stages of subduction and are host to several Cu-Mo-Fe porphyry and skarn deposits.

The South Apuseni Mountains represent a somewhat isolated massif of volcanism and ore deposits within the Carpathian orogenic belt. The southern portion of the Apuseni Mountains, where the RVP is located, consists of a complex area of Palaeozoic (and older) metamorphic rocks, Mesozoic ophiolites and sedimentary rocks, and Tertiary igneous and sedimentary rocks.

On a local level, the property covers a sequence of Neogene-aged subvolcanic intrusive rocks, which in other parts of the Golden Quadrilateral host epithermal and porphyry-style mineralization. ESM’s exploration programmes have identified Au-rich porphyry systems (the Rovina, Colnic, and Ciresata deposits) hosted by these Neogene subvolcanic intrusives. The Rovina and Colnic porphyry deposits lie within an 8 km to 10 km diameter north-eastern volcanic outlier, Neogene-aged, Brad-Barza volcanic field. The Brad-Barza volcanic field is well known for hosting high-grade gold veins with historical gold production dating back to the Roman period (ca. 2,000 years ago). The Ciresata porphyry, 4.5 km south of Colnic, lies within the eastern part of the Brad-Barza volcanic field.

The main mineralised targets on the Rovina property are the Rovina Cu-Au porphyry, Colnic Au-Cu porphyry, and the Ciresata Au-Cu porphyry. Porphyry deposits are generally large, low- to medium-grade deposits in which primary (hypogene) sulphide minerals are dominantly structurally controlled, and which are spatially and genetically related to felsic to intermediate porphyritic intrusions.

The mineralised porphyries at Rovina, Colnic, and Ciresata display moderate to intense potassic hydrothermal altered cores, and strong quartz stockwork veining. The Au-Cu mineralisation manifests as stockworks and disseminations centred on porphyritic, subvolcanic-intrusive complexes of hornblende-plagioclase diorites. These porphyries classify as gold-rich, especially Ciresata and Colnic, and contain many of the features common in gold-rich porphyries (i.e. dioritic, calc-alkaline stock associated and abundant magnetite alteration). Oxidation is restricted to the uppermost few metres of the prospect and no significant oxide cap or supergene enriched horizons have been encountered to date.

1.5.2 Mineral Resource Statement

In 2007, AMEC completed the maiden mineral resource estimate for the Colnic and Rovina deposits. In 2009, after further resource definition drilling, PEG Mining Consultants Inc. (PEG) updated the mineral resource estimate for the Colnic and Rovina deposits and completed a maiden mineral resource estimate for the Ciresata deposit. After completion of an intensive infill drilling programme in preparations for a PEA, ESM commissioned AGP in 2012 to complete an updated mineral resource estimate on the three deposits.

In February 2019, AGP completed an updated PEA study on the Rovina, Colnic and Ciresata deposits. There were 16 additional holes drilled on the RVP since the 2012 mineral resource estimate, ten holes at Ciresata deposit and three each at Rovina and Colnic deposits. These drillholes were either for metallurgical samples, twin confirmation drilling as part of the ESM-Barrick exploration collaboration or brownfield exploration. AGP assessed the possible impact of the additional drilling and concluded that it is unlikely to have a significant impact on the mineral resources. The 2019 PEA, therefore, used the same geological and mineral resource block models of 2012 to report an updated mineral resource estimate reflecting current metal prices and operation parameters at that time (2019).

In March 2020, ESM commissioned SENET to complete a DFS on the open-pit Rovina and Colnic deposits. As part of this study, CCIC MinRes completed a detailed technical audit of the resource models, including an assessment on the possible impact of the ESM-Barrick exploration collaboration drilling on the mineral resource estimates. CCIC MinRes also recommended that ESM not update the 2012 geological and mineral resource block models until more holes are added to the resource database. The outcome of the technical audit confirmed the robustness of the AGP mineral resource models for the Rovina and Colnic deposits.

The March 2021 mineral resource estimate for the Rovina and Colnic deposits is, therefore, updated to reflect current metal prices and updated operating parameters derived during the DFS and to make it current and in conformance with the 2014 Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Mineral Resource and Mineral Reserve definitions referred to in the NI 43-101, Standards of Disclosure for Mineral Projects. Mr Sivanesan Subramani, BSc Hons (Geology), Pri.Sci.Nat. (400184/06), is the QP for this mineral resource estimate. The mineral resources are constrained to a Lerchs-Grossmann pit shell using different metal equivalent cut-off grades for the Rovina and Colnic deposits. The geological model and mineral resource block models remain unchanged in this current estimate. The mineral resource estimate for Ciresata remains unchanged from February 2019.

Table 1.3 summarises the mineral resource estimates for the Rovina and Colnic deposits, stated above a 0.25 % Cu equivalent grade cut-off for the Rovina deposit, and above a 0.35 g/t Au equivalent grade cut-off for the Colnic deposit. The total Measured mineral resources for the Rovina and Colnic deposits amount to 62.2 Mt grading at 0.49 g/t Au and 0.21 % Cu, containing 0.99 Moz Au and 287 Mlb Cu; with the Au equivalent grading of 0.79 g/t. The total Indicated mineral resources for the Rovina and Colnic deposits amount to an additional 175.6 Mt grading at 0.39 g/t Au and 0.15 % Cu, containing 2.19 Moz Au and 589 Mlb Cu, with the Au equivalent grading of 0.60 g/t.

Table 1.3: 2021 Mineral Resource Estimate – Rovina and Colnic Deposits

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Colnic	Measured	29.1	0.65	0.12	0.61	74	0.81	0.76
	Indicated	97.5	0.49	0.10	1.53	210	0.62	1.96
	Inferred	1.6	0.41	0.09	0.02	3	0.49	0.03
Rovina	Measured	33.1	0.36	0.29	0.38	212	0.77	0.82
	Indicated	78.1	0.26	0.22	0.66	379	0.57	1.44
	Inferred	16.0	0.18	0.19	0.09	66	0.44	0.23
Total	Measured	62.2	0.49	0.21	0.99	287	0.79	1.58
	Indicated	175.6	0.39	0.15	2.19	589	0.60	3.40
	Inferred	17.6	0.20	0.18	0.11	69	0.45	0.26
Grand Total	Measured and Indicated	237.7	0.42	0.17	3.18	875	0.65	4.97

NOTES:
1. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are contained within conceptual pit shells that are generated using the same economic and technical parameters used for Mineral Reserves but at a gold price of US$1,700/oz and a copper price of US$3.50/lb.
3. The Colnic and Rovina deposits are amenable to open-pit mining and Mineral Resources are pit constrained and tabulated at a base case cut-off grade of 0.35 g/t AuEq for Colnic and 0.25 % CuEq for Rovina.
4. Minor summation differences may occur as a result of rounding.

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,700/oz and a copper price of US$3.50/lb with metallurgical recoveries not taken into account.

The Ciresata underground mineral resource estimate remains unchanged from the 20 February 2019 estimate by AGP. Table 1.4 summarises the mineral resource estimate for Ciresata, stated at above a 0.65 g/t Au equivalent grade cut-off. The Measured mineral resources amount to 28.5 Mt grading at 0.88 g/t Au and 0.16 % Cu, containing 0.81 Moz Au and 102 Mlb Cu, with the Au equivalent grading of 1.13 g/t. The Indicated mineral resources amount to an additional 125.9 Mt grading at 0.74 g/t Au and 0.15 % Cu, containing 3.01 Moz Au and 413 Mlb Cu, with the Au equivalent grading of 0.97 g/t.

Table 1.4: 2019 Mineral Resource Estimate – Ciresata Deposit

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Ciresata	Measured	28.5	0.88	0.16	0.81	102	1.13	1.03
	Indicated	125.9	0.74	0.15	3.01	413	0.97	3.92
	Inferred	8.6	0.70	0.14	0.19	26	0.94	0.25
Total	**Measured & Indicated**	**154.4**	**0.77**	**0.15**	**3.82**	**515**	**1.00**	**4.95**

NOTES:

1. The Ciresata deposit is amenable to bulk underground mining and resources are tabulated at a base case 0.65 g/t AuEq.
2. No Mineral Reserves have been defined at the Ciresata deposit. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Minor summation differences may occur as a result of rounding.

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,500/oz and a copper price of US$3.50/lb.

Source: From Table 14-20, AGP PEA NI 43-101 2019 Report (available on SEDAR)

It must be noted that the quantity and grade of Inferred resource reported above are conceptual in nature and are estimated based on limited geological evidence and sampling. Geological evidence is sufficient to imply, but not verify, geological and grade or quality continuity. For these reasons, an Inferred mineral resource has a lower level of confidence than an Indicated mineral resource, and it is reasonably expected that the majority of Inferred mineral resources could be upgraded to an Indicated mineral resource with continued exploration. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Rounding of tonnes as required by reporting guidelines may result in apparent differences between tonnes, grade, and contained metal content.

Changes in the current metal prices and updated operating parameters from the 2012 mineral resource estimate resulted in a shrinkage of the Lerchs-Grossmann mineral resource constraining shell and, therefore, a reduction in the overall mineral resource estimates for the Rovina and Colnic deposits. The total Measured mineral resource tonnage increased by 1.4 %, with the Au and Cu grades remaining the same. The total Indicated mineral resource tonnage decreased by 2.8 %, from 180.7 Mt to 175.6 Mt, with the Au and Cu grades remaining the same. The total Inferred mineral resource tonnage decreased by 10.4 %, from 19.6 Mt to 17.6 Mt, with the Au and Cu grades remaining the same.

1.5.3 Status of Exploration

Most of the exploration on the property has been performed by three companies: Minexfor between 1974 and 1998, and again in 2001, Rio Tinto from 1999 to 2000, and ESM since 2004. In September 2011, Barrick Gold and ESM formed an exploration collaboration group to evaluate further exploration targets on the Rovina licence.

Exploration activities on the Rovina Licence property were halted in 2012, as required by the process of conversion to an Exploitation Licence. Since granting of the Exploitation Licence in November 2018, ESM has focused all efforts on the Feasibility Study and towards mine development.

1.5.4 Development and Operations

The RVP is an advanced exploration project and is currently not in operation.

1.5.5 Conclusions

CCIC MinRes concludes that, effective 01 March 2021 and utilising approximately 120,256 m of diamond drillhole data drilled by ESM from 2006 to 2012, the mineral resource of the RVP (inclusive of all three deposits) amounts to 90.7 Mt of Measured Resources grading at 0.62 g/t Au and 0.19 % Cu containing 1.80 Moz Au and 389 Mlb Cu. Indicated resources amounted to an additional 301.5 Mt grading 0.54 g/t Au and 0.15 % Cu containing 5.20 Moz Au and 1,002 Mlb Cu. The total Measured and Indicated resources amounted to 392.1 Mt grading at 0.55 g/t Au and 0.16 g/t Cu containing 6.99 Moz Au and 1,390 Mlb Cu. Inferred resources added an additional 26.2 Mt grading 0.36 g/t Au and 0.16 % Cu containing 0.30 Moz Au and 95 Mlb Cu.

CCIC MinRes concluded the following:

- While other companies conducted exploration drilling on the property, only the ESM sampling data, collected since 2006, was used in the mineral resource estimate. This ensured that modern assaying techniques and proper quality assurance/quality control (QA/QC) protocols were in place for the entire drill programme and eliminated any need to rely on historical data.
- CCIC MinRes has reviewed the methods and procedures used to collect and compile the geological and assaying information for the RVP and found that they met accepted industry standards for an advanced-stage project and are sufficient for the porphyry style of mineralisation.
- A review of the QA/QC results for all drilling since 2006 concluded that, despite minor insufficiencies of the QA/QC programme, the Au and Cu assays from drillhole sampling are sufficiently precise and accurate for mineral resource estimation purposes.
- During a site visit and data verification conducted by Mr Subramani, between 9 and 12 November 2020, he concluded that the data collection and verification procedures implemented by ESM are to industry standards, and that the geological database is of sufficient reliability for use in mineral resource estimation.
- CCIC MinRes assessed the possible impact of the six additional drillholes (from the ESM-Barrick exploration collaboration) on the mineral resource estimates for the Rovina and Colnic deposits and concluded that there is no risk of overestimation and, therefore, recommended that ESM not update the 2012 geological and mineral resource block models until more holes are added to the resource database.
- CCIC MinRes completed a detailed technical audit of the geological and mineral resource block models for Rovina and Colnic and confirmed the robustness of the AGP mineral resource models for these deposits.

1.5.6 Recommendations

In preparation for mine development, ESM should consider the following recommendations for operational readiness:

- Conduct pre-production drilling within the mine planning footprint for the first 24 to 36 months of ore mining. This will ensure that all mineral resources mined during this period are converted to Proven mineral reserves, thereby minimising geological risks during this crucial payback period.
- Implement database management software to manage and monitor sampling QA/QC programmes. This system will flag batches that are beyond the threshold limits and allow for immediate remedial action. Also, where there is evidence of positive or negative drifts, the laboratory can be notified to calibrate their equipment more regularly.
- Research the implementation of Leapfrog Implicit modelling of geological and geostatistical domains during grade-control modelling. This will allow for quick and efficient updating of the geological models when turnaround times are crucial.
- Undertake a geostatistical study to determine the optimum drill spacing for grade-control modelling. Optimum drill spacing will assist with time and costs during mine production.
- Research the correlation between the results from a mobile X-ray fluorescence (XRF) scanner and those from a laboratory. This will be useful for grade-control sampling and modelling as follows:

  - If there is a reliable proxy between the Cu and Au grades in the deposit, then the XRF readings for Cu can be used as a proxy for anticipating the Au grades.
  - If a reliable proxy can be established, this will facilitate quick and cost-efficient turnaround times for grade-control assays.

1.6 MINERAL RESERVE AND MINE PLANNING

1.6.1 Mineral Reserve Estimates

The process to develop the Mineral Reserve estimate for the RVP was as follows:

- The open-pit optimisation has been undertaken on the Measured and Indicated Resources only.
- The geological losses in the block model allow for a 2.5 % loss on Measured resources and a 3.5 % on Indicated resources.
- The grades and tonnes of the mineral resource model have been modified by mining/geological recovery and mining dilution based on orebody geometry and mining methodology. The mining model contains undiluted ore tonnes and ore grade. Owing to the massive nature of both the Colnic and Rovina orebodies, fixed dilution and recovery percentages of 2 % and 97.5 %, respectively, were applied in the Whittle optimisations.

- The Whittle suite of optimisation software was used to perform the pit optimisations. Whittle is an accepted industry optimisation tool. A range of operating costs and production parameters were applied. The source of the parameters is summarised below, along with the source of the information:
  - A base gold price of US$1,500/oz with a government royalty of 6.0 % of the revenue. This resulted in a net gold price of ~US$1,382.50/oz at the Colnic pit and ~$1,277.88/oz at the Rovina pit. The difference is due to the net smelter return (NSR) calculations, which differ for each area.
  - A base copper price of US$3.00/lb with a government royalty of 5.0 % of the revenue. This resulted in a net copper price of ~US$5,089.83/t for both pits.
  - Pit slope inter-ramp angles ranging from 44.0° to 49.9°, depending on the pit slope geometry. The resulting overall pit slope angles account for access ramps where applicable.
  - Gold recovery ranging from 77.6 % to 85.5 %, depending on the mining area and the ore type being processed.
  - Processing throughput of 7.2 Mt/a.
  - The Owner’s mining fleet capital and operating costs are based on budget pricing submissions from various European original equipment manufacturers (OEMs), and all costs have been converted to United States dollars.
  - An average annual processing cost per tonne of ore, inclusive of general and administration costs and overland haulage.

A sensitivity assessment was done on gold prices of US$1,400/oz and US$1,700/oz. A scenario assessment with a gold price of US$2,500/oz was also done to determine the surface infrastructure boundaries required to ensure that no potential future resource is sterilised. This indicated that the optimal shell inventory (i.e. the size and shape of the optimal shell and, therefore, the ore and waste generated) was robust for all mining areas.

Optimal shells were selected for each deposit, and these were then used as the basis for the ultimate pit designs. The shell selection criteria were conservative and were based on a gold price of US$1,500/oz and a copper price of US$3.00/lb.

Various cut-off grades were applied, based on projected incremental revenue per tonne and high-grade and low-grade stockpiles were generated based on a two-stage pit design with pushbacks to ensure that optimal revenue is delivered in the early life of mine (LOM).

Table 1.5 summarises the Mineral Reserve Statement based on the work detailed above, undertaken as part of the RVP.

Table 1.5: Rovina Valley Project Mineral Reserve Statement Summary

Deposit	Classification	Tonnage (Mt)	Au Grade (g/t)	Cu Grade (%)	Au (koz)	Cu (t)
Colnic	Proven	24.27	0.64	0.11	500.5	26,860.9
	Probable	49.49	0.52	0.08	828.7	41,004.7
Rovina	Proven	24.01	0.32	0.28	247.8	67,469.3
	Probable	35.62	0.22	0.20	249.5	72,896.1
Total	Proven	48.28	0.48	0.20	748.3	94,330.2
	Probable	85.11	0.39	0.13	1,078.2	113,900.8
Grand Total	**Proven + Probable.**	**133.40**	**0.43**	**0.16**	**1,826.5**	**208,231.0**
NOTE: All tonnes quoted are dry tonnes. Differences in the addition of deposit tonnes to the total displayed is due to rounding.						

The Mineral Reserve estimate has been classified and reported in accordance with the Canadian National Instrument 43-101, "Standards of Disclosure for Mineral Projects" of June 2011 (NI 43-101), and the classifications adopted by the CIM Council in November 2010. Furthermore, the Mineral Reserve classifications are also consistent with the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" of 2012 (JORC Code) as prepared by the Australasian Joint Ore Reserves Committee comprising representatives from the Australasian Institute of Mining and Metallurgy, the Australian Institute of Geoscientists, and the Minerals Council of Australia, with the minor exception that the JORC Code refers to Ore Reserves while NI 43-101 refers to Mineral Reserves.

The RVP Mineral Reserve estimate is not at this stage materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issue. Furthermore, the estimate of Project Reserves is not materially affected by any known mining, metallurgical, infrastructure, or other relevant factor.

DRA is confident that sufficient geological work has been undertaken, and sufficient geological understanding gained, to enable the construction of an orebody model suitable for the derivation of Mineral Resource and Mineral Reserve estimates. DRA considers that both the modelling and the grade interpolation have been carried out in an unbiased manner and that the resulting grade and tonnage estimates should be reliable within the context of the classification applied. In addition, DRA is not aware of any metallurgical, infrastructural, environmental, legal, title, taxation, socio-economic, or marketing issues that would impact on the Mineral Resource or Mineral Reserve statements as presented.

1.6.2 Mining Methods, Mining Equipment and Infrastructure

1.6.2.1 Introduction

A LOM schedule has been developed to supply one processing plant for the full LOM. The processing plant has a planned throughput of 7.2 Mt/a (Colnic pit and then Rovina pit) and a LOM of 16 years, excluding the Colnic low-grade stockpile. The LOM schedule considers the in-pit and stockpile blending requirements during the life of each pit, as well as the changeover from Colnic to Rovina ore supply to the processing plant.

The Owner's mining fleet will be responsible for all mining-related earthmoving activities.

All deposits will be mined utilising conventional truck and shovel methods to supply ore to the run of mine (ROM) tip and waste to the respective pit's waste crushing and conveying station, which will transport the waste to the Colnic stockpile area and backfill the Colnic pit.

Most of the ore and waste materials will be drilled and blasted as there is a nominal amount of free dig/oxidised materials.

Free-dig and blasted waste will be loaded with 200 t class hydraulic backhoe excavators, hauled with 90 t haul trucks, and stockpiled at designated waste stockpile locations, which will be systematically dozed and levelled to allow the stockpiles to be raised in accordance with the design parameters.

Free-dig and blasted ore will be loaded with 200 t class hydraulic backhoe excavators and hauled with 90 t haul trucks to the plant feed ROM pad. There the ore will either be direct tipped into the crushing facility or placed on the ROM pad stockpile areas, depending on the grade control strategy being applied.

The project is to be mined utilising proven drilling, blasting and earthmoving equipment operated by an experienced Owner's team and well-trained workforce.

Owing to the significantly higher elevation and limited capacity of the Colnic waste storage facility, a waste-only crushing and conveying system will be installed for waste transportation only during the operational life of the Colnic pit. This crushing and conveying system will be reclaimed at the end of the Colnic pit's life and reused for both waste and ore batched transportation to the Colnic backfill site and the Colnic ROM pad position from Year 9 until the end of the Rovina pit life.

The Colnic pit will commence production first as it has the best incremental value per tonne and as such will assist in delivering higher mill feed grades early in the project life.

Approximately six to seven months of pre-stripping will be required to expose sufficient ore to maintain a constant ore feed of 7.2 Mt/a post commissioning and the planned process plant feed build-up.

The mining of the two deposits runs for a period of approximately 16 years based on the current production schedule and design parameters.

The peak production requirements of the total mining fleets have been capped at an estimated 27.4 Mt/a (total material movement).

1.6.2.2 Project Design and Operation

1.6.2.2.1 Slope Design

The preliminary pit slope design parameters below were provided by KCB. The slopes were provided based on the weathering codes within the block model (i.e. oxide/transition/fresh). The preliminary design parameters are detailed in Table 1.6.

Table 1.6: Colnic and Rovina Slope Design Parameters

Area	Bench Face Angle	Bench Height	Berm Width	Maximum Inter-Ramp Height	Maximum Height Width Geotechnical Berm if No Ramp	Geotechnical Berm Width	Ramp Width	Inter-Ramp Angle
	degrees	m	m	m	m	m	m	degrees
Controlled Blasting Area	65	24	9	120	120	25	28	49.9
Overburden	33	3	9					

It is important to note that these preliminary slope angle design parameters may be subject to changes based on the 2020 geotechnical and geohydrological studies currently being undertaken by KCB. The final summary results for most of the Colnic pit BFA increased from 65° to 70°.

1.6.2.2.2 Processing Recoveries and Costs

Processing costs and recoveries for each metallurgical domain were received from DRA and modelled in 3D using the ESM geological model for each of the domains of the open pit deposits.

1.6.2.2.3 Financial Parameters

The gold price and discount rate used in the optimisations are summarised in Table 1.7.

Table 1.7: Optimisation Financial Parameters

Parameter	Unit	Colnic	Rovina
Discount Rate	%	10	10
Base Price:			
Gold	US$/oz	1,500	1,500
Copper	US$/lb	3.00	3.00
Government Royalty:			
Gold	%	6.0	6.0
Copper	%	5.0	5.0
Processing Cost	US$/t ROM	9.77	7.53
Net Price:			
Gold	US$/g	44.448	41.085
Copper	US$/t	5,089.83	5,089.83

1.6.2.3 Bench Height Selection

1.6.2.3.1 Motivation for Bench Height Analysis

The geological resource block model was generated on a parent block cell size of 10 m × 10 m × 12 m (X, Y and Z, respectively) for both the Rovina and Colnic pits.

1.6.2.3.2 Ore Dilution and Ore Loss

A 2 % dilution and 2.5 % loss were assumed because the ore domains are continuous and will be clearly delineated and marked. Sampling of blast holes would be the basis for grade control in this analysis. The accuracy of the resulting ore/waste boundary is limited by the resolution of the grade control, which is a function of the density of the drilling pattern. The lower the flitch height, the smaller the pattern, the smaller the distance between “ore holes” and “waste holes” and hence the smaller the potential for ore loss and/or ore dilution. These dilution and loss percentages are accepted as being in line with smaller flitch heights such as the 6 m flitches associated with this mining operation.

1.6.2.3.3 Colnic Shell Selection Run 29

By selecting Shell 26 of Run 29, the total waste mined was limited to 108.6 Mt and ore to 73.2 Mt at an average gold grade of 0.57 g/t and an average copper grade of 0.09 %. This is necessary due to the limited capacity of the co-disposal storage facility, which at this stage was estimated at approximately 180 Mt. Whittle Optimisation results for this are shown in Figure 1.1.

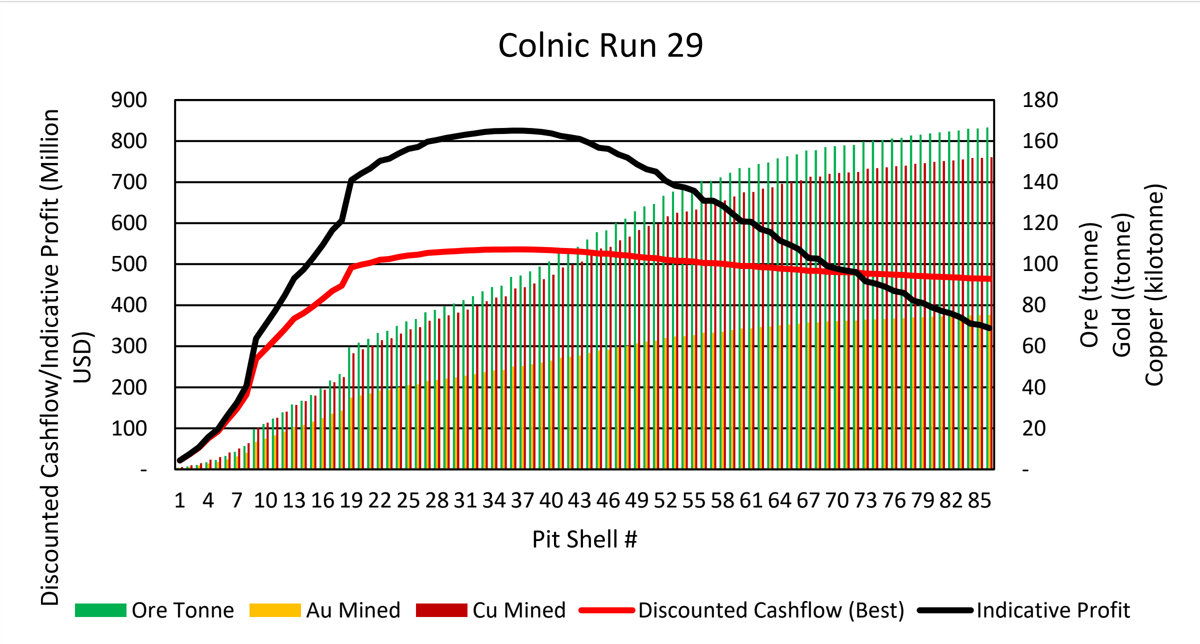


Figure 1.1: Colnic Whittle Analysis

1.6.2.3.4 Rovina Shell Selection Run 8

By selecting Shell 32 of Run 8, the total waste mined was limited to 125.9 Mt and ore to 64.3 Mt at an average gold grade of 0.25 g/t and an average copper grade of 0.22 %. This is necessary due to the limited capacity of the Colnic pit backfill co-disposal storage facility, which at this stage was estimated at approximately 190 Mt. Whittle Optimisation results for this are shown in Figure 1.2

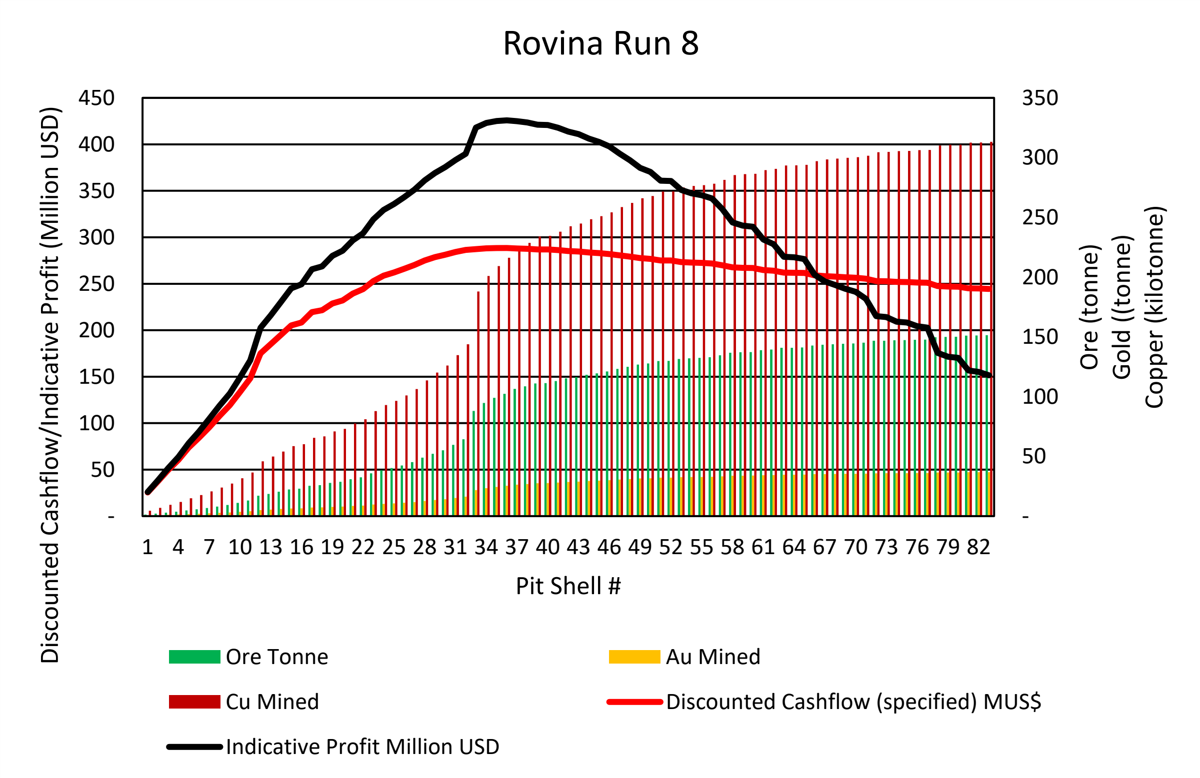


Figure 1.2: Rovina Whittle Analysis

1.6.2.3.5 Shell Selection Summary

The inventories of the selected shells are summarised in Table 1.8.

Table 1.8: Summary of Selected Shells

Area	Pit Shell No.	Discounted Cash Flow (Specified)	Indicative Profit	Waste	Ore	Au Mined	Grade
		US$ million	US$ million	Mt	Mt	t	g/t
Colnic	26	523.03	785.64	108.63	73.24	41.48	0.57
Rovina	32	286.36	389.79	125.92	64.29	6.22	0.25

1.6.2.4 Project Design

All the pit designs were developed using the Deswik.CAD suite of software packages. They were based on the optimal shells selected and utilised the latest ESM resource block models that have been reviewed and signed off by CCIC MinRes.

The models were coded with appropriate batter angles, berm widths, bench, and stack heights for different rock/material types for each deposit. These slope design parameters were based on the geotechnical design criteria provided by KCB as used in the open-pit optimisation (see Table 1.6).

The criteria for pit and waste stockpile ramp designs were based on the width and turning circle of 90 t dump trucks, as this size truck is likely to be used as the OEM trucking fleet. Ramp gradients are 10 %. Wherever possible, the ramp exits were located at the closest possible distance to the waste storage facilities to minimise ex-pit haulage.

A minimum mining width of 20 m is maintained.

1.6.2.4.1 Haul Roads

Where possible, existing haul roads will be used for ore and waste. However, a number of temporary haul roads will be required during the LOM of both pits. All haul roads have been laid out on the overall site plans.

1.6.2.4.2 Pit Designs

The RVP area consists of two mining areas containing one pit each. These are the Colnic pit in Figure 1.3 and the Rovina pit in Figure 1.4.

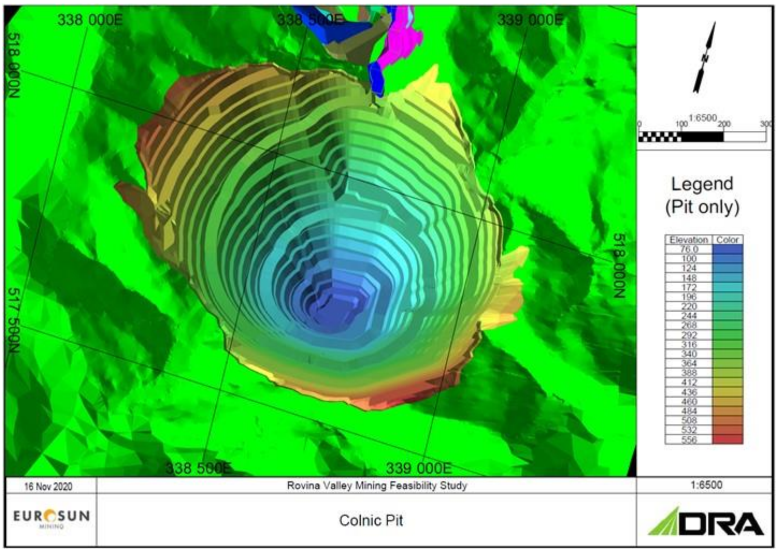


Figure 1.3: Colnic Pit

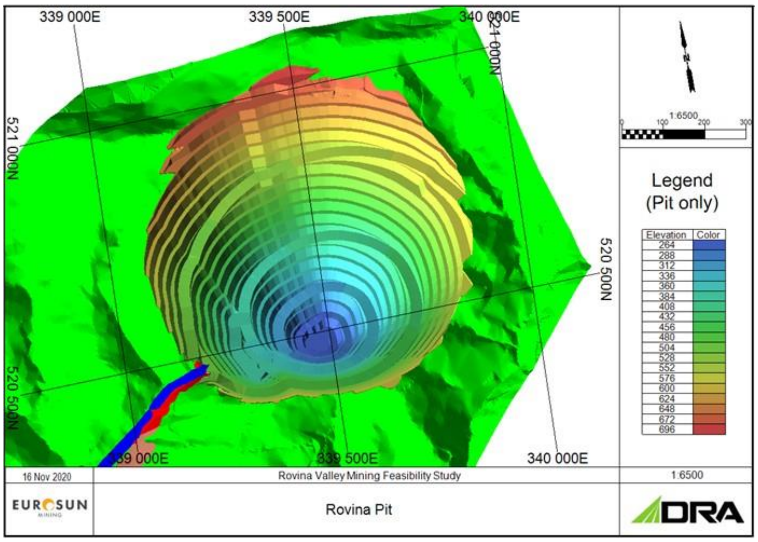


Figure 1.4: Rovina Pit

1.6.2.5 Scheduling Results

The scheduling results are summarised in Figure 1.5, which depicts the RVP combined LOM schedule. The results show that the schedule is a practical solution that targets value and meets all mining and processing goals monthly. The key features of the final base case schedule include the following:

- The requirement of 10.71 Mt of pre-strip material movement over six months of pre-stripping in the Colnic mining area.
- The requirement for the Rovina pit to be pre-stripped while the Colnic high-grade stockpiles are being depleted. The pre-strip requirement for the Rovina pit is 14.05 Mt.
- The requirement that a maximum annual materials movement of 27.3 Mt be maintained throughout each production year. The two front-end loaders (FELs) can be utilised to assist in achieving this production performance when planned maintenance and breakdowns would impede achieving this total productivity.

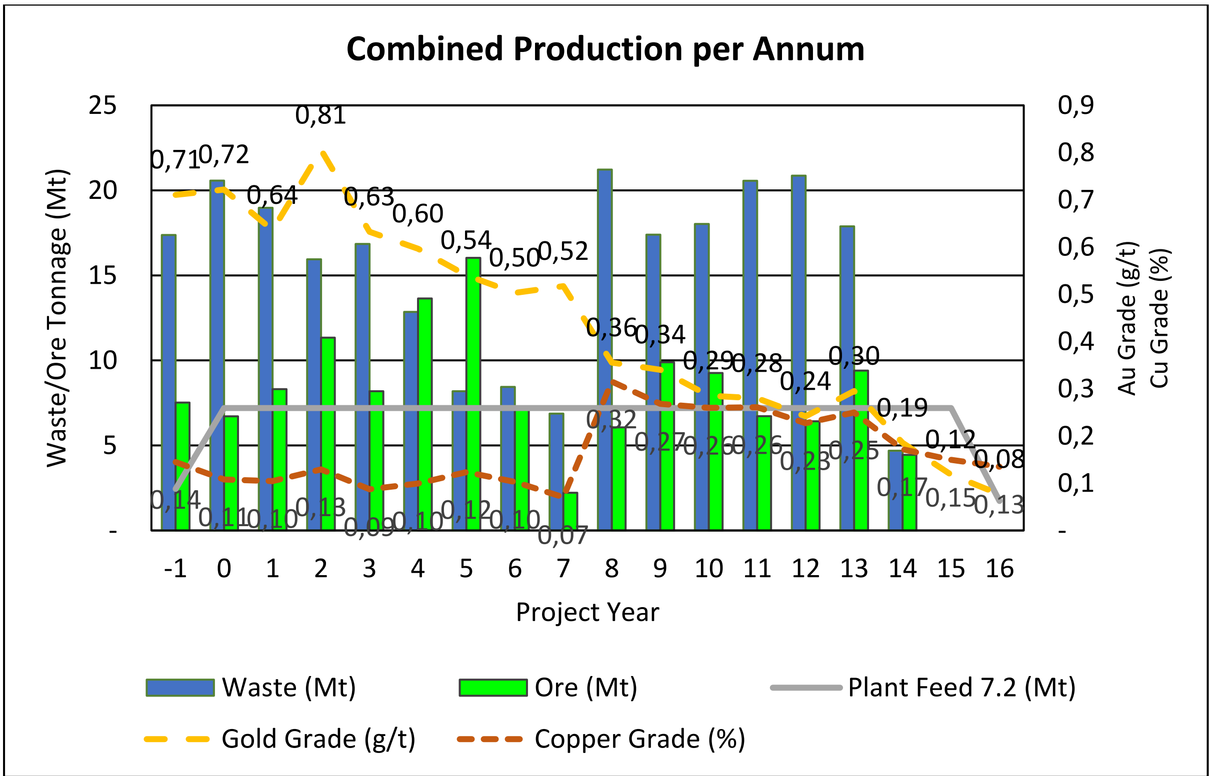


Figure 1.5: Total Project Material Movements for LOM

1.6.2.6 Equipment Fleets

The scheduling in XPAC is driven by the excavating capability during each period (i.e. the product of the number of excavators and their productivity).

For scheduling purposes, it was assumed that three 200 t excavators with 12.5 m^3 buckets would be deployed on both waste and ore production. These excavators will be loading rigid dump trucks with a payload capacity of 90 t. The first principal productivity calculations for determining period-by-period material movement were based on 200 t excavators and 90 t rigid dump trucks, which will be allocated to each mining area, taking cognisance of the hauling routes that each fleet will use. The envisioned excavator fleet deployment is shown in Figure 1.6.

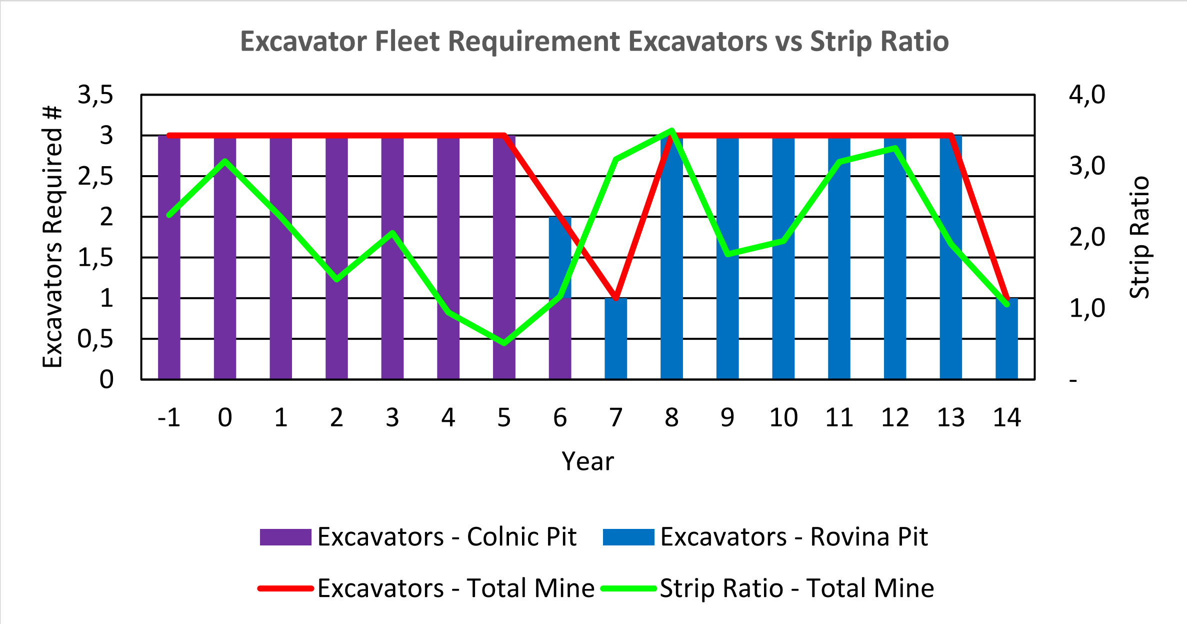


Figure 1.6: LOM Excavator Fleet

1.6.2.7 Excavator Productivity

Excavator productivity varies with the two loader types as indicated in Table 1.9.

Table 1.9: Loaders Productivity

Loader Type	Truck Type	Annual Capacity	Unit
Hydraulic Excavator 230 t – 12 m³	DT 90 t	9,874,851	t/a
FEL 10 m³	DT 90 t	4,240,989	t/a

During the first four months of operation, the loader productivity is reduced to reflect commissioning, shift ramp-up, and the ramping up of each operator's skills level.

1.6.2.8 Waste Rock Stockpiles

The placement of the waste rock stockpiles is constrained and should be optimised to limit hauling and conveying distances over the LOM. These optimisation results are shown in Figure 1.7 and Figure 1.8 for waste rock and ore, showing the haul distances for ore and waste for Colnic to the Colnic waste storage facility and Rovina to the north of the Colnic waste storage and then backfilling of the Colnic Pit. The Rovina waste and ore will be batch-conveyed from the Rovina crushing station to the ore ROM pad and the Colnic backfill stockpile. The Rovina low-grade will use the depleted Colnic high-grade stockpile area once it is empty. Owing to its location, road hauling of the Rovina low-grade is planned but this may be optimised to conveying if not cost prohibitive.

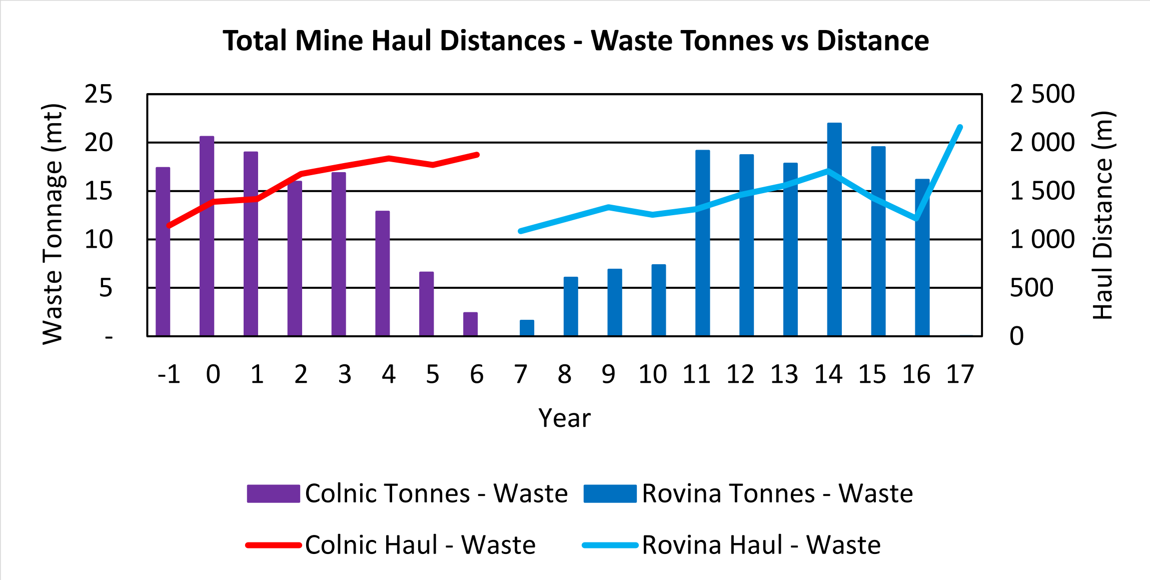


Figure 1.7: Weight Averaged One-Way Haul Distances – Total Mine Waste

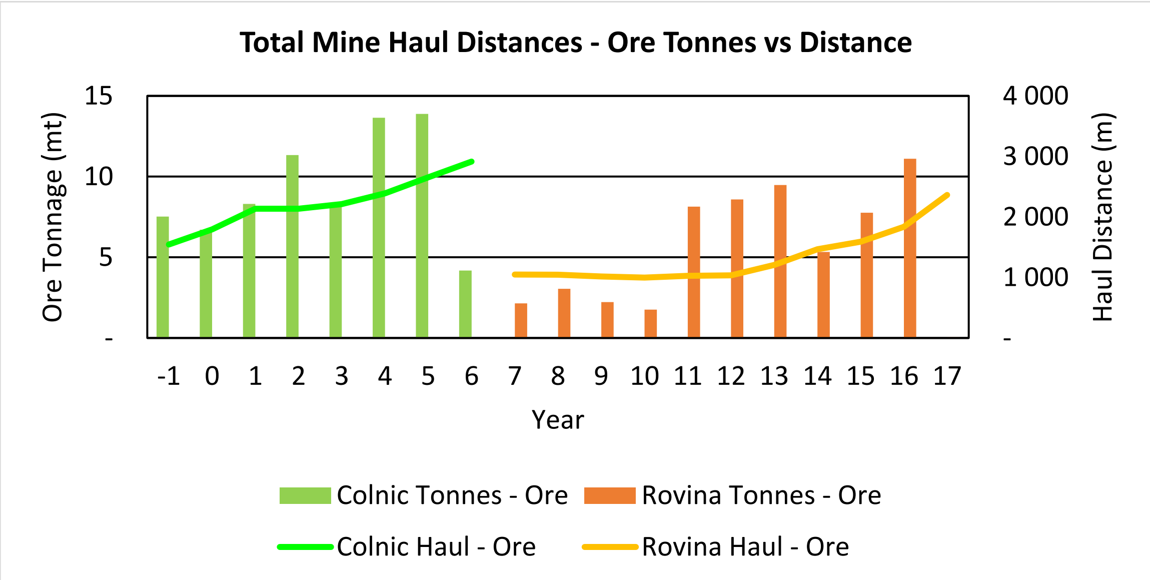


Figure 1.8: Weight Averaged One-Way Haul Distances – Total Mine Ore

1.6.2.9 Waste Management Facility – Conveyors and Infrastructure.

The Colnic waste will be hauled to the waste crushing facility situated along the haul road to the ore crushing pad. Waste will be tipped into the receiving bin and fed from the bin via an apron feeder to the semi-mobile crushing station.

The waste passes over a vibrating grizzly feeder (VGF) cutting at a 450 mm particle size, with the oversize reporting to a C200 jaw crusher and the undersize reporting directly to the waste extraction conveyor. The jaw crusher product (−450 mm) also reports to the extraction conveyor.

The crushing system has an average capacity of 3,891 t/h and can handle up to 5,000 t/h peaks.

Table 1.10 highlights the key parameters included in the crusher selection process, and Figure 1.9 shows the general arrangement (GA) and flow of waste material through the tipping and crushing process.

Table 1.10: C200 Waste Crusher System Key Parameters

C200 Crusher Parameters Description	Unit	Value
Maximum product size	mm	450
Design percentage passing ± 5 %	%	77
VGF undersize	t/h	2,743
Crusher feed	t/h	817
Closed crusher size	mm	300
Operational efficiency	%	80%
C200 crushing throughput	t/h	1,148
C200 crusher production	t/h	3,891

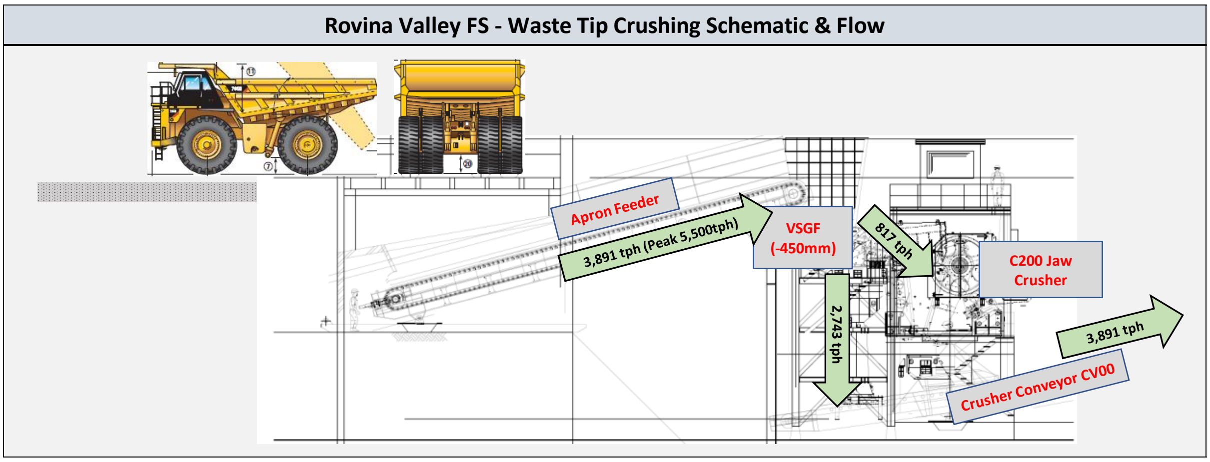


Figure 1.9: Waste Crusher GA and Flow

The crushing system waste stream (see Figure 1.10) is transferred via the extraction conveyor (CV-00) to the waste facility conveyor system. The waste facility conveyor system has a total of five conveyors and a telescopic spreader/stacker, scheduled in the capital costing according to the waste facility building process.

The first three conveyors in the system (CV-01, CV-02 and CV-03) have an average design capacity of 3,891 t/h. The last conveyors in the system (CV-04 and CV-05), as well as the spreader/stacker, have an average design capacity of 4,800 t/h to cater for plant tailings added at the transfer point between CV-02 and CV-03.

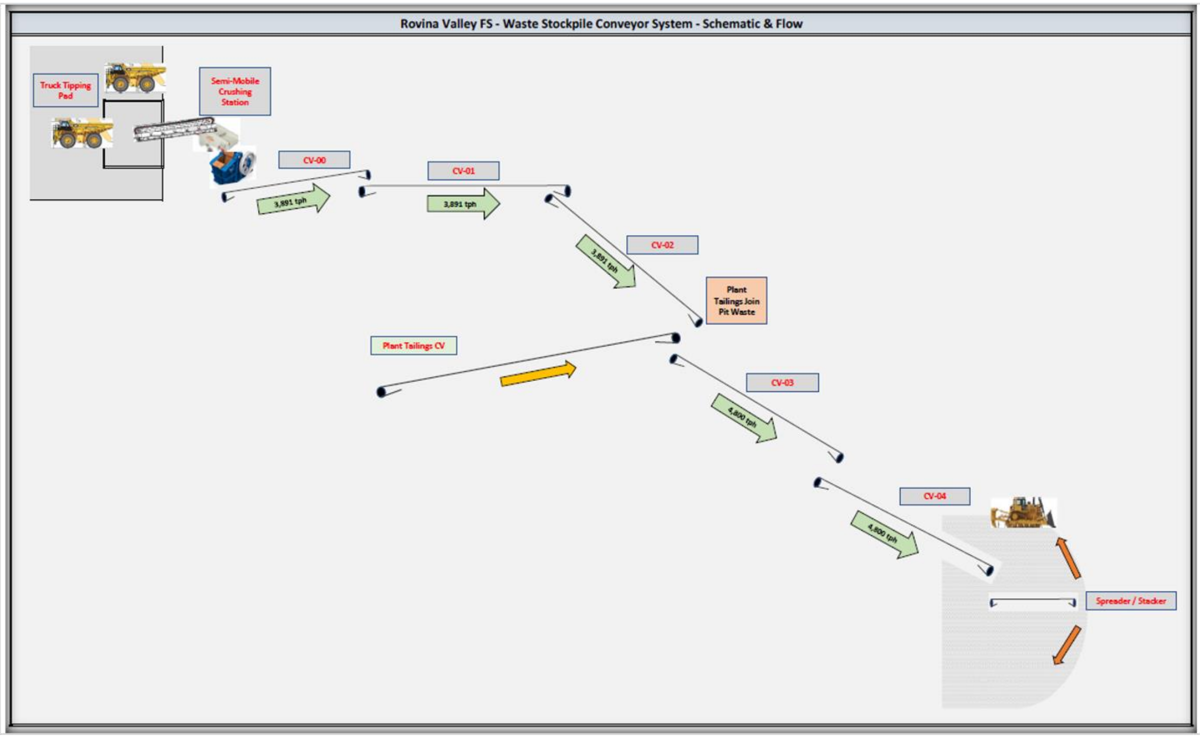


Figure 1.10: Waste Conveyor System Schematic

A comprehensive request for budget pricing (RFBP) process was conducted to obtain pricing for the capital costing and scheduling aligned with the waste facility planning and design.

Some key items addressed during design:

- Skid-mounted stringer sections that are 2.4 m long to ensure ease of movement and extension of conveyors
- Standardisation of conveyor drive units to minimise spares keeping
- Standardisation of the class of belting to minimise spare quantity
- Track-mounted telescopic spreader stacker to facilitate final waste storage facility construction with dozer assistance

The waste facility conveyor and semi-mobile crusher system will be utilised for the life of the Colnic pit mining. Once mining of the Rovina pit commences, these crusher and conveyor systems will be reclaimed and installed (reused) from the Rovina pit to be able to backfill the Colnic pit with Rovina waste and rougher tailing. This move will be done once the waste handling facility has reached the design limit. The timing of the move can, however, be optimised during the LOM planning process.

Once the crusher and conveyor systems have been moved to the Rovina pit position, the system will be utilised for both ore and waste in a batching regime. Ore will be conveyed to the existing plant ROM tip crushing system with a diversion chute arrangement. Waste will be conveyed via the same system to backfill the Colnic pit.

The system design in the Rovina pit configuration has been adapted to handle both waste and ore and will have a capacity of 4,800 t/h. This is possible as the conveyors in this configuration have lower inclinations and the system will also require less absorbed power.

1.7 METALLURGICAL TEST WORK AND PROCESS DESIGN

1.7.1 Metallurgical Test Work

Comprehensive metallurgical test work has been conducted over the years on the composite samples of the Rovina Valley orebodies to support the studies conducted in 2010, 2012 and the latest PEA conducted in 2019. Historical work included preliminary evaluation of grindability, mineralogical gold deportment, geometallurgical populations and mineralogy, and bench-scale flotation (batch and locked cycle). In 2019, a column flotation pilot campaign was conducted on three composite samples representing the Colnic K1 domain, Rovina domain and Colnic K2K3 domain. Since all historical test work conducted to support the process flowsheet was based on composite samples only, SENET, therefore, proposed a comprehensive metallurgical test work programme to establish the degree of comminution characteristics within the ore domains, solid-liquid separation, and transport moisture limits tests for filter cake conveying and stacking.

It should be noted that the flotation or recovery variability and concentrate solid-liquid separation test work could not be conducted due to a lack of samples; therefore, it was agreed that the column flotation pilot plant results would be used for the flotation plant design. The cleaner tails solid-liquid separation results will be applied for the concentrate equipment design.

Table 1.11 outlines a summary of the test work results.

Table 1.11: Summary of Test Work Results

Test Work	Outcome
Comminution	The comminution test work was performed in two phases: • Phase 1 comminution test work (Bond Ball Work index (BBWi), Bond Abrasion index (Ai), SAG Mill Comminution (SMC)) on variability samples • Phase 2 comminution test work (BBWi, BRWi, Ai SMC) on composite samples The results indicated that the Rovina orebody can be classified as moderately hard to very hard, and medium abrasive to abrasive. Therefore, the energy requirements for crushing and milling are expected to be high. Liner wear and media wear are also expected to be high. The A × b values indicate that the Rovina ore is a good candidate for Semi Autogenous Grinding (SAG) milling. From the test work results, Orway Mineral Consulting (OMC) selected the following values for the design of the comminution circuit. It should be noted that since no Crushability Work index (CWi) test work was conducted on the Rovina ore, the Colnic ore results have been applied in the design. Colnic ore: • BBWi average 18.1 kWh/t • BRWi average 18.4kWh/t • Ai average 0.323 • CWi average 15.9 kWh/t • A × b average 26

Test Work	Outcome
	Rovina ore: • BBWi average 15.9 kWh/t • BRWi average 14.3 kWh/t • Ai average 0.285 • CWi average N/A • A × b average 25
Head Assays	Multi-head assays show that the Colnic orebody has a high gold content (averaging 0.62 g/t) and low copper content (averaging 0.10 %). The Rovina orebody shows a low gold content (averaging 0.23 g/t) and high copper content (averaging 0.26 %)
Mineralogy	Sulphide mineralogy consists primarily of pyrite and chalcopyrite with minor amounts of pyrrhotite in both the Colnic and Rovina units. Of note, is the ratio of pyrite to chalcopyrite, which ranges from 8.1 in Rovina up to 18.6 in Colnic. Limited mineralogy indicates gold population as fine-grained particles occurring as native gold, attached, or locked in sulphides and also locked in silicates.

Eriez Column Flotation Pilot Plant Results	The results from the Eriez pilot plant are summarised below:
	• Colnic Ore:
	  • Gold recovery range: 77 % to 86%
	  • Gold concentrate grade range: 83 g/t to 111.4 g/t
	  • Copper recovery range: 92.4 % to 94.7 %
	  • Copper concentrate grade range: 21.13 % to 22.60 %
	• Rovina Ore:
	  • Gold recovery range: 70.6 % to 75.7 %
	  • Gold concentrate grade range: 10.7 g/t to 13 g/t
	  • Copper recovery range: 94 % to 95.5 %
	  • Copper concentrate grade range: 19.1 % - 22.65 %

IsaMill™ Fine Grind Test Work (Signature Plot Test Work)	Since the selected flotation circuit involved fine milling the rougher concentrate, it was imperative that IsaMill signature plot test work be conducted to determine the power requirements for fine milling the concentrates from P_{80} 75 µm to P_{80} 13.5 µm. However, there were no flotation concentrate samples available to conduct the fine milling test work. To give an indication of the power requirements for fine milling, IsaMill signature plot test work was conducted on ROM samples of the Colnic and Rovina Valley samples. The test work indicated power requirements of 76.1 kWh/t and 61.2 kWh/t for the Rovina ore and Colnic ore, respectively. The values were on the higher side; this was due to the fact that the test work was done on ROM samples not concentrate. The ROM material would have gangue, which is difficult to fine mill.

Thickening and Rheology	Thickening and rheology test work conducted on the column flotation pilot plant tails slurry showed that the ore settles easily, and the following was determined:
	• Scavenger Tails:
	  • The flocculant SNF AN905 SH gave better overall performance and was able to produce clear supernatant liquor at 25 g/t to 35 g/t.
	  • The optimum feed well density was 17.5 % w/w to 25 % w/w.
	  • The dynamic thickener achieved an underflow density of 57 % to 64 %.
	  • The solids flux rate was 0.578 $t/h/m^2$ to 0.698 $t/h/m^2$.
	  • The apparent viscosity range was 6.47 Pa·s (at 5 s^{-1} shear rate) to 0.17 Pa·s (at 1,000 s^{-1} shear rate).
	• Cleaner Tails:
	  • The flocculant SNF AN905 SH at a consumption range of at 40 g/t to 50 g/t gave better overall performance and was able to produce clear supernatant liquor.

<table>
<tr><th>Test Work</th><th>Outcome</th></tr>
<tr><td></td><td>The optimum feed well density was 20 % w/w to 25 % w/w.
The dynamic thickener achieved an underflow density of 57 % to 63.5 %.
The solids flux rate was 0.649 t/h/m^2 to 0.897 t/h/m^2.
The apparent viscosity range was 4.31 Pa·s (at 5 s^{-1} shear rate) to 0.12 Pa·s (at 1,000 s^{-1} shear rate).</td></tr>
<tr><td>Filtration and Transportable Moisture Limits</td><td>Filtration and transportable moisture limit test work was conducted on thickened slurry from the column flotation pilot plant tails, and the following was determined:
<u>Scavenger Tails:</u>
The feed density was 50 % w/w.
The filtration rate range was 156.6 kg/h/m^2 to 183.49 kg/h/m^2.
The filter cake moisture range was 13.7 % to 14.5 %.
The filter cake thickness range was 54.2 mm to 57 mm.
The transportable moisture limit values range from 15 % to 16.5 %.
<u>Cleaner Tails:</u>
The feed density range was 57 % w/w to 60 % w/w.
The filtration rate range was 157.26 kg/h/m2 to 206 kg/h/m2.
The filter cake moisture range was 13.7 % to 16.6 %.
The filter cake thickness range was 51.9 mm to 57.4 mm.
The transportable moisture limit values range from 10.4 % to 17 %.
The filtration test work compared pressure filtration against vacuum filtration. Vacuum filtration generally gave a higher moisture content compared to pressure filtration. Pressure filtration was, therefore, selected as the preferred filtration technology.</td></tr>
<tr><td>Acid Base Accounting (ABA) and Net Acid Generation</td><td>The ABA and net acid generation tests were conducted on the column flotation pilot plant tails slurry and the following was determined:
<u>Scavenger Tails</u> are non-acid generating, and their disposal will not produce acidity or significant metal leaching (ML).
<u>Cleaner Tails</u>, however, are strongly acid generating, will apparently oxidise readily, and will represent a high risk for the generation of acidic drainage containing significant metal.
The tests showed that if the scavenger and cleaner tailings are not kept separate in the processing plant, then the combined tailings will be acid generating and would not only represent a risk to the quality of water draining from the waste facility, but could also increase the risk of an earlier onset of ARD from the co-deposited waste rock.

A mineralogical analysis was performed on the tailings samples. With respect to ARD, the minerals of interest are the sulphides providing the potential for acid generation and those minerals capable of neutralising the acidity generated by sulphide oxidation. Pyrite is the dominant sulphide present, with lesser but significant quantities of chalcopyrite and pyrrhotite.</td></tr>
</table>

1.7.2 Flowsheet Development

The Rovina Valley process plant flowsheet was developed from the interpretation of the results of various test work programmes conducted by SGS Lakefield, Eriez and Pocock laboratories on samples from the Colnic and Rovina porphyry orebodies. The flowsheet comprises primary crushing and stockpiling, milling (SAG, ball and pebble crushing), roughing and scavenger flotation, concentrate regrind, three-stage cleaning, concentrate dewatering and bagging, tails dewatering and dry stacking.

1.7.3 Process Design

The proposed process plant design is based on a well-known and established conventional concentrator plant flowsheet with major unit operations selected based on their suitability for the duty, reliability and ease of operation and maintenance.

Table 1.12 outlines the key process design criteria developed for the Rovina Valley selected flowsheet/processing route, which is shown in Figure 1.11.

Table 1.12: Summary of Key Design Criteria

Parameter	Unit	Value		Source
		Colnic Open Pit	Rovina Open Pit	
OPERATING SCHEDULE				
Plant				
Throughput	Mt/a	7.2	7.2	Client
Crushing				
Overall Utilisation/Availability	%	70	70	SENET/Industry
Annual Operating Hours	h	6,171	6,171	Calculated
Primary Crushing Product Size P_{80}	mm	150	150	OMC
Milling				
Overall Utilisation/Availability	%	91	91	SENET/OMC
Annual Operating Hours	h	8,000	8,000	Calculated
Mill Circuit Product Size (P_{80})	µm	75	75	Test work
Concentrate Regrind Product Size (P80)	µm	13.5	13.5	Test work
PRODUCTION SCHEDULE				
Copper Production				
Overall Head Grade – LOM	%	0.10	0.24	DRA Mining
Overall Recovery – LOM	%	88.5	92.5	SENET/Test work
Overall Product Grade – LOM	%	20.3	22.1	SENET/Test work
Gold Production				
Head Grade – LOM	g/t	0.62	0.26	DRA Mining
Overall Recovery – LOM	%	80.0	78.8	SENET/Test work
Product Grade – LOM	g/t	65.87	22.92	SENET/Test work
FILTRATION				
Scavenger Tails Filter Cake Moisture Content	%	14.5	14.5	Test work
Cleaner Tails Filter Cake Moisture Content	%	14.5	14.5	Test work
Concentrate Filter Cake Moisture Content	%	8.5	8.5	Test work

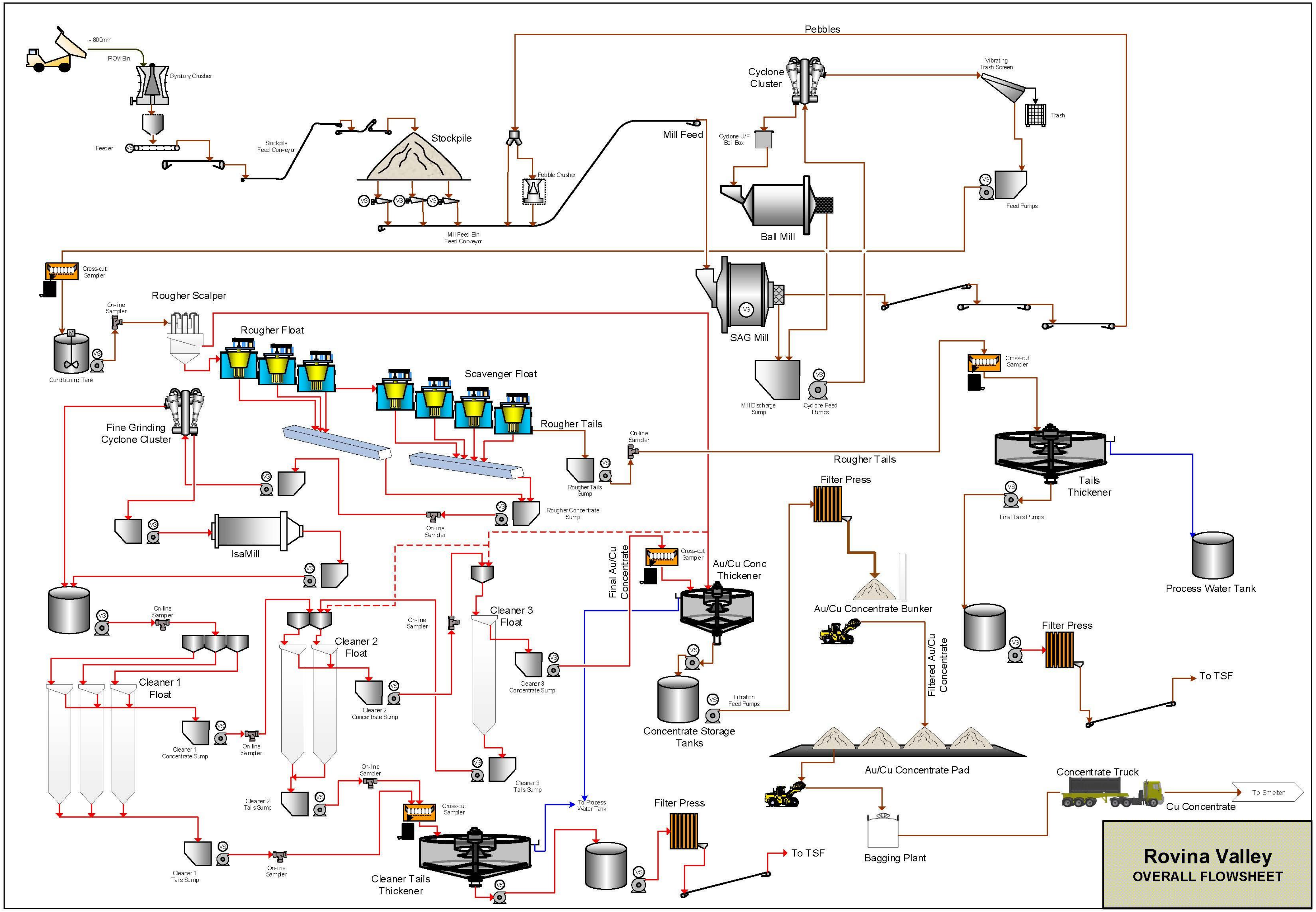


Figure 1.11: Process Flowsheet

1.8 WASTE MANAGEMENT FACILITY

Geotechnical and geochemical test programmes have been undertaken on samples of tailings produced from a flotation test programme carried out in 2018 (ERIEZ 2018). It is anticipated that two tailings products will be produced from the process plant:

- A cleaner tailings stream
- A rougher-scavenger tailings stream

Geochemical testing has shown that the cleaner and rougher-scavenger tailings are strongly PAG and Non-PAG, respectively. These materials will therefore be transported separately to the waste storage facility by means of conveyors and will be placed in separate designated areas of the waste storage facilities. It is anticipated that the majority of the rougher-scavenger tailings stream will be mixed on-conveyor with waste rock prior to placement. Mixing the Non-PAG tailings with the predominantly PAG waste rock will provide several geochemical benefits and will lower the risk of acid generation and associated metal leaching from the waste rock at least in the short to medium term, allowing rehabilitation of the facility to prevent risks of ARD in the long term. Production waste rock is anticipated to be crushed prior to transport to the waste storage facilities by conveyor. Separate storage of the PAG cleaner tails will allow full containment, collection and treatment of ARD that will likely be produced.

The waste from the Colnic pit and Rovina pit will be stored in two waste management areas:

- **Co-Disposal Facility (CDF)** – Designed to maximise capacity for storage of tailings and waste rock from the Colnic pit and the Rovina pit, within the given permitted boundary. Will receive the following:
    - All cleaner tailings sourced from both the Colnic pit and the Rovina pit.
    - Co-mingled rougher-scavenger tailings and waste rock sourced from the Colnic Pit and the Rovina Pit, from Year -1 to Year 9.
- **Colnic Pit Infill** – Designed to store the remaining waste rock and rougher-scavenger tailings produced from the Rovina Pit, from Year 10 to Year 16.

The general arrangement of the waste management facilities is shown in plan in Figure 1.12.

The CDF configuration extends north-south between the Rovina pit and Colnic pit and stores cleaner and co-mingled waste on an annual basis to accommodate the production schedule. The geometry of the CDF footprint was constrained to an area already permitted by ESM, bounded by the access and haul road alignments, and was developed with a design philosophy to store all cleaner tailings in one area. Underdrains will be constructed in stream beds within the footprint of the CDF to collect groundwater seepage and precipitation infiltration, and thereby draw down the phreatic surface within the CDF.

The Colnic Pit will be backfilled with co-mingled rougher-scavenger tailings and waste rock once the storage capacity of the CDF is reached, following Year 9. Cleaner tailings will continue to be stored at the CDF during this period. The geometry of the Colnic Pit infill was developed to allow for the return of the Rovina Valley drainage to its natural course at project closure; a commitment made by ESM in project permitting. To achieve this arrangement, the facility was designed to allow for a 20 m wide channel to pass through the facility, and with slopes that mesh well with the natural topography at closure.

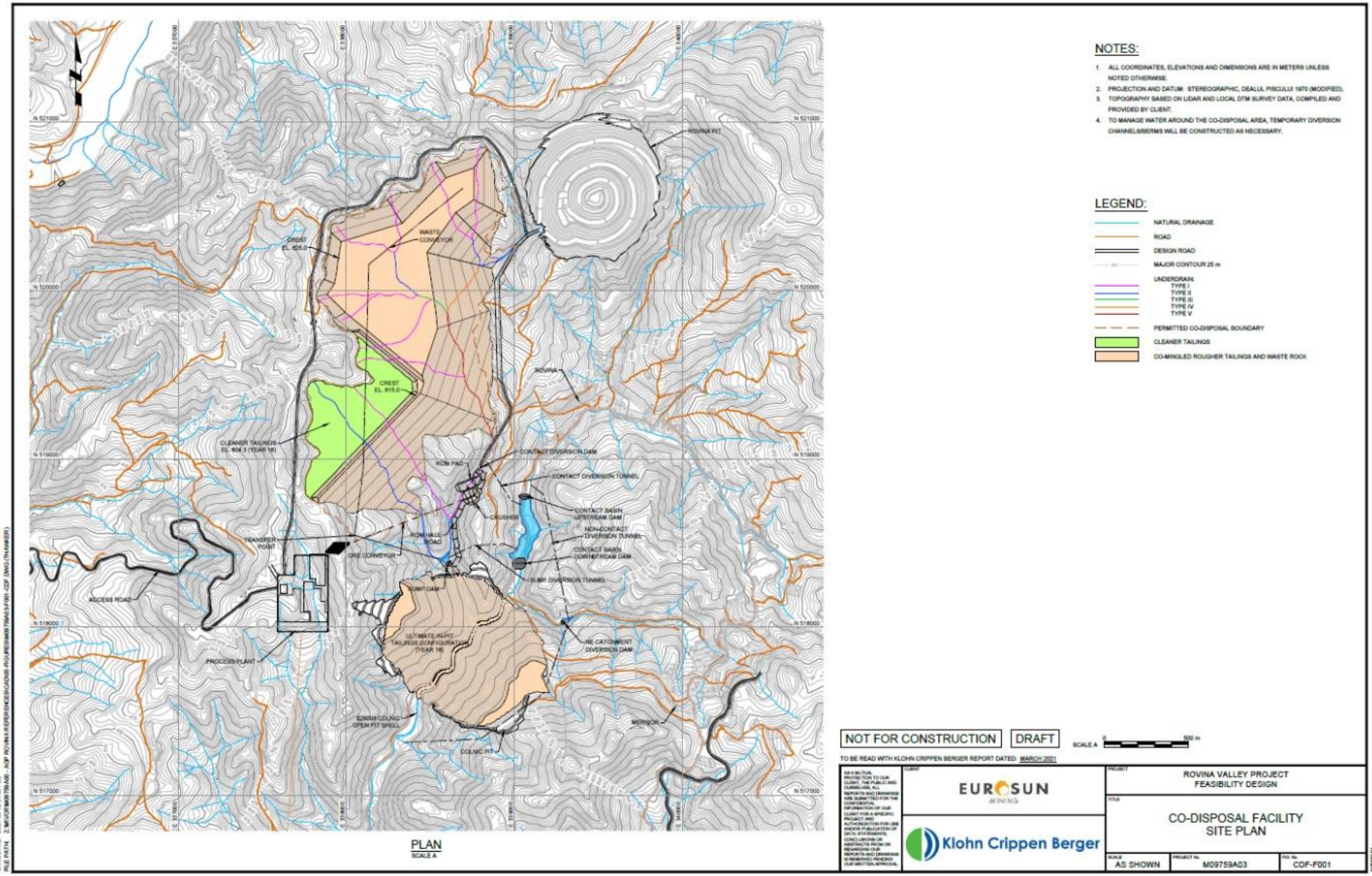


Figure 1.12: Waste Management Facilities – Plan

1.9 HUMAN RESOURCE AND OPERATIONAL READINESS

1.9.1 Human Resources

The ESM human resource element is a very vital part of ensuring the operational success of the RVP. Significant consideration has been given in the recruitment strategy to ensure a seamless transition between commissioning and normal operation. ESM also recognises the necessity for the operation to employ a sustainable localisation plan within the Rovina Valley mine area and nationally, and as such part of the policy is to recruit locally as far as practicable and implement a skills development plan for Romanian nationals with focus on those local to the mine site.

In order to effectively manage the operations for the RVP, a labour schedule was drawn up to include labour for mining, process plant and administration duties, and to describe the labour complement that will be required for the RVP.

1.9.2 Overall Mine Management Structure

Figure 1.13 shows the overall management structure proposed for the RVP. The mine management will be structured in five main departments: the process plant, mining, finance and material, human resources, and the environmental departments. All the respective departmental managers will report to a general manager, who will be responsible for the mine's overall operation.

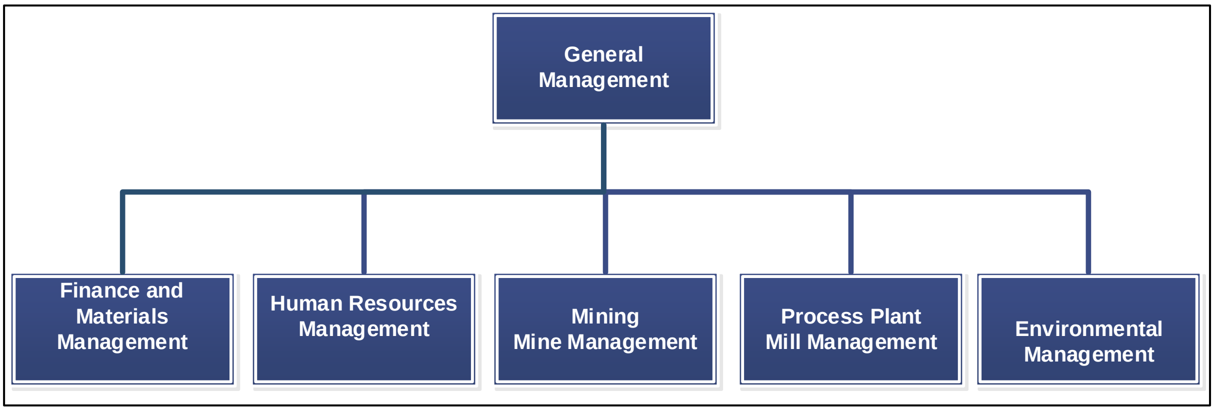


Figure 1.13: Overall Mine Management Structure

The process plant division will include a labour complement for the following:

- Mill Operations
- Mill Maintenance and Engineering
- Mill Metallurgy

The mining division will include the following departments:

- Mine Management
- Open-Pit Engineering
- Geology and Grade Control

- Mining Technical Services
- Mine Planning
- Mine Supervision
- Mining Operations

The mining complement changes according to the strip ratio and grade distribution during the mining operation duration of the Colnic pit and then Rovina Pit. These changes are reflected in Figure 1.14.

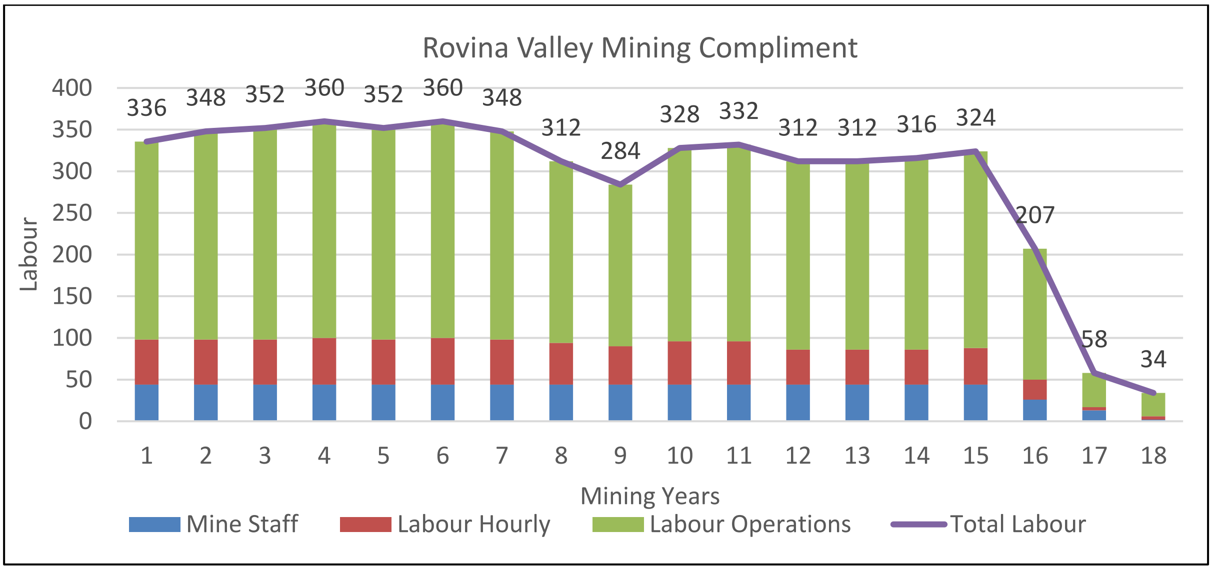


Figure 1.14: Rovina Valley Mining Complement

The finance and materials division will include the following personnel:

- Manager – Finance and Administration
- Senior Administrative Assistant
- Senior Accountant – Supervisor
- Accountant
- Manager – Purchasing and Logistics
- Materials Management Superintendent
- Senior Logistics Officer
- Senior Purchasing Officer
- IT Coordinator
- IT Assistant

The human resources division will include the following personnel:

- Manager – Human Resources
- Human Resources Officer
- Community Relations Coordinator
- Manager Health, Safety and Training
- Senior Officer – Health, Safety and Training
- Training Foreman

- Nurse
- Assistant Nurse

The environmental division will include the following personnel:

- Environmental Manager
- Environmental Officer

Table 1.13 gives a summary of the total labour complement, while Figure 1.15 shows the overall labour distribution as a percentage per division.

Table 1.13: Total Average Labour

Distribution	Number	Percentage
Mining Operations	**332**	75 %
Process Plant	**84**	19 %
General and Administration	**27**	6 %
Total	**443**	**100 %**

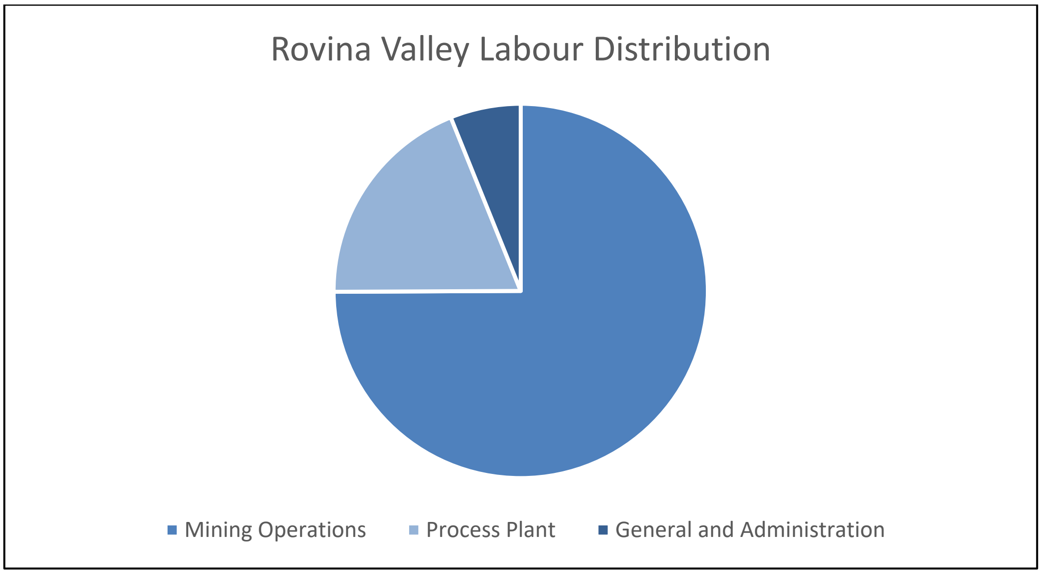


Figure 1.15: Total Labour Distribution

1.9.3 Operational Readiness

An Operational Readiness Plan (ORP) is recognised as a key factor in the successful delivery of capital projects. The ORP guides the organisation from the conceptual phase through design, construction, commissioning, ramp-up and steady-state operations by addressing all the aspects required to implement a successful operation. These include developing systems, programmes and materials required for start-up, maintenance systems and operational manuals. Training programmes linked to the personnel ramp-up, hands-on training, systems roll-out, commissioning planning and production ramp-up. The preparation of operational systems and procedures, such as safe work instructions to ensure that the plant can be operated in an efficient and safe manner, must be planned for.

The purpose of an implemented ORP is to mitigate risks during the transitional period from project to an operating asset. Risks are mitigated by early identification of operational alignment and asset management issues. The mitigation includes developing clear staffing dates to ensure sufficient time for operations, maintenance and support groups to be trained and ready to operate on functioning and implemented systems, processes and procedures.

The ORP spells out how the project will be phased into the operating business. It describes what will be done, how it will be done, in what sequence it will be done, and who will do it.

The ORP can be split into four main categories:

- Business Readiness
- Commissioning Readiness
- Maintenance Readiness
- Processing Readiness

1.10 PROJECT ON-SITE INFRASTRUCTURE

The RVP is a greenfield project, and as such, infrastructure has yet to be established on the project site. The on-site infrastructure required will be related primarily to the processing plant and the supporting facilities as follows:

- In-plant access roads
- Plant prefabricated buildings, i.e. plant administration offices, change rooms, metallurgical/assay laboratory, gatehouse, and weighbridge control room
- Plant steel and cladded infrastructure buildings, i.e. process plant warehouse, plant reagents and consumables stores, including various compressor buildings
- Security buildings and perimeter security fencing
- Process plant site-wide water drainage system including terrace pollution control culverts
- Process plant infrastructure buildings sewage reticulation and treatment
- Raw water storage facilities and reticulation
- Process water storage facilities and reticulation
- Potable water treatment, storage facilities and reticulation
- Fire water storage, pumping system and reticulation
- Compressed air facilities and reticulation
- Communication systems and reticulation
- Power distribution/reticulation including backup diesel generator facilities

1.11 PROJECT OFF-SITE INFRASTRUCTURE

The proposed project off-site infrastructure will support the mining operations, process plant peripherals, and waste facility operations.

The main off-site infrastructure required for the development of the RVP will be the following:

- Within the mining licence boundary:
  - A waste facility perimeter road providing access from the process plant to the mining industrial area (MIA)
  - Mining haul road networks between the Colnic and Rovina pits, including the ROM pad and mine waste crushing station
  - Overland conveyor networks for the transportation of the mine ore, mine waste and rougher/cleaner dry tailings
  - MIA prefabricated buildings, i.e. mining administration offices, mine change rooms and a gatehouse
  - MIA steel and cladded infrastructure buildings, i.e. mine warehouse, mine heavy-vehicle workshop, welding workshop, mine-heavy vehicle wash bay and tyre store
  - MIA site-wide water drainage system, including terrace pollution control systems and oil/water separators
  - Power distribution from process plant switchyard and reticulation to all operating nodes, i.e. MIA, mine ore and waste crushing stations, Colnic and Rovina pit dewatering pump stations, raw water catchment dam and wastewater treatment plant pump station
  - Fuel storage facility and dispensing systems, i.e. diesel fuel dispensing for the mining fleet and support vehicles
  - MIA communication systems and reticulation
  - MIA potable water storage facility, pumping system and reticulation including feed to potable water storage vessels for potable water supply to nearby villages namely Rovina and Merişor
  - MIA fire water storage facility, pumping system and reticulation
  - MIA raw water storage facility, pumping system and reticulation
  - MIA infrastructure buildings sewage reticulation and treatment
  - Series of contact and non-contact water catchment dams, water collection sumps, water transfer tunnels, and water diversion channels
  - Wastewater treatment facility and pump station

- Outside of the mining licence boundary:
  - A main access road from the DN74 to the process plant
  - A new public access road from the DJ741 to the Merişor and Rovina villages
  - Potable water storage to allow for a supply to the nearby Rovina and Merişor villages

1.12 MINE CLOSURE AND REHABILITATION

ESM is committed to following international best practices regarding Environmental, Social and Governance (ESG) principles throughout the development and operation of the project. Some examples of these best practices are that ESM has consciously designed the project with a dry-stack tailings deposition facility and eliminated the use of cyanide in the ore processing. ESM have chosen a high degree of stakeholder engagement from the outset, with consultation on the layout of infrastructure, and have responded to community concerns, such as tailings stability and the use of cyanide. The decision to undertake early, comprehensive surveys enabled ESM to receive an Archaeological Discharge Certificate and plan a layout that avoids impacting any sites in this historical region. Together, these decisions by ESM mark their commitment to responsible and sustainable mining.

An RVP Mine Closure Plan (MCP) was developed in 2014 during a preliminary Environmental Impact Assessment (EIA). This remains relevant to the current design and layout. The RVP MCP is designed to be in line with national requirements and international good practice, e.g. European Union (EU) Extractive Waste Directive, EU Best Available Techniques, International Finance Corporation (IFC) Health and Safety Guidelines on Mining, and the International Council on Mining and Metals Good Practice Guide. Romanian Mining Law (85/2003, and the norms and orders relating to it) requires a financial guarantee for environmental rehabilitation, to be constituted at a bank approved by the National Agency for Mineral Resources (NAMR), to cover the operations for each subsequent year, to be renewed annually.

Beyond the progressive rehabilitation of the waste facilities and passive care for the surface water management systems which are part of the RVP operating costs, mine closure is estimated to be US$20 million.

The approach to closure will be to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use. The current MCP vision is to restore most of the site to pre-mining land use and status, namely mixed upland forest. The aim of the MCP will be to minimise or eliminate long-term active aftercare such as water treatment requirements.

Monitoring of groundwater and surface water (levels and chemistry) will be undertaken throughout the LOM to monitor the impacts of mine dewatering and develop an appropriate closure strategy to ensure that adverse impacts on hydrogeology and hydrology do not occur post-closure. Upon closure, there will be a phase of further site investigation, risk assessment and regulatory liaison to identify any sensitivities or requirements relating to post-closure soil or water quality and to develop remedial action plans that may be required in this regard. Closure impacts on the community will be assessed at closure with community continuity projects developed to assess and mitigate these impacts.

Mine closure activities will vary between the main project elements:

- Processing Plant and MIA
- Rovina Pit
- Waste Facilities including the Colnic Pit

1.12.1 Processing Plant and MIA

Structures no longer required will be dismantled or demolished and removed from site with foundations being removed and voids backfilled. Waste shall be removed from site to a suitable disposal area, and the land shall be restored through topsoil replacement and planting of forest.

Drinking water facilities and associated infrastructure used to serve the local communities will be retained and transferred to the local authorities.

1.12.2 Rovina Pit

The Rovina pit closure concept is for this to form a lake. To achieve this, the pit slope gradients will be adjusted, and subsoil and topsoil will be deposited on the upper part of the pit slopes, which are above the design lake elevation, and planted to forest.

1.12.3 Waste Facilities

The waste management facility (WMF) closure and rehabilitation comprise closure of both the CDF and the infilled Colnic pit.

1.12.3.1 CDF Closure and Rehabilitation

The reclamation strategy involves placing a 0.5 m thick vegetative cover followed by grass mixes for long-term cover. The reclamation strategy for the CDF will be executed separately for cleaner tailings and co-mingled waste:

- For cleaner tailings, closure design involves installation of a liner over the exposed tailings surface after Year 16. The liner will consist of a primary geomembrane and secondary clay layer with an appropriate vegetated protective cover to encapsulate the cleaner tailings.
- For the co-mingled waste areas, progressive reclamation will be carried out as soon as the final facility exterior slopes are established, and will consist of applying a vegetative cover to the final facility exterior slopes. This activity will be executed as an ongoing operation on the final exposed slope.

1.12.3.2 Colnic Pit Rehabilitation and Closure

At closure of the Colnic pit infill, after Year 16, the reclamation strategy involves placing a 0.5 m thick vegetative cover followed by grass mixes for long-term cover to the final facility exterior slopes. This activity will be executed annually for a portion of the final exposed slope after each annual raise. A final closure rehabilitation programme will be executed for the co-mingled tailings area slopes and crest after Year 16.

After completion of the Colnic pit infill, and as part of the closure operations, the 20 m wide channel will re-establish the Rovina Valley stream flow back to the streambed downgradient of the Colnic pit.

1.12.4 Monitoring in Closure

Throughout decommissioning and into the closure period, site monitoring will include a programme of surface water and groundwater, air emissions and ecological monitoring.

1.13 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT STUDIES

1.13.1 Setting and Baseline

The RVP is located in a rural, mainly forested location in the Rovina Valley in the Apuseni Mountains. All the main project components are located in a small tributary valley running north from the Rovina Valley, apart from the Colnic pit which straddles the Rovina Valley

Stream. The tributary valley will be significantly altered by the Rovina pit in the upper headwaters and the waste facilities of co-mingled tailings and waste rock that run the entire length of the western slopes. The Rovina Valley Stream will be diverted in a tunnel around the Colnic pit and downstream of the diversion, the stream course will be dammed and used as the contact water pond, fed by drainage captured in the tributary valley and from pit dewatering.

Although critical habitats are not likely to be present, the forests, streams and meadows have high biodiversity and host numerous listed species. Despite the presence of small areas of historical mine workings and waste, soil, water and groundwater quality is good in the area, as is air quality.

The village of Rovina is the closest settlement, and eleven plots in its western outlying district are within the footprint of the waste facilities and haul roads. Most of the village is close to the mine footprint and will be impacted to some extent. Residents currently rely on fountain and well water, which baseline surveys have found to be of poor quality, and part of the RVP plan is to use some of the raw water supply to treat and provide potable water to the residents of Rovina.

1.13.2 Risks, Impacts and Mitigation

Key environmental and social risks are similar to those associated with other gold mining projects and include the social license to operate; air quality and noise impacts; safeguarding rivers and biodiversity by mitigating permanent effects; and risks associated with land acquisition and resettlement.

The RVP has attracted the attention of international and national NGOs and national media interest. Local community relationships in this historical mining region are good, although community engagement has highlighted concerns around the environmental impact and consequent effects on human health. Land acquisition will be complex, and resettlement will also pose challenges that need to be carefully managed.

Air quality from dust and noise impacts are likely to affect the closest residents, although the extent is not currently known as baseline surveys and mitigation studies are not yet complete.

The diverse forest habitats in the RVP area may meet the criteria for European Bank for Reconstruction and Development (EBRD) priority biodiversity features and IFC natural habitat, and will require further investigation and evaluation. Impacts will be managed by the application of the mitigation hierarchy and careful biodiversity-led management of re-forestation during the closure phase.

The RVP will cause high-magnitude impacts on soils, surface water and groundwater. However, these impacts and risks can be mitigated, to the extent that they are not likely to be significant, by the designs adopted by the RVP and by careful management of the specific structures, practices and plans that have been developed to protect them. The mine closure process is a critical element of this mitigation, based on a closure concept of re-forestation of the waste facilities, plant areas, and one of the pits, with the other pit being converted to a lake. In the long term, once geochemical and geotechnical stability has been achieved, the Rovina Valley, stream though altered, can return to flow without requiring active treatment.

1.14 CAPITAL COSTS

The capital cost (CAPEX) estimate includes engineering, procurement, construction, start-up and cold commissioning for the mining, process plant, WMF and related infrastructure. Provision has also been made for the Owner's costs.

The estimate is within the required accuracy level of +15 % −10 %. The estimate covers the direct field costs of executing the project; the indirect costs associated with the design, construction, and commissioning of the new facilities; and the Owner's support costs for items such as management teams, operational staff recruitment and training, environmental, permitting, insurance and utilities such as water supply, bulk power, and construction power.

The total CAPEX for the RVP was estimated to be **US$446,976,422,** which includes project execution, EPCM, contingency and sustaining capital costs. The initial CAPEX and total CAPEX by work breakdown structure (WBS) areas is summarised in Table 1.14.

Table 1.14: Initial/Development and Sustaining CAPEX Summary

WBS Area	Area Description	Initial CAPEX (US$)	Total CAPEX (US$)
1000	Mine	47,659,761	95,394,603
2000	Feed Preparation	12,369,609	12,369,609
3000	Process Plant	157,606,769	157,606,769
4000	Waste Management	28,294,405	28,294,405
5000	Plant Infrastructure	24,401,210	24,401,210
6000	Off-Site Infrastructure	21,420,000	21,420,000
7000	Indirect Costs	62,778,880	62,778,880
9000	Owner's Costs, Freight and Contingency	44,710,946	44,710,946
	TOTAL	**399,241,580**	**446,976,422**

The total sustaining capital cost for the RVP was estimated to be **US$47,734,842,** which includes project execution, EPCM and contingency costs.

1.15 OPERATING COSTS

The purpose of this operating cost (OPEX) estimate is to provide operating costs, and the associated general and administrative (G&A) costs, to an accuracy of +15 % −10 % that can be utilised for the economic analysis of the RVP.

The project's annual OPEX estimate for the LOM consists of the following:

- Mining operating costs estimated by DRA Mining
- Process plant operating costs estimated by SENET
- WMF and Surface Water Management (SWM) operating costs estimated by KCB
- Site G&A costs estimated by ESM

The overall LOM OPEX for the RVP is summarised in Table 1.15.

Table 1.15: Overall LOM Plant OPEX Summary

Description	Cost		Cost Distribution
	US$/t Feed	US$ Million	%
Mining	6.23	743.8	43.6
Process Plant, Assay	7.34	876.1	51.3
WMF and SWM	0.44	52.2	3.1
G&A	0.29	34.6	2.0
TOTAL	**14.30**	**1,706.7**	**100**

1.16 MARKETING AND FINANCIAL ANALYSIS

1.16.1 Marketing and Sales

A study was conducted to identify copper smelters which might provide the best commercial terms in the European and Asian markets for the concentrate produced by the RVP. The presence of gold, together with the acceptable levels of impurities, makes the RVP copper concentrate marketable.

A key finding from this review is that the market for copper and gold concentrate is very strong at present with a positive outlook.

The RVP concentrate is of a high quality with regard to contained penalty elements. Table 1.16 lists typical penalties for deleterious elements in copper concentrates and the magnitude of the penalty when limits are exceeded.

Table 1.16: Typical Penalty Schedule for Copper Concentrate

Element	RVP Expected Concentrate Range (%)	Threshold (%)	Penalty (US$/t per extra 0.1 %)
Arsenic	< 0.005–0.015	0.2	2
Antimony	< 0.001	0.05	15
Bismuth	< 0.022	0.02	25
Cadmium	0.003–0.013	0.03	30
Cobalt	0.006–0.013	0.5	1.0
Fluorine	Not measured	0.03	15
Lead	0.049–0.13	1.00	0.3
Mercury	Not measured	0.0005	3,000
Nickel	0.007–0.014	0.5	1.0
Selenium	0.007–0.024	0.03	15
Zinc	0.39–1.763	3.00	0.3

The design moisture content for the RVP concentrate is < 8.5 % moisture and is within the smelter moisture threshold limit of < 10.65 % moisture.

In the long-term, the number of importers of copper concentrate is projected to increase in Asia Pacific. China is expected to continue to hold a major market share in the global total imports of the copper concentrate in the coming years.

India is an emerging country in terms of copper production and is expected to be a leading importer of copper concentrate during the forecast period.

Smelters which are within the RVP region and have capacity for custom concentrate are located as follows:

- Bor, Serbia
- Pirdop, Bulgaria
- Hamburg, Germany
- Huelva, Spain

With the RVP concentrate not containing unacceptable amounts of any deleterious elements, and the valuable metals (e.g. gold) contained within the concentrate, it would be considered marketable in the main international markets with no penalties for impurities.

1.16.2 Financial Analysis

Financial analysis of the RVP was done by building a discounted cash flow model which applied the market price, production schedules, capital expenditure and operating cost data from the DFS to forecast the project cash flows from construction through to closure.

The resulting cash flows are shown in Figure 1.16 for each year and on a cumulative basis. The first two years (2023/2024) are negative due to the establishment capital expenditure. Cash flow is positive thereafter as full operations commence. The project breaks even on a cash basis in 2029, 4.8 years after the start of production. The net present value (NPV) (at a real discount rate of 5 %) before tax is US$447 million with an internal rate of return (IRR) of 21.3 %, and the NPV after tax is US$359 million with an IRR of 19.2 %. The project is robust in the face of changes to key value drivers, showing a positive NPV for decreases of up to 32 % in the gold price.

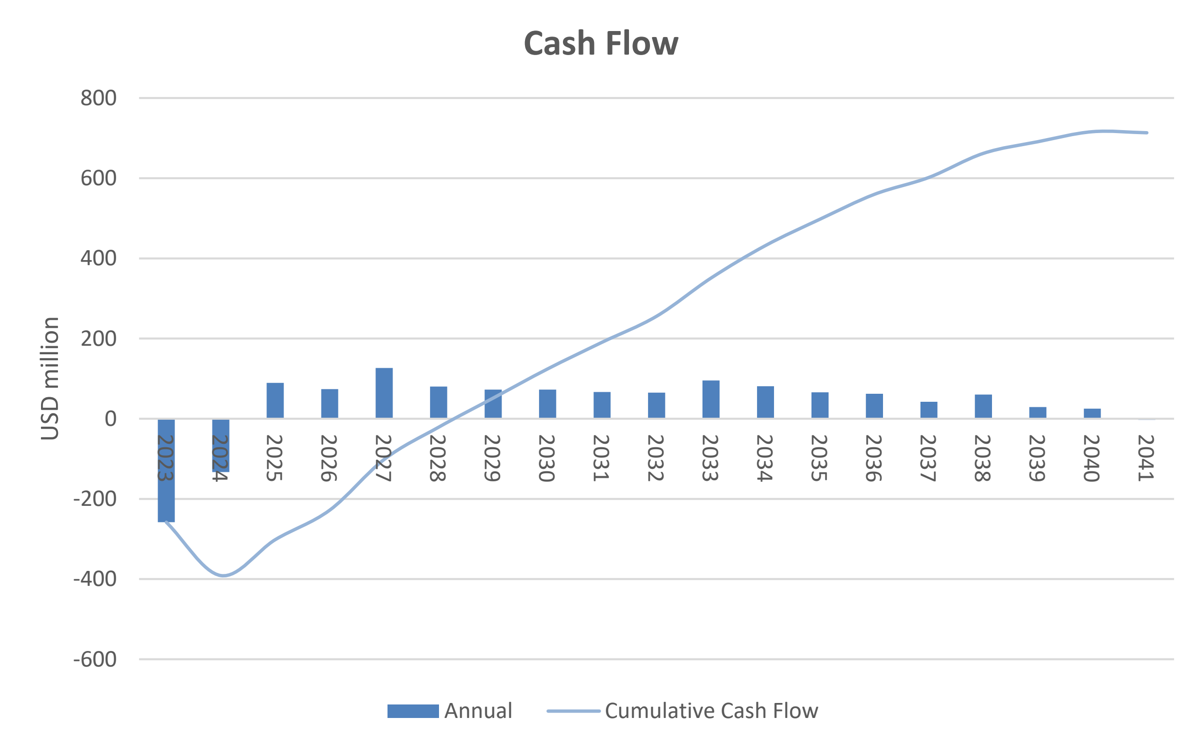


Figure 1.16: Project Cash Flow

1.17 IMPLEMENTATION

The project implementation schedule has been compiled to ensure that the engineering, procurement, and construction management (EPCM) activities are aligned for successful project execution. ESM will appoint an EPCM Contractor to execute or manage and oversee the execution (by appointed EPCM/EPC/turnkey subcontractors, who will report to and be managed by the EPCM Contractor) of the following work packages:

- Detailed Surveys
- Geotechnical Investigations
- Roads Designs and Construction
- Power supply from main grid
- Raw water supply including catchment dams and tunnels
- Mine industrial area infrastructure
- Process plant construction and on-site infrastructure
- Waste facility predevelopment and drainage
- Mining operations – Colnic pit predevelopment
- Process plant commissioning and ramp-up

The following packages will be owner operated and managed directly by ESM, who will execute the predevelopment requirements and progress these packages into an operational environment:

- Mining operation including pre-strip
- WMF including predevelopment

The project schedule assumes that there will be a seamless advancement between the various phases of the project evolution. It is also recognised that this is a moderately aggressive schedule and that it will require diligent progress monitoring and coordination of all the parties involved. The project milestones are given in Table 1.17.

Table 1.17: Project Execution Milestones

Project Milestone	Project Month
Commence early works design	Month −13
Commence road designs	Month −13
Commence geotechnical site investigations	Month −10
Commence main power line design	Month −11
Commence raw water supply design	Month −13
Complete process plant access road and Merișor road designs	Month −4
Commence process plant access road and Merișor road construction	Month −4
Complete geotechnical site investigations and report	Month −1
Complete initial process design	Month −6
Place orders for long-lead delivery items (e.g. ball and SAG mills)	Month 1
Commence process plant, mining, and waste facility detailed designs	Month 1
Commence procurement and contracts administration activities	Month 1
Complete main power line design	Month 2
Commence main 36 km buried power line construction	Month 3
Complete process plant, mining, perimeter road and waste facility detailed designs	Month 4
Complete process plant access road and Merișor road construction	Month 20
Commence construction activities, i.e. waste facility including perimeter roads, dam walls and tunnels, mining terracing and MIA interim infrastructure, and process plant infrastructure	Month 14
Complete main 36 km buried power line construction	Month 14
Commence main power line switchyard construction	Month 25
Complete dam walls, water tunnels and water diversion channels construction	Month 20
Complete perimeter road (Colnic to MIA) and waste facility underdrains and pre-strip construction activities	Month 20
Commence MIA and fuel depot construction including Colnic pit predevelopment	Month 14
Complete procurement, manufacture, and equipment deliveries to site	Month 24
Complete process plant, MIA, mining predevelopment and main power line switchyard construction	Month 32
Commence process plant commissioning	Month 33
Complete process plant commissioning	Month 36
Ramp-up to 100 % of nameplate production	Month 37

Placing the purchase orders for the long-lead equipment is crucial not only to ensure that the equipment is on site in time to allow for a seamless construction sequence and a successful project execution but also to obtain the certified information from the supply vendors on their equipment to complete the detailed engineering phase of the project.

Based on a manufacturing period of greater than five months, the identified long-lead equipment for supply to the RVP is as follows:

- Ball and SAG Mills
- Regrind Mill
- Gyratory and Pebble Crushers
- Apron and Pan Feeders
- Flotation Cells
- Thickeners
- Filter Press Plants
- Cyclone Cluster
- Pumps (Cyclone Feed Pumps and Overland Pumps)
- Medium-Voltage Switchgear
- Mining Fleet

1.18 RISKS, OPPORTUNITIES, CONCLUSIONS AND RECOMMENDATIONS

1.18.1 Hazard and Operability Studies

Hazard and operability (HAZOP) studies are used to identify potential hazards in a system and to determine where operability problems could be encountered. The HAZOP process promotes the undertaking of HAZOP 1 and HAZOP 2 studies at the process development stage and process and project definition stage, respectively, of a project. Further HAZOP studies are then undertaken from the project design stage through to operation. In total, there are six different HAZOP stages to progress through over the life of a project from start to full operation.

During the March RVP DFS development, HAZOP 1 and HAZOP 2 studies were performed with an external facilitator, ESM personnel and various consultants.

1.18.2 Risks

A qualitative risk assessment was performed with an external facilitator to ensure a broad view on the potential project risks. The technical and management disciplines within the project group were represented by the stakeholders' contribution to the risk register development.

The risk register was developed from first principles and evaluated the pre-control probability, frequency, and impact, as well as the post-control probability, frequency, and impact, of each risk.

The risks given in Table 1.18 were identified.

Table 1.18: Pre-Control and Post-Control Risks Identified

No.	Risk Name	Risk Realisation	Pre-Control Risk Weight	Post-Control Risk Weight
R01	Community and NGO Challenge to Mine	Project	25	24
R02	Environmental Permits	Project	13	9
R03	Land Acquisition	Project	13	1

No.	Risk Name	Risk Realisation	Pre-Control Risk Weight	Post-Control Risk Weight
R04	Geology Yield	Operations	6	6
R06	Slope Angles	Operations	13	8
R08	Colnic Pit Water Inflow	Operations	25	10
R09	Mine Operations	Operations	21	13
R10	Consumables Supply	Operations	9	9
R11	Plant Recovery	Operations	13	13
R13	Final Concentrate Moisture Content	Operations	13	13
R15	Moisture	Operations	13	9
R16	Hardness	Operations	18	18
R17	Equipment Selection	Project	9	6
R18	Contamination	Operations	13	8
R19	Catastrophic Failure of Embankment	Operations	15	10
R20	Forest/Vegetation Fire Hazard	Operations	18	18
R21	Pandemics	Project	13	13
R22	Qualified Workforce	Operations	21	18
R23	Project Funding	Project	19	1
R24	Commodity Price, Forex, Escalation	Projects and Operations	9	5
R26	a. CAPEX b. OPEX	Project	6	3
R28	Political Change	Project	18	18
R29	Water Balance	Operations	6	6
R30	Equipment Transport Delays	Project	18	18
R31	Late Design Changes	Project	17	9
R32	Board Approval of Changes	Project	13	8

1.18.3 Opportunities

The opportunities given Table 1.19 in were identified.

Table 1.19: Pre-Control and Post-Control Opportunities Identified

No.	Risk Name	Pre-Control Risk Weight	Post-Control Risk Weight
R05	Geology Yield	6	6
R07	Slope Angles	13	13
R12	Plant Recovery	13	13
R14	Final Concentrate Moisture Content	13	13
R25	Commodity Price, Forex, Escalation	9	5
R27	a. CAPEX b. OPEX	6	3

1.18.4 Conclusions

The RVP has undergone a significant amount of exploration, testing and study work to demonstrate what is believed to be a significant resource and reserve. This can be seen with the amount of detail expressed in this report and within the referenced documentation supporting this report. The level of accuracy used herein is sufficient to consider this report to be compliant with the NI 43-101 requirements with its demonstration of the technical feasibility to develop a concentrator at the RVP that will produce the current reserve of 1,826 koz of gold and 208,231 t of copper in saleable concentrate form over a 16-year LOM.

This report has demonstrated that the RVP ore deposits can be economically mined using the open-pit mining method with waste crushing and conveying and processed through a flotation circuit at a planned annual throughput of 7.2 Mt/a.

There is a risk of a high bench turnover rate in the final four years of the Colnic pit production schedule. This risk will be mitigated in the next optimisation phase based on the new information detailed below:

- The recently completed geotechnical and hydrogeological studies have shown that there is some potential to steepen the pit bench slope angles in the Colnic pit by 5°, which will improve the project economics by reducing the strip ratio and lowering the total mining cost in the early RVP LOM.
- The completed Colnic waste storage facility design has shown that there is sufficient storage capacity for waste rock and processed tailings so that the mining rate at Colnic can be slowed. This will still allow for the waste crushing and conveying infrastructure to be moved from the Colnic pit during the Rovina pit predevelopment phase. This extra Colnic waste storage facility capacity will mitigate operational risk in operational life of both the Colnic and Rovina pits.

1.18.5 Recommendations

1.18.5.1 Geology

In preparation for mine development, ESM should consider the following recommendations for operational readiness:

- Conduct pre-production drilling within the mine planning footprint for the first 24 to 36 months of ore mining. This will ensure that all mineral resources mined during this period are converted to Proven mineral reserves, thereby minimising geological risks during this crucial payback period.
- Implement database management software to manage and monitor sampling QA/QC programmes. This system will flag batches that are beyond the threshold limits and allow for immediate remedial action. Also, where there is evidence of positive or negative drifts, the laboratory can be notified to calibrate their equipment more regularly.
- Research the implementation of Leapfrog Implicit modelling of geological and geostatistical domains during grade-control modelling. This will allow for quick and efficient updating of the geological models when turnaround times are crucial.

- Undertake a geostatistical study to determine the optimum drill spacing for grade-control modelling. Optimum drill spacing will assist with time and costs during mine production.
- Research the correlation between the results from a mobile X-ray fluorescence (XRF) scanner and those from a laboratory. This will be useful for grade-control sampling and modelling as follows:
  - If there is a reliable proxy between the Cu and Au grades in the deposit, then the XRF readings for Cu can be used as a proxy for anticipating the Au grades.
  - If a reliable proxy can be established, this will facilitate quick and cost-efficient turnaround times for grade-control assays.

1.18.5.2 Flotation Configuration

Dilution cleaner test work has been carried out to determine the performance of the scalper rougher Jameson flotation on the Rovina Valley ores. When available, the results will confirm and validate the rougher circuit configuration and equipment sizing while minimising the capital and operational costs.

An investigation of the mineralogical characteristics of the high zinc content and corresponding flotation reagents schemes is required during operations to effectively depress zinc without decreasing gold recovery.

1.18.5.3 IsaMill Signature Plots

IsaMill signature plots tested on primary milled composites from the Colnic and Rovina orebodies showed significantly higher specific energy requirements compared to initial estimates used in the DFS. Since the mainstream composite samples are not necessarily representative of the concentrate from ore, for a better understanding of what the true specific energy would be, it is recommended that a further signature plot test on a rougher-scavenger concentrate be investigated.

1.18.5.4 Mining

The recently completed geotechnical and hydrogeological studies have shown that there is some potential to steepen the pit bench slope angles in the Colnic pit by 5°, which will improve the project economics by reducing the strip ratio and lowering the total mining cost in the early RVP LOM. This is to be incorporated in design optimisations.

1.18.5.5 Hazard and Operability

It is recommended that the further stages of the HAZOP studies be followed as the project progresses through design and development. This would start with the HAZOP 3 and electrical area classification studies.

1.18.5.6 Engineering Studies

A more detailed field based seismic hazard assessment will need to be carried out in the early stages of detailed design. This is necessary to identify and assess any potentially active faults in the vicinity of the project site. Additional climate studies that include precipitation, snowpack, evaporation monitoring, wind studies and collection of continuous flow and precipitation monitoring for the catchments will refine the design assumptions regarding precipitation, base flows, time of concentration and losses in the local catchments.

1.18.5.7 Site Investigations

Additional geotechnical and hydrogeological research and testing in the following areas will allow final design documents to be submitted for the on-going permitting as well as the construction tender process.

- Foundations at the processing plant site.
- Subsoil conditions along the proposed road alignments.
- The proposed CDF and WMFs, including the location of each proposed dam.
- The proposed tunnel alignments and their portals.

Furthermore, an additional geotechnical investigation at the Rovina Pit and additional geotechnical characterisation of the Flysch unit located in the upper NE parts of the Colnic Pit will bolster the designs presented in this report.

1.18.5.8 General

Given the promising results from this March 2021 DFS, SENET would expect ESM to continue to advance the RVP. The next steps are expected to include ongoing permitting activities and culminating in obtaining an operating licence. During this phase it would be prudent to commence with detailed engineering and SENET recommends that ESM start the Front-End Engineering Phase of the project to be ready for project construction once all licences are obtained.

2 INTRODUCTION

2.1 OVERVIEW

Euro Sun Mining Inc. (ESM) is a mid-tier Canadian exploration and development company with its head office based in Toronto, Ontario and is listed on the Toronto Stock Exchange, TSX:ESM.

The RVP is situated in the Hunedoara county of Transylvania in western-central Romania. The RVP consists of three deposits: Rovina to the north, Colnic centrally, and the Ciresata deposit to the south. This report deals with only the Rovina and the Colnic deposits; the Ciresata deposit can be brought into the project for development later. The Rovina exploration licence is held by SAMAX Romania SRL (SAMAX), a Romanian registered company which is a wholly owned subsidiary of ESM. Since November 2018, ESM has possessed an exploitation permit and mining licence with a renewable 20-year validity.

The Colnic and Ciresata deposits are described as gold-copper porphyries while the Rovina deposit is termed a copper-gold porphyry. All three of these deposits have access to a central processing plant.

The RVP processing facility is being designed to produce a gold and copper concentrate from the Colnic and Rovina deposits.

The RVP is within the Golden Quadrilateral Mining District of the South Apuseni Mountains, an area with a history of mining dating back to Roman times. This has supported the development of excellent infrastructure including rail, power, and paved access roads. In addition to this, there are two international airports in the cities of Timisoara and Sibiu, less than 180 km from the project location. There is also the town of Brad (population 15,000) within 5 km of the project site, from where there will be a good source of local skilled labour. Sourcing the right skills and resources locally supports ESM’s community upliftment opportunities.

This NI 43-101 Technical Report was compiled by SENET for ESM with contributions from the QPs as set out in Table 2.1 to support ESM’s press release dated 01 March 2021 and to summarise the results of a DFS of the RVP.

Table 2.1: Qualified Persons and Their Contributions

Qualified Person	Company	Contribution
Nicholas Dempers	SENET (South Africa)	Metallurgical test work interpretation Processing plant and project infrastructure Economic evaluation Coordination and compilation of report
David Alan Thompson	DRA (South Africa)	Mining, mineral reserves
Sivanesan Subramani	CCIC MinRes (South Africa)	Geology and mineral resources
Robert Cross	KCB (Canada)	Waste storage facility Geotechnical
Carlos Diaz Cobos	KCB (Canada)	Hydrotechnical
Andrew Hovey	KCB (Canada)	Hydrogeology

Qualified Person	Company	Contribution
Richard W. Lawrence	Lawrence Consulting Ltd	Geochemistry
Kevin Leahy	ERM (United Kingdom)	Environmental Social Assessment Permitting

2.2 TERMS OF REFERENCE AND PURPOSE OF THE REPORT

SENET was commissioned by ESM to compile an NI 43-101 Technical Report on the RVP by coordinating the contributions from SENET's QP and the QPs from the other consultants. The report is compiled in accordance with National Instrument 43-101 (NI 43-101) and Form 43-101F1.

The intention of this report is to present the findings of a DFS undertaken on the Colnic and Rovina open-pit deposits of the RVP. The mineral resource model supporting this DFS was completed in July 2012 by AGP and had a data cut-off date of May 31, 2012. Since then, two new exploration drillholes have been added to the Colnic deposit, and three new exploration drillholes have been added to the Rovina deposit. In 2019, AGP validated that the new drilling data will not materially affect the mineral resources prior to reporting the February 2019 updated mineral resource estimate. In 2021, CCIC MinRes also assessed the new drillholes at Colnic and Rovina to ensure that they would not materially affect the 2021 mineral resource estimate. Following a thorough audit of the July 2012 resource model and the 2019 mineral resource estimate, the 2021 mineral resource estimate was updated by incorporating current metal prices and operating parameters to determine a new Lerchs-Grossmann mineral resource constraining shell.

The Ciresata (underground) deposit was outside the scope of this study, hence the mineral resource estimate remains unchanged since the February 2019 estimate by AGP.

2.3 SOURCES OF INFORMATION AND DATA CONTAINED IN THE REPORT

SENET would like to acknowledge that most of the text for Sections 4 to 12 and 14 in this report was sourced from the AGP Mining Consultants Inc. "Rovina Valley Project Preliminary Economic Assessment, NI 43-101, Rovina Valley, Romania". Effective date 20 February 2019.

SENET would also like to acknowledge the contributions made by Mr Randall K Ruff, P. Geo., Executive Vice President, Exploration with ESM, for providing the regional, local geological, and historical information on the Rovina Valley deposits. While not a QP on this report, Mr Ruff was involved during the primary data collection and on-site geologic interpretations.

For further details on references, please refer to Section 27.

This report represents the independent opinions of the QPs based on the available source data, as supplied by ESM. The opinions are premised on historical data received from AGP as well as additional recent exploration drilling data. AGP has confirmed that, to the best of their knowledge, the information provided by them was true, accurate and complete, and not incorrect, misleading or irrelevant in any aspect. The QPs do not have any reason to believe that any material facts have been withheld. The historical data supplied by AGP was checked and verified to the extent possible.

2.4 QUALIFIED PERSONS' PERSONAL INSPECTION OF THE PROPERTY

A summary of the QP's qualifications and responsibilities is given below.

Mr Nicholas Dempers, MSc Eng (Chem), BSc Eng (Chem), BCom (Man), Pr Eng (RSA), Reg. No. 20150196, FSAIMM (RSA), a principal process engineer at SENET, is the QP for the mineral processing, metal recoveries and metallurgical testing sections and oversaw the compilation of the project infrastructure and the capital and operating costs as per the SENET quality management system. Mr Dempers did not visit the project area. By virtue of his education, as well as relevant work experience and membership of recognised professional associations, he is a QP as defined by the NI 43-101 guidelines.

Mr David Alan Thompson a Principal Mining Engineer for DRA Projects (PTY) Limited of 3 Inyanga Close, Sunninghill, Johannesburg, South Africa, a graduate of University of Johannesburg with a Baccalaureus Technologies Degree in Mining Engineering is the Qualified Person for the mining sections of this report while contributing to a number of other sections as detailed in this report. Mr Thompson visited the property from the 09 November to 13 November 2020 and inspected the project site and all relevant areas of interest with regards to the Project and reviewed all technical documentation available for the project to date. By virtue of his education, as well as relevant work experience and membership of recognised professional associations, he is a QP as defined by the NI 43-101 guidelines.

Mr Sivanesan Subramani, a principal mineral resource geologist at Caracle Creek International Consulting MinRes (Pty) Ltd, holds a BSc Honours degree (Geology and Economic Geology), and is a registered Professional Natural Scientist with the South African Council for Natural Scientific Professionals (Pr.Sci.Nat. Reg. No. 400184/06). Mr Subramani is a member of the Geological Society of South Africa and a member of the Geostatistical Association of Southern Africa. Mr Subramani visited the project site and core storage facilities from 9 to 12 November 2020.

Mr Cross is a registered Professional Engineer with Professional Engineers Ontario (PEO Reg. No. 100173823) and is a registered Professional Geoscientist with the Association of Professional Geoscientists of Ontario (APGO Reg. No. 2845). He graduated with a Master of Engineering (MEng) degree in Geological Engineering from the University of British Columbia in 2009 and a Bachelor of Applied Science (BASc) degree in Geological Engineering from the University of British Columbia in 2007. Mr Cross has practiced his profession since graduation and has been directly involved in the geotechnical engineering of tailings and water dams, including design and construction projects, construction monitoring, and field investigations. He has worked in Canada, USA, Mexico, Peru, Mauritania, Burkina Faso, and Kazakhstan. Mr Cross was involved in the planning and execution of the geotechnical and hydrogeological investigation campaign undertaken during 2020. During this time, he visited the project for a site visit in November 2020. During this site visit, he reviewed and assisted in the implementation of the logging and sampling protocols and carried out a site reconnaissance walkover of the various design areas.

Mr Diaz is a registered Professional Engineer with Professional Engineers Ontario (PEO Reg. No. 100191866). He graduated from Universidad Pontificia Bolivariana, Bucaramanga, Colombia in 2001 with a Bachelor of Engineering degree in Civil Engineering, and from the University of Toronto in 2005 with a Master of Applied Science degree. Mr Diaz has practiced his profession since graduation and has been directly involved in the civil engineering design and review of water retention and conveyance structures, including for operational and closed mine facilities in Canada, Finland, Romania, Mexico, Brazil, Argentina, Papua New Guinea and Mauritania. Mr Diaz did not visit the project area during the DFS.

Mr Hovey is a registered Professional Geoscientist with the Australian Institute of Geosciences (AIG). He graduated from the University of Queensland in 2001 with a Bachelor of Science (Honours) degree in Earth Sciences. Mr Hovey has practised his profession since graduation and has been involved in the assessment and design of mine pits, underground operations and mine residue storage facilities in Australia, Canada, Papua New Guinea, Romania, Vietnam, Saudi Arabia, Mongolia, the Philippines and Indonesia. Mr Hovey did not visit the project area during the DFS.

Dr Kevin Leahy, BSc (Hons) (Geological Sciences), PhD (Diamond Exploration), CGeol (Geological Society, UK), SiLC (Specialist in Land Condition, Land Forum, UK), is a Technical Director at Environmental Resources Management (UK) and is the QP for the environmental and social assessment and permitting section. Dr Leahy did not visit the project area but has worked closely with ERM colleagues in Romania who have been undertaking baseline surveys for RVP, and is involved in the ongoing environmental and social impact assessment.

Dr Richard W Lawrence, BSc (Mining Engineering), PhD (Biohydrometallurgy), P.Eng. (British Columbia, Canada, Reg. No. 22564) is the Principal of Lawrence Consulting Ltd and is the QP for the geochemistry sections. He visited the project site in June 2011 to inspect the locations of the major mine components and examined the exploration drill core for examples of typical lithology and alteration. He subsequently selected samples and managed geochemical test programmes to determine the acid rock drainage (ARD) and metal leaching (ML) potential of the low-grade ore and waste rock. More recently, he managed a geochemical test programme on flotation tailings with the same objectives.

3 RELIANCE ON OTHER EXPERTS

The information, conclusions, opinions, and estimates contained in this report are based on the following:

- Information available at the time of preparation of this report
- Assumptions, conditions, and qualifications as set forth in this report
- Data, reports, and other information as supplied by ESM and other third-party sources

For this report, the authors have relied on ownership information provided by ESM. In the consideration of all the legal aspects relating to the RVP, the authors have relied on ESM and assumed that the information relating to the legal aspects and the status of surface and mineral rights is accurate.

Property information in this report has been sourced from previous reports supplied by ESM. The authors are not responsible for the accuracy of any property data, and do not make any claim or state any opinion as to the validity of the property disposition described herein.

For the preparation of this report, the authors relied on maps, documents, and electronic files generated by the current and past exploration crews on behalf of ESM. To the extent possible under the mandate of an NI 43-101 compliant report, the data has been verified relating to the material facts.

Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party’s sole risk.

According to ESM, there are no known litigations potentially affecting the RVP.

4 PROPERTY DESCRIPTION AND LOCATION

The information for this section was sourced from the AGP Mining Consultants Inc. “Rovina Valley Project Preliminary Economic Assessment, NI 43-101, Rovina Valley, Romania”. Effective date 20 February 2019 and edited where necessary.

The Rovina Exploitation Licence lies in the Judetul (County) Hunedoara, a part of the Development Region of Transylvania. Regionally, it is located in the Golden Quadrilateral Mining District of the South Apuseni Mountains in west-central Romania, approximately 300 km northwest of the city of Bucharest (the capital city of Romania), and 140 km east–northeast of the city of Timisoara (see Figure 4.1).

Figure 4.1: RVP Location Map

Locally, the property is approximately 25 km north of the city of Deva, which is the administrative centre for the county, and 7 km east of the town of Brad for which mining has played an important role (see Figure 4.2). The Golden Quadrilateral has a long history of gold mining, which predates the Roman occupation through several periods of activity to the results of modern exploration efforts, which have defined two other advanced-stage gold projects, Rosia Montana (Gabriel Resources) and Certej (Eldorado Gold). From the Rovina Licence, Rosia Montana is approximately 25 km northeast, and Certej is 17 km southeast.

The property is centred at approximately latitude 46°07’ N and longitude 22° 54’ E or 515,000 N and 340,000 E using the “Stereo70” projection of the Romanian National Geodetic System. Elevations on the property range from 300 m to 780 m above sea level.

The Rovina, Colnic, and Ciresata porphyry deposits are the principal exploration targets on the property, with their locations defining a north-northeast trend. The Rovina porphyry is the northern-most deposit with the Colnic porphyry lying approximately 2.5 km south of the Rovina porphyry, and the Ciresata porphyry approximately 4.5 km south of the Colnic porphyry (see Figure 4.2). ESM has termed these three deposits the RVP.

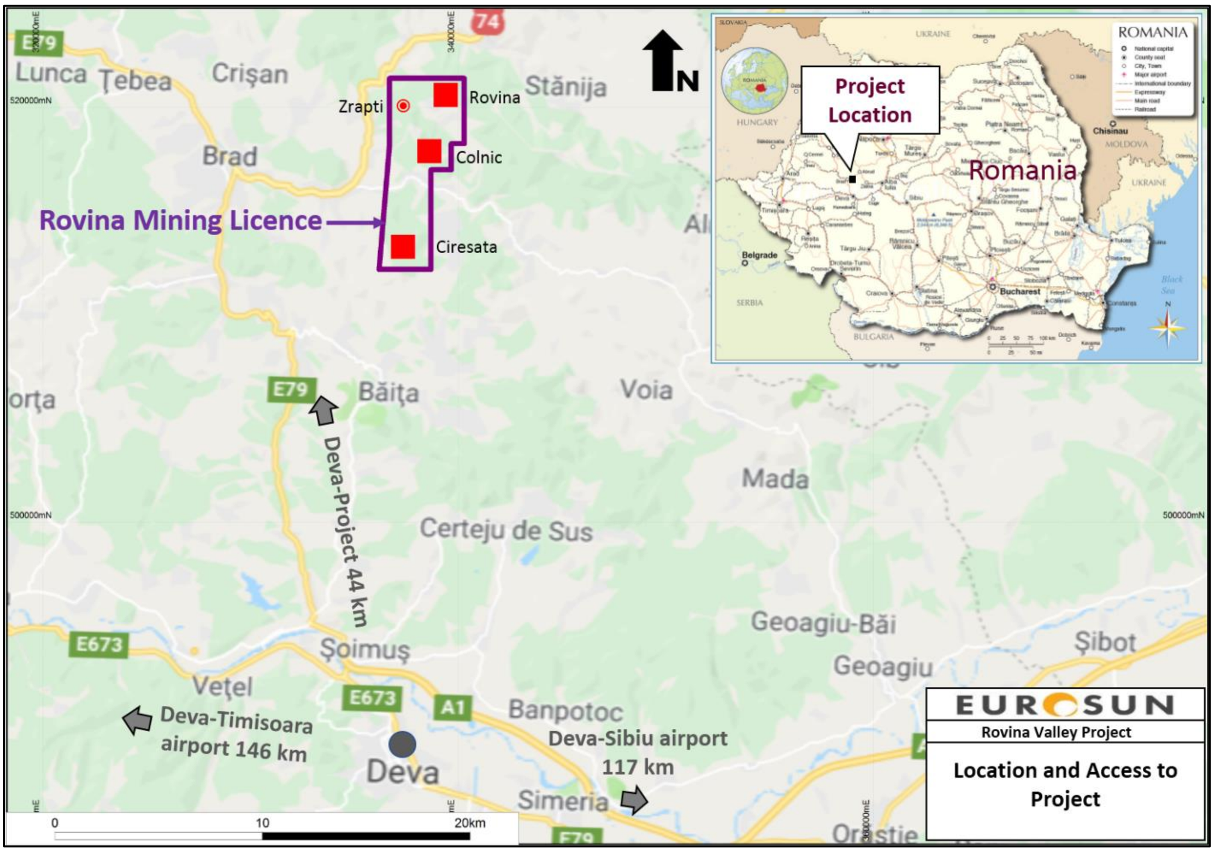


Figure 4.2: Location, Access, and Perimeter of Rovina Mining Licence

4.1 MINERAL DEPOSIT TENURE

The new General Mining Law of Romania came into effect in 1998 and was re-written in 2003. The scope of the law is aligned to international current practice with the intent to ensure maximum transparency in mineral rights administration and fair competition without discrimination between operators, depending on the property type and the origin of the capital. Subterranean and aboveground mineral resources located within Romanian territory, within the continental shelf, and in Romania’s Black Sea economic area are part of the State’s public property.

The mineral rights of Romania are administrated by the state National Agency for Mineral Resources (NAMR), subordinated to the General Secretary of the Government, first created in 1993. The governing of mineral rights is guided by the Mining Law Nr. 85/2003, Norms for Applying the Mining Law Nr. 85/2003 (Official Gazette of Romania, yr. 171, No. 772, November 4th, 2003), and subsequent amendments. The Mining Law does not require state ownership beyond the prescribed royalties and Mining Licence tax fees. The following description of mineral property title in Romania is summarised from an English translation of the 85/2003 mining law provided by the NAMR:

- http://www.namr.ro/wp-content/uploads/2014/03/MLaw_85.pdf
- http://www.namr.ro/wp-content/uploads/2014/03/MNorms1208.pdf

The Mining Law defines and regulates different categories of exploration and mining activities, from early-stage sampling and prospecting to formal exploration and finally commercialisation, exploitation, and processing.

No exploration or mining activity can be legally carried out without an appropriate permit. All companies seeking to conduct exploration or mining activities must first contact the NAMR.

The rights granted by an exploration or exploitation licence are exclusive to the holder, chargeable, defensible against third parties, and are transferable with the consent of NAMR. Applicants are not automatically granted surface rights, and these must be acquired from the existing owner through sale, land exchange, rental, expropriation, concession, association, or other process, as allowed by law. Expropriation is only allowed for public projects deemed to be of national interest.

Foreign operators must set up a permanent subsidiary in Romania within 90 d of obtaining any exploration or mining licence, and the subsidiary needs to be maintained throughout the period of operation.

There are three types of permits available for exploration and mining activities in Romania: Prospecting, Exploration, or Exploitation, as described in the following sections.

4.1.1 Prospecting Permit

The Mining Law defines prospecting as the performance of studies on the surface of a site required to identify the possible existence of an accumulation of mineral resources. To obtain a Prospecting Permit, a company must make a request to NAMR, which must issue a Prospecting Permit within 30 d of receipt of the request. The permit is valid for a maximum three-year period. The Prospecting Permit is non-exclusive and, therefore, does not guarantee the applicant will be granted the rights to the exploration or exploitation of any mineral resources located within the licence area.

Prospecting Permit holders must pay annual land fees equal to RON358/km^2 (approximately US$90/km^2). This land fee may be adjusted for currency inflation. The latest adjustment was made on 8 March 2019. The titleholder of the Prospecting Permit must present annual reports to the NAMR documenting the work completed and the supporting results.

4.1.2 Exploration Licence

The Mining Law defines exploration as including all operations and studies conducted to identify, evaluate, and determine the optimal technical and economic conditions necessary to exploit a particular mineral deposit. The Exploration Licence is valid for a maximum period of five years, with a renewal right of three years and provides the holder with an exclusive right to carry out operations.

Exploration licences are granted by the NAMR following a public tender and bid process. Bids are evaluated on a score-card list of bid attributes including an obligatory first-year work programme. Exploration Licence holders must pay annual land fees to the Romanian government equal to RON1,437/km^2 (approximately US$360/km^2). This fee is doubled after two years and increases five-fold after four years and may be adjusted for currency inflation. In addition, the applicant must submit an appropriate financial guarantee for environmental rehabilitation as set out in an environmental rehabilitation plan.

Proposed yearly work programmes must be filed with and approved by the NAMR; also, half-yearly and annual reports documenting the work completed and the results must be submitted as well as annual inspections from an NAMR representative. The holder of an Exploration Licence has the exclusive right to apply for an Exploitation Licence within the property boundary.

4.1.3 Exploitation Licence

The Romanian Mining Law defines exploitation as including all operations executed at the surface, and beneath it, for the extraction, treatment, and delivery of mineral resources. An Exploitation Licence is granted at the discretion of NAMR and can be awarded to an Exploration Licence holder through directed negotiations or based on a public offer in the case of no current Exploration Licence. Prior to receiving an Exploitation Licence, the successful party must submit the following:

- A feasibility study for the mining operations
- An environmental impact assessment (EIA) and environmental audit
- A minimum investment and development plan
- A mining waste management plan
- A remediation plan to remedy any environmental damage caused by mining operations
- A social impact statement and a social mitigation plan

Holders of an Exploitation Licence must pay annual land fees of RON35,923/km^2 (approximately US$8,980/km^2) of terrain subject to exploitation at surface or underground. This land fee may be adjusted for currency inflation. The latest adjustment was made on 8 March 2019. In addition, a royalty must be paid to the State budget equal to 5 % of the value of the polymetallic and 6 % for precious metals mineral resources extracted from a particular site and is payable quarterly.

According to ESM, the Exploitation Licence holder is also subject to the national corporate profits taxation, which is represented by a flat-rate tax of 16 %. After-tax profits can be repatriated.

Companies legally engaged in exploration and/or mining activities are exempt from duty on imported equipment required for exploration and/or mining.

4.2 ISSUER'S TITLE TO/INTEREST IN THE PROPERTY

The Rovina property consists of one Exploitation Licence (the Rovina Exploitation Licence, Number 18174/2015 for Cu-Au). The corner coordinates for the Rovina Exploitation Licence are described in Table 4.1. These corner coordinates are designated by the NAMR and are not surveyed in the field. The total area covered by the Rovina Property is approximately 2,768 ha.

Table 4.1: Rovina Licence Boundary (Stereo 70 Grid System)

Point	Easting (m)	Northing (m)
1	336,200	518,250
2	336,700	518,250
3	336,700	521,400
4	340,300	521,400
5	340,300	518,250
6	339,670	518,250
7	339,670	517,000
8	338,700	517,000
9	338,700	512,200
10	336,200	512,200

4.2.1 Rovina Exploitation Licence

ESM, through intermediary subsidiaries, owns 100 % of SAMAX Romania SRL (SAMAX), which in turn owns 100 % of the Rovina Exploitation Licence. SAMAX is a duly registered company in the city of Crişcior, Romania.

The Rovina Property was acquired by ESM, through SAMAX, on 27 April 2004 as a one-year non-exclusive prospecting permit covering 102.3 km^2. Following an initial exploration campaign, SAMAX applied for an Exploration Licence, and following a public tender and bid application process, was officially awarded 100 % interest in the Rovina Licence (covering 9,351 ha) on 29 August 2005, for a period of four years (Licence Nr. 6386/2005). As provided by the Mining Law, a three-year extension may be granted. SAMAX applied for this three-year extension which was granted by the NAMR on 20 October 2009 (Act Additional NR. 1, la Licenta De Explorare Nr. 6386/2005), extending the Rovina Licence expiry to 28 August 2012. The final exploration report and resource estimate for the Rovina Licence was submitted, and on 17 July 2012, the NAMR accepted this report and resource statement and invited SAMAX to submit documentation required for a Mining Licence application.

At any time during the valid period of the exploration licence, and 90 d thereafter, SAMAX may apply to convert any part of the licence to an Exploitation Licence. In late 2009, SAMAX retained a consortium of Romanian qualified and certified consulting firms to complete the studies required to apply for a Mining Licence. The topics covered by these studies are listed below:

- Technical Study, to evaluate the economic feasibility of the proposed mining operation with the goal of obtaining state registered reserves.
- Environmental baseline and Impact Assessment Studies, which include water and soil resources, biodiversity, air quality, landscape, and cultural-heritage resources.
- Health and Safety Baseline and Impact Studies, which include local population health status, medical resources, local health and safety issues, and possible impacts over the life cycle of the proposed mining operation.
- Social Study, to define baseline community and financial resources and evaluation of the positive and negative impacts of the proposed mining operation with proposed mitigation measures.

The required studies were completed and were officially submitted to the NAMR on 14 August 2012, comprising the complete Mining Licence Application file. These studies will provide a foundation for the definitive studies required to obtain the operational permits and Licences.

On 26 May 2015, SAMAX signed with NAMR the Rovina Exploitation Licence Contract (No. 18174/2015) for Gold- and Copper- bearing Ore Exploitation. In compliance with the Romanian Mining Law, the Rovina Exploitation Licence was ratified by the Romanian Government on the 9 November 2018 (Governmental Decision No. 900/9 November 2018) and published in the Romanian Official Monitor (Gazette), Part I, No. 970/16 November 2018. The Rovina Exploitation Licence is valid for 20 years, starting on 16 November 2018, and renewable for periods of five years.

Upon any production, SAMAX must pay a 5 % to 6 % royalty to the Romanian Government, as detailed under the Exploitation Licence above. SENET has been informed by ESM that there are no underlying payments or encumbrances to third parties relating to the Rovina Property, beyond the government requirements of royalties and Mining Licence taxes.

4.2.2 Barrick Agreement

On 12 August 2011, ESM (Carpathian Gold at that time) closed a private placement with Barrick Gold Corporation for a non-brokered CDN$20 million private placement to purchase 38,461,538 common shares of the Corporation at a price of US$0.52 per share.

After the private placement, Barrick held approximately 9 % of the issued capital of Carpathian. The agreement provided the proceeds to a certain amount be allocated to the advancement of the RVP, under the guidance of a joint Technical Advisory Committee. Provided that Barrick does not dispose of common shares of the corporation where its interest would fall below 8.5 % in the share capital of the corporation, Barrick has the right to participate in any future equity offerings by the corporation to maintain its pro-rata common share ownership and a right of first refusal, at the asset level only, on any disposition or sale by the corporation of any Romanian property or mineral rights. This agreement does not include any rights to ownership of the RVP (Carpathian News Release dated July 18, 2011 and Annual Information Form 2011 filed on SEDAR.com). Carpathian was subsequently notified by Barrick that they had sold their Carpathian shares and thus no longer retain a first right of refusal.

4.2.3 Surface Rights

ESM does not hold any surface rights on their Rovina property. Romanian law does not vest surface rights with mineral rights and any proposed development requires the developer to either purchase the surface rights or enter into an appropriate agreement with the surface rights owners to have access to the property. According to Romanian Mining Law, upon conversion of their Exploration Licences to an Exploitation Licence, ESM has the right to legally acquire these rights through one of the following processes:

- Sale
- Land exchange
- Rental
- Expropriation, if in the public or national interest
- Application
- Association with an existing owner
- Other process allowed by law

Numerous private individual local landowners and the state forestry hold surface rights over the Rovina, Colnic, and Ciresata deposits. Exploitation activities that result in surface disturbance require permission from the surface rights owners in addition to the required government permits. Land use in the RVP is predominantly low productive deciduous forests, pastureland in valley bottoms, and vegetable fields near households. There are no houses above the Rovina, Colnic, and Ciresata deposits. Mine-site planning from the PEA (Technical Report, February 2019) and March 2021 DFS was designed with consideration to minimise direct impacts to the community, with preliminary indications of direct impact on the number of isolated houses in the range of 5 to 11.

ESM has initiated a land acquisition programme with three phases:

1) Public information campaign
2) Surveying and registration of land parcels not officially registered with the local government cadastral map
3) Acquisition of surface rights using one of the methods listed above

This programme is being implemented by SAMAX through a Social-Community Relations Manager, Legal Team, and Survey Team. The information campaign is well advanced, informing landowners of the legal process of cadastral registration through public postings and public meetings held between 2012 and 2015. ESM informed SENET that public response has been favourable thus far, in part due to SAMAX's long-standing community consultation programme during the project's advancement. Owing to the large number of unregistered land parcels, SAMAX anticipates 1.5 years from the completion of the identification and surveying of all plots to complete all cadastral registrations prior to implementing a land acquisition strategy.

SENET has relied on the terms and the land tenure documentation supplied by ESM and ESM's lawyers and has not reviewed the mineral titles or agreements to assess the validity of the stated ownership.

4.3 ENVIRONMENTAL AND SOCIO-ECONOMIC IMPACT STUDIES

Since joining the European Union (EU) in January 2007, Romania has adopted the General Framework Law 294/2003, which requires a compulsory EIA process for certain projects. The Guidance Document No. 918/2002 (transposing EU Directives) and four Ministerial Ordinances were adopted, which establish competencies, procedural stages, and instructions including public participation. The environmental protection laws and procedures generally meet with international best practices; however, governmental institutional capacities are still in the building stage, though recently, experience has been gained from large infrastructure projects and the advancement of permitting on other large mining projects. The Romanian EIA process has adopted some guidelines from the World Bank for public involvement and consultation in project planning.

Romanian Government environmental regulations include the 1995 Environmental Protection Law. The major provisions set out in the environmental code include the following:

- Principles and strategic elements that are the basis of the laws
- Right to access information on environmental quality
- Right to information and consultation on the siting of industrial facilities as set out in the Law on Environmental Impact Assessment
- Implementation of EIAs, the results of which are to be made available to the public
- Establishment of liabilities regarding environmental quality rehabilitation
- Management regime for dangerous substances, hazardous waste, chemical fertilisers, and pesticides
- Protection against ionising radiation and safety of radiation sources
- protection of natural resources and biodiversity conservation
- prompt action and reporting when accidental pollution occurs
- prerogatives and responsibilities of the environmental protection authorities, central and local authorities, natural and legal persons
- right to appeal to the administrative or judicial authorities

The Romanian government takes cognisance of environmental concerns relating to mining activities in Romania and their potential trans-boundary impacts. The major areas of environmental concern include soil erosion and degradation, water pollution, air pollution in the south of Romania from industrial effluents, and contamination of Danube delta wetlands. The EU has reviewed mining practices and developed criteria for responsible mining which are included in the EU Mining Waste Directive. This directive came into force in April 2006. Romania became a full member of the EU on 1 January 2007 and adopted the EU Mining Waste Directive in 2008.

In Romania, environmental activism in the form of non-governmental organisations (NGOs) is present and mainly dependent on international organisations. Although several thousand NGOs are registered, only 100 are active (approximately).

The Golden Quadrilateral Mining District has a long history of mining and contains areas with extensive mining disturbance. State-owned and subsidised mining operations were closed in 2007 after a long period of declining investment in operations resulting in legacy environmental issues and high unemployment. The Rovina property lies just east of the Brad-Barza sub-mining district, which was operated by the State until closing in 2007. On the

Rovina Property, there are no previous state-owned mining operations, with previous activities restricted to exploration utilising drilling and limited underground gallery excavations. Previous surface disturbance is restricted to exploration gallery waste dumps at the Rovina deposit, from state exploration in the 1980s and several widely spaced exploration galleries – presently collapsed – in the Colnic deposit area from the 19th century. Under the Romanian regulation, ESM does not assume environmental liability for any of the previous exploration activities.

ESM has completed baseline environmental and social studies under the lead of ERM as required for continuing the Mining Licence Application. These baseline studies serve to document current environmental and social conditions (see Section 20).

ESM has informed SENET that an archaeology baseline study directly over the Rovina, Colnic, and Ciresata deposits, and over the plant and waste WMF locations, was commissioned in 2008 and continued in more detail in 2013 and 2016. These studies were completed by the Dacian and Roman Civilisation Museum, in Deva. No findings were made of archaeological or cultural significance (Dacian and Roman Civilisation Museum, 2016). ESM, in anticipation of completing an ESIA (Environmental and Social Impact Assessment) to international standards, commissioned and received the following studies for the RVP:

- “Position paper for development of the Rovina Licence, Romania,” Golder Associates Canada Ltd, 2007.
- “Environmental and Socioeconomic Compliance with International Standards,” AECOM Canada Ltd, 2009.
- “Rovina Project ESIA Gap Analysis and Work Programme”, AMEC Earth and Infrastructure UK Ltd, 2012.

There are several small rural villages within the boundaries of the property. State-operated mines were the dominant employer in the area prior to closure in 2007. Replacement jobs in the local areas have not been developed and unemployment is high. Many of the residents are engaged in subsistence agricultural activities or have left the area seeking employment. Although there are no houses recorded in the direct vicinity of the Rovina, Colnic, and Ciresata deposits, isolated houses and farms occur within 1,000 m south of Colnic and 980 m east of Rovina (the village of Rovina) along with inside of the WMF footprint.

ESM has informed SENET that as part of its stakeholder engagement approach it is currently active in community partnership programmes and project-communication programmes with the surrounding communities as well as assisting in the funding of small basic infrastructure improvements for communities near the Colnic and Rovina deposits. In addition, ESM maintains close contact with the local mayor and villagers as part of their community relation efforts.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

5.1 ACCESS TO THE PROPERTY

The project lies near three international airports within approximately a two- to three-hour drive from the cities of Sibiu, Timisoara and Cluj. The city of Sibiu has the nearest international airport to the property with the most regularly scheduled commercial flights from various European destinations. From Sibiu, the principal access to the property is via a four-lane highway to Deva and then another 40 km via a paved two-lane highway leading to the historical gold mining town of Brad, followed by secondary paved roads eastward for 7 km, passing through the village of Crişcior and on to the village of Bucureşci, which is located within the property. These roads provide the principal access to the Rovina, Colnic, and Ciresata deposits. The western boundary of the Rovina Licence is located less than 1 km east of the village of Crişcior (population approx. 3,000), where ESM's main office is located.

Access to other portions of the Rovina Licence is via various paved and gravel roads, with tracks suitable for four-wheel-drive vehicles, or along footpaths. Access to the property by road is possible year-round; however, short periods of blockage are possible in the winter due to snow, especially in the higher areas of the Apuseni Mountains.

5.2 CLIMATE AND LENGTH OF OPERATING SEASON

The regional climate is regarded as mild temperate continental. Generally, the winter months are from December to March, and snow is common, though accumulation is typically less than 30 cm. Mean winter temperatures are between –3 °C and –5 °C; however, periods of severe temperatures (as low as –20 °C) can occur. Although field activities can continue year-round in this part of Romania, occasional heavy snowfalls can hamper access for short periods during the winter months.

Springtime temperatures of 5 °C to 10 °C may start in early April, but patchy snow cover could last until mid-May in the forested areas. The summer months, from June to September, have temperatures ranging from 10 °C to 20 °C, with rare maximum highs near 35 °C.

The typical annual precipitation is 800 mm to 1,100 mm.

5.3 LOCAL RESOURCES AND INFRASTRUCTURE

The Golden Quadrilateral Mining district, where the Rovina property is located, has a long history of mining activity, with developed infrastructure to provide electrical power and highway and rail transport. The towns of Deva (pop. 61,100) and Brad (pop. 14,500) are the closest major centres to the Rovina Licence and are, respectively, about 1 h and 20 min drive from the Colnic deposit. The town of Zlatna (pop. 7,500) is situated approximately 30 km east of the property. Local unemployment is high (approx. 50 %). Although the local towns can provide the most basic mining and exploration needs for the early stages of exploration and project development (including accommodation and labour requirements, food, communication services, and other supplies), specialised mining-related equipment and services may be obtained from Timisoara (pop. 319,300), Bucharest (pop. 1.9 M), or other European locations. The majority of the workforce, however, will likely be recruited locally. Population figures were sourced from the October 2011 Census.

In the Golden Quadrilateral Mining district, electrical power is distributed from an existing 110 kV utility grid that is connected to regional power producers. The nearest electrical power source to the deposit is in the town of Crişcior (Gura Barza, adjacent to the Brad-Barza Mine processing plant), located approximately 5 km to the southwest. Most locations on the property have cellular phone service, except in valleys where signals may be blocked. Most nearby towns have line telephone service, the majority of which are capable of international calls.

The closest rail line available for use is in the town of Brad, 5.3 km by road from Crişcior, with another rail line in Deva, located 41 km by road from Crişcior. The most significant source of surface water in the Rovina Licence area is the Cris River, which flows just to the north and west of the licence boundaries. A civic works water dam on the Cris River is under construction 2.8 km northwest from the Rovina deposit. Within the Rovina Licence, the smaller, year-round Bucureşci River (south of Colnic) and its tributaries in the property area, have provided an adequate water supply for historical exploratory drilling programmes. With proper upgrades, the property has sufficient sources of power and water to support a mining operation.

5.4 PHYSIOGRAPHY, FLORA, AND FAUNA

The southern Apuseni Mountains are mostly gently rolling with some abrupt slopes and cliff-forming rock exposures. The highest points near or within the property are the Duba (969 masl), Coasta Mare (786 masl), and Cornetel (695 masl) peaks. In the areas of the Rovina, Colnic, and Ciresata deposits, the terrain is hilly to mountainous, with access through relatively gently sloped narrow valleys with moderately steep slopes to rounded ridges. The minimum and maximum elevation ranges, for each of the deposits are Colnic – 350 m to 540 m, Rovina – 500 m to 680 m, and Ciresata – 420 m to 480 m.

The property is mostly forested with deciduous trees (beech and oak), with occasional conifers, particularly at higher elevations.

Wildlife on the property includes deer, fox, and wild pigs. Local streams on the property are not known to have fish.

6 HISTORY

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

6.1 SUMMARY

Mining has played a significant role in the history of the Southern Apuseni Mountains and has been traced back to pre-Roman times (~2,000 years). Gold and base metal mining has occurred principally within the Metaliferi Mountains in an area covering 2,400 km^2 and has become known as the Golden Quadrilateral (GQ) for its prolific historical gold production. Initially gold production came from alluvial deposits and high-grade veins in various locations, including Rosia Montana, Baia de Aires, Zlatna, Brad, and Sacaramb. According to studies of early papers and historical documents, the GQ has produced an estimated 55 Moz of gold with approximately half this production attributed to the Roman Period (Vlad and Orlandea, 2004).

The exploration history on the Rovina property, and particularly on the Colnic, Rovina, and Ciresata deposit areas, can be divided into the following six work phases:

- Local prospectors and miners (19th century)
- Romanian government (1960s)
- Minexfor–Deva, which is the local Romanian state exploration company (mid-1970s to 1997)
- Rio Tinto (1999 to 2000)
- Minexfor–Deva (2000 to 2003)
- ESM (formerly Carpathian Gold) (2004 to 2012)
- ESM (2020)

6.2 LOCAL PROSPECTORS AND MINERS (19TH CENTURY)

The first recorded exploration in the area dates to the 19th century, during the Austro-Hungarian period and was focused on Au–Ag vein-style mineralisation. Within the Colnic Porphyry alteration halo, 17 documented underground galleries were excavated (see Figure 6.1). Few records exist apart from location maps which accurately document their extent; however, according to ESM, no significant production has been recorded and most of the veins were reportedly determined to be sub-economic.

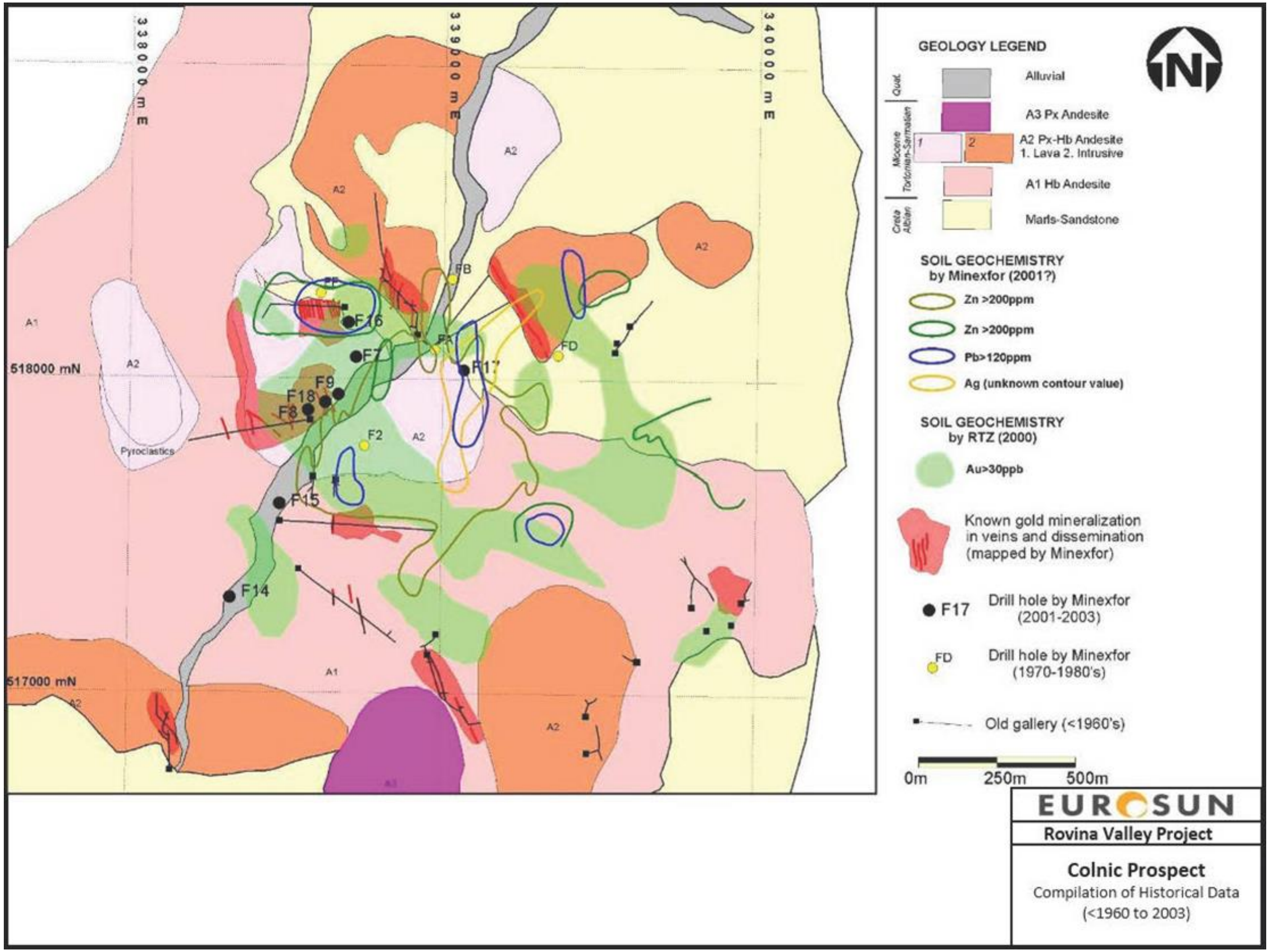


Figure 6.1: Colnic Deposit – Compilation of Historical Work

6.3 ROMANIAN GOVERNMENT (1960S)

In the 1960s, three additional galleries, Colnic, Ursoi, and Mihai, were completed by the Romanian government within the alteration footprint of the Colnic porphyry. The Ursoi tunnel was approximately 500 m in length and reportedly intersected 46 north-trending veins, eight of which were tested by crosscuts. Veins were recorded as 3 cm to 1.5 m in width (mainly around 20 cm) and mineralised with pyrite and traces of sphalerite, chalcopyrite, and covellite, in a gangue of calcite, clay, and rare quartz.

In addition, many of the previously excavated galleries from the 19th century were reopened and sampled. Unfortunately, all these tunnels are now collapsed and no longer accessible. ESM has obtained some underground assay data for the Colnic and Ursoi tunnels and has incorporated this data into the exploration database, to aid geological understanding.

The Romanian Government initiated porphyry exploration in the South Apuseni Mountains in the early to mid-1970s. This work included airborne geophysics (magnetics) and surface sampling, which led to the discovery of the Rosia Poieni porphyry deposit, located 27 km northeast from the Colnic porphyry and 3 km from the Rosia Montana gold deposit. This deposit has been mined by the state mining company and is presently active on a limited basis.

6.4 MINEXFOR-DEVA (MID-1970S TO 1997 AND 2000 TO 2003)

Following discovery of the Rosia Poieni deposit, regional work by the Romanian State exploration company, Minexfor–Deva (Minexfor), identified the Colnic and Rovina areas as prospective for porphyry Cu mineralisation. In the mid-1970s, Minexfor drilled a 650 m deep vertical hole at Colnic (drillhole F2 in Figure 6.1) targeting a magnetic high anomaly identified from the Romanian Government's airborne magnetic survey. Only a summary log was provided to ESM, which documents weakly anomalous Cu grades (averaging 400 ppm) between surface and 240 m depth and stronger anomalies (averaging 1,000 ppm) from 240 m to 650 m depth. There were no gold analyses.

In 1974, Minexfor initiated a diamond drilling programme at the Rovina deposit (referred to as the Bucureşci-Rovina Property in historical documents) to test the extent of the porphyry style mineralisation. Over the next ten years, 34 holes totalling 23,119 m were drilled and sampled for Cu and intermittently for Au, Ag, Pb, Zn, Mo, Fe, and S. Holes were drilled vertical and averaged 680 m in depth. In addition to the drilling, two levels of underground galleries were excavated in the form of grid patterns through the mineralised porphyry body to provide channel sampling and underground drilling stations (see Figure 6.2).

In 1976 and 1977, induced polarisation (IP)/resistivity, natural polarisation, and gamma ray geophysical surveys were reportedly completed by the Romanian government at Colnic. The surveys were done over ten north–south-oriented lines, spaced 200 m apart. These results, however, have not been provided to ESM.

In 1982, Minexfor returned to the Colnic Deposit area to complete 500 m of trenches and surface pits, 500 m of exploration tunnels, and approximately 3,550 m of core drilling. The drilling consisted of four wide-spaced drillholes (FA, FB, FD, FF) located to the north and east of ESM's current drilling (see Figure 6.1). These holes were vertical and drilled to depths ranging from 490 m to 1,200 m. Only summarised details of these holes were provided to ESM; however, reportedly long intervals of weakly to moderately anomalous Cu grades (230 ppm to 710 ppm) were intersected, starting at depths generally greater than 300 m. There were no gold analyses.

Minexfor completed additional magnetic and gamma ray geophysical surveys during 1983 to 1984, together with a soil geochemistry survey. ESM did not receive any of the results of these surveys. From 1986 to 1987, Minexfor briefly explored the Rugina intrusive-sediment contact-related prospect. Rugina is defined by anomalous values of lead and zinc from surface sampling and is located approximately 500 m to the northeast of Colnic. ESM indicated that to the best of their knowledge this prospect was never drill-tested.

Figure 6.2: Underground Portal Now Collapsed at Rovina

In 1986, after ten years of drilling at the Rovina deposit and development of underground access (see Figure 6.2) a resource estimate for copper was completed. ESM purchased this data from the NAMR. As the resource is not NI 43-101 compliant, and has been superseded by more recent estimates, it is not included in this report.

In 2000, Minexfor returned to the Colnic deposit and completed additional trenching and rock sampling, followed by eight core holes (F7 to F9 and F14 to F18) totalling 1,100 m (see Figure 6.1). These holes were drilled at angles ranging from vertical to almost horizontal in the area of ESM's current drilling, and intersected stockwork-style mineralisation. Gold values appear to be in a similar range as those reported by ESM, but copper was not assayed for by Minexfor. Two of the holes (F14 and F15) were drilled to the south and returned only weak Au anomalies. No additional work was reported by Minexfor after this drilling. ESM purchased the paper drill logs with assay results of this programme from the NAMR. Some of this drill core was preserved and ESM reviewed the available core. Although the core was not in good condition, key mineralisation features were still observable.

In 2002–2003, Minexfor completed a diamond drill programme in Valea Garzii within the Ciresata area. Six vertical drillholes were completed for 1,200 m of drilling to a maximum depth of 200 m. One of these holes (F-4) contained anomalous Au mineralisation increasing with depth; copper was not assayed. ESM purchased the paper drilling logs with assays of this programme from the NAMR.

6.5 RIO TINTO (1999 TO 2000)

In 1999, Rio Tinto was granted a non-exclusive prospecting permit by the Romanian government over an area covering approximately 24 km × 30 km. This area included the current boundaries of ESM's Rovina Licence. After the completion of a reconnaissance-style exploration programme in December of that year, Rio Tinto applied for and was granted an Exploration Licence over this same area. A more detailed exploration programme commenced, comprising the following:

- Regional stream sediment geochemical sampling
- Grid soil geochemical sampling
- Rock chip sampling over selected target areas
- Helicopter-borne magnetic/radiometric survey (Furgo Airborne Corp., Canada)

The exploration work by Rio Tinto identified several targets within their licence; however, the licence was relinquished ahead of schedule after only one year of exploration. ESM purchased Rio Tinto's final exploration report from the NAMR.

6.6 ESM (2004 TO 2020)

In 2004, the Rovina property was acquired by ESM's wholly owned subsidiary, SAMAX, as a one-year non-exclusive Prospecting Permit. That year, ESM purchased and compiled historical data, and completed property-wide reconnaissance sampling and mapping with a focus on known prospects.

In 2005, ESM applied for an Exploration Licence and following an open public tender and application process, was officially awarded the Rovina Licence (covering 9,351 ha).

ESM completed generative exploration work during 2005 and 2006, which included reconnaissance-style rock chip sampling, geologic mapping, and sampling at 1:5,000 scale, soil geochemistry grids and ground magnetometer surveys and IP resistivity surveys.

In 2006, drilling commenced on the property, and to date ESM has completed 314 diamond drillholes, including the 2020 geotechnical drilling, for a total of 140,440 m drilled.

In 2007, field programmes included geologic mapping and sampling. This was followed up with soil geochemistry, ground magnetometer, and IP resistivity surveys. In total, approximately 34 km^2 of ground magnetometer surveys were completed over the property along with 24.55-line km of IP ground surveys. Over 19 km^2 of soil geochemistry surveys have also been completed on the property.

In 2007, AMEC Americas Limited (AMEC) completed an NI 43-101 mineral resource estimate for the Colnic and Rovina porphyry deposits. Only the drilling completed during 2006 was incorporated into this mineral resource estimate, which included 49 drillholes (15, 714 m) from the Colnic deposit and 17 drillholes (8, 435 m) from the Rovina deposit.

Following the 2007 generative exploration programmes and drilling, the Ciresata deposit was discovered in early 2008 with Au-Cu mineralisation occurring 50 m to 100 m below the surface.

In 2009, PEG Mining Consultants Inc. (PEG) completed an NI 43-101 resource estimate for the Colnic, Rovina, and Ciresata deposits incorporating the late 2007 and 2008 drilling results available as of 8 April 2008 for Rovina and as of 30 September 2008 for Colnic and Ciresata. The effective date of the resources was 30 September 2008. Based on the 2008 resource estimate, PEG completed a PEA in April 2010.

Between 2009 and 2010, ESM interrupted the drilling programme and only did geological mapping and surface sampling on the RVP to focus its exploration effort on the more advanced RDM project in Brazil.

Following completion of the PEA in early 2010, ESM returned to Romania and completed four vertical, deep drillholes on the Ciresata deposit to assess the continuity of the upper part of the Ciresata deposit as well as the vertical extent of the mineralisation.

In preparation for a prefeasibility study, ESM resumed drilling on the property in 2011. The aggressive drill programme added 21 drillholes at Ciresata, 18 new infill drillholes on the Colnic porphyry, and 19 additional infill drillholes on the Rovina porphyry. For Colnic and Rovina, the drill programme was focused on converting Inferred resources to Indicated.

In September 2011, through an exploration collaboration agreement between ESM and Barrick Gold Corporation, Barrick initiated a drill target generation and drilling programme to test depth and lateral extensions at the Ciresata deposit as well as satellite targets. Drilling on the Rovina Exploration Licence was halted in 2012 as required by the process of conversion to a Mining Licence.

In October–December 2020, ESM conducted geotechnical drilling at the Colnic and Rovina deposits, as part of a geotechnical study for the March 2021 DFS. In addition to the 2011 geotechnical drilling, three holes were drilled at Colnic and one hole was drilled at the Rovina deposit, and another twelve holes in the WMF and plant areas, totalling 1,311.70 m.

The exploration programme conducted by ESM since 2004 is described in greater detail in the Sections 9 and 10. No further exploration activity was conducted on the Rovina Exploration Licence since the end of the geotechnical drilling programme in December 2020.

7 GEOLOGICAL SETTING AND MINERALISATION

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

7.1 REGIONAL GEOLOGIC AND METALLOGENIC SETTINGS

Most of the mineral deposits in the Romanian region are in the Carpathian Fold Belt, an arcuate orogenic belt which is part of a much larger belt extending westward into Austria and Switzerland and southward into Serbia and Bulgaria, as shown in Figure 7.1. These belts developed during the late Cretaceous and Tertiary periods, following closure of the Tethys Ocean due to the collision of continental fragments of Gondwana with continental Europe and the related subduction of small, intervening oceanic basins (Alderton and Fallick, 2000).

Figure 7.1: Schematic Drawing of the Carpathian Arc and Associated Mineral Deposits

The development of the Carpathian Fold Belt was accompanied by widespread igneous activity, including a suite of late Cretaceous to early Eocene acidic to intermediate intrusive and extrusive rocks known as “banatites”. These rocks are believed to have formed during the early stages of subduction and are host to several copper-molybdenum-iron porphyry and skarn deposits in southeast Romania and extending northeast to the West Apuseni Mountains. The prominent Z-shaped Carpathian fold-and-thrust belt through Hungary, Slovakia, Ukraine, Romania, and Serbia (see Figure 7.1) is associated with the Neogene Alpine orogeny, whereby north-south compressive tectonics in western Europe resulted in the eastward extrusion of the Alpaca and Tisia microplates into present Romania (Neubauer et al, 2005). The South Apuseni Mountains represent a somewhat isolated massif of volcanism and ore deposits within the Carpathian orogenic belt (see Figure 7.1). Neubauer et al. (2005) have proposed a plate tectonic setting for the Carpathian-Panonian orogenic system, whereby the volcanism in the South Apuseni Mountains (which lie approximately 250 km west of the westward-verging subduction front) is related to slab roll-back and break-off. This general subduction tectonic setting, while more complicated than the American Cordillera due to the micro-plate interaction, is typical of most gold (± copper) porphyry deposits worldwide (i.e. Cajamarca Belt, Peru; Maricunga Belt, Chile; Cordillera Central of Luzon, Philippines; [Sillitoe, 2000 and Seedorff et al, 2005]). Associated with this tectonic event are Neogene volcanic and subvolcanic rocks.

In the South Apuseni Mountains, where the RVP is located, these Neogene volcanic and subvolcanic rocks are subdivided into three main groups (Alderton and Fallick, 2000):

- Early Miocene acidic tuffs and ignimbrites
- Mid-Miocene to Pliocene calc-alkaline stratovolcanoes (associated with epithermal and porphyry mineralisation)
- Pliocene to Pleistocene alkaline volcanic rocks

These volcanics intrude and overlie a basement of Palaeozoic (and older) metamorphic rocks, Mesozoic ophiolites, and sedimentary flysch rocks. The structural setting in this area is interpreted to be associated with an extension within a strike-slip regime in the Carpatho-Pannonian realm (Milu et al. 2004, Neubauer et al. 2005), which created pull-apart basins. Intersections of these basins with major east–west and northeast-trending pre-Laramian tectono magmatic lineaments and northwest-trending Laramian lineaments are believed to have concentrated areas of increased Tertiary volcanic and metallogenic activity (Balintoni 1994; Rosu et al. 2004, 2000a in Milu et al. 2004).

Regional dating studies from various locations within the South Apuseni Mountains indicate several Neogene calc-alkaline volcanic centres, representing three main episodes of volcanism, were active between ca. 15 Ma and 1.6 Ma (Rosu et. al., 2004, in Manske and Hedenquist, 2006). Volcanism commenced in the east of the Apuseni Mountains (Rosia Montana-Bucium and areas adjacent to the Brad district), moved westward (into the Zarand basin), and then back eastward (Baia de Aries, Zlatna, Sacaramb, and Deva areas) to the final phase at Uroi (see Figure 7.2 and Figure 7.3).

The first volcanic episode was explosive, occurring ca. 15 Ma, is poorly developed/preserved and is represented by rhyodacite to dacitic tuffs that are interbedded with marls and deep-sea pelagic sediments (Cioflica et al. 1996 and Rosu et al. 2000b, in Milu et al. 2004). The second episode (7.4 to 14.8 Ma) is predominantly represented by calc-alkaline intermediate subvolcanic intrusives, with some andesite extrusives and local areas of dacite intrusives. The third episode, at ca. 1.6 Ma, consists of deposition of alkaline rocks including trachyandesites (Milu et al. 2004).

Different volcanic products have been described for the episodes, such as lava or extrusive domes, flows, pyroclastic and volcano-sedimentary deposits, in addition to intrusive bodies (dykes, domes, and micro-laccolithes) that partly represent the rooted area of the volcanoes, and a large variety of intrusive breccias (Borcos and Vlad, 1994; Tamas, 2002). The volcanoes have been described as calderas, stratovolcanoes, simple or composite volcanoes, and extrusive domes (Ianovici et al., 1969; Berbeleac, 1975). Generally, the volcanoes preserve less than a third of their superstructure, as they were affected by an intense erosion activity such as in the Sacaramb area, but some domes, like Caraci and Cetras, have retained an original morphology (Borcos in Cioflica et al., 1973).

The first two volcanic-intrusive episodes are associated with the majority of the metallogenic activity. The first, mid-Miocene episode resulted in Au-Ag epithermal mineralisation, such as that of the Rosia Montana deposit (Manske and Hedenquist, 2006). The second, mid- to late Miocene episode is represented by Au-Ag (Te) epithermal mineralisation (e.g. Sacaramb, Stanija, Baia de Aries), Pb-Zn-Cu (Au-Ag) mineralisation (e.g. Troiuta, Coranda, Hanes) and porphyry Cu (Au-Mo) mineralisation (e.g. Rosia Poieni, Deva, Bolcana, and presumably Colnic and Rovina; Milu et al., 2004).

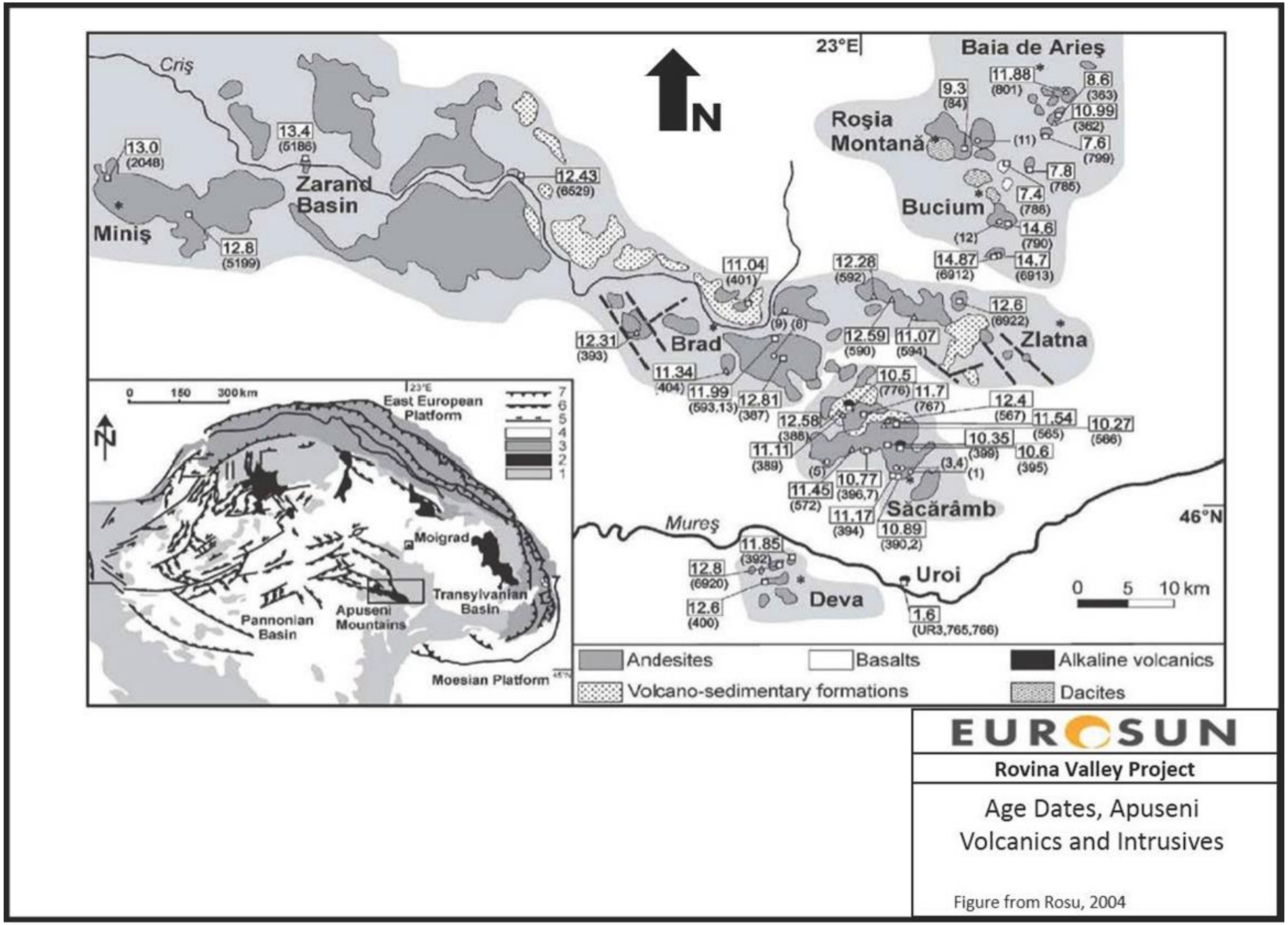


Figure 7.2: Age Dates, Apuseni Mountains Volcanism and Intrusive Activity

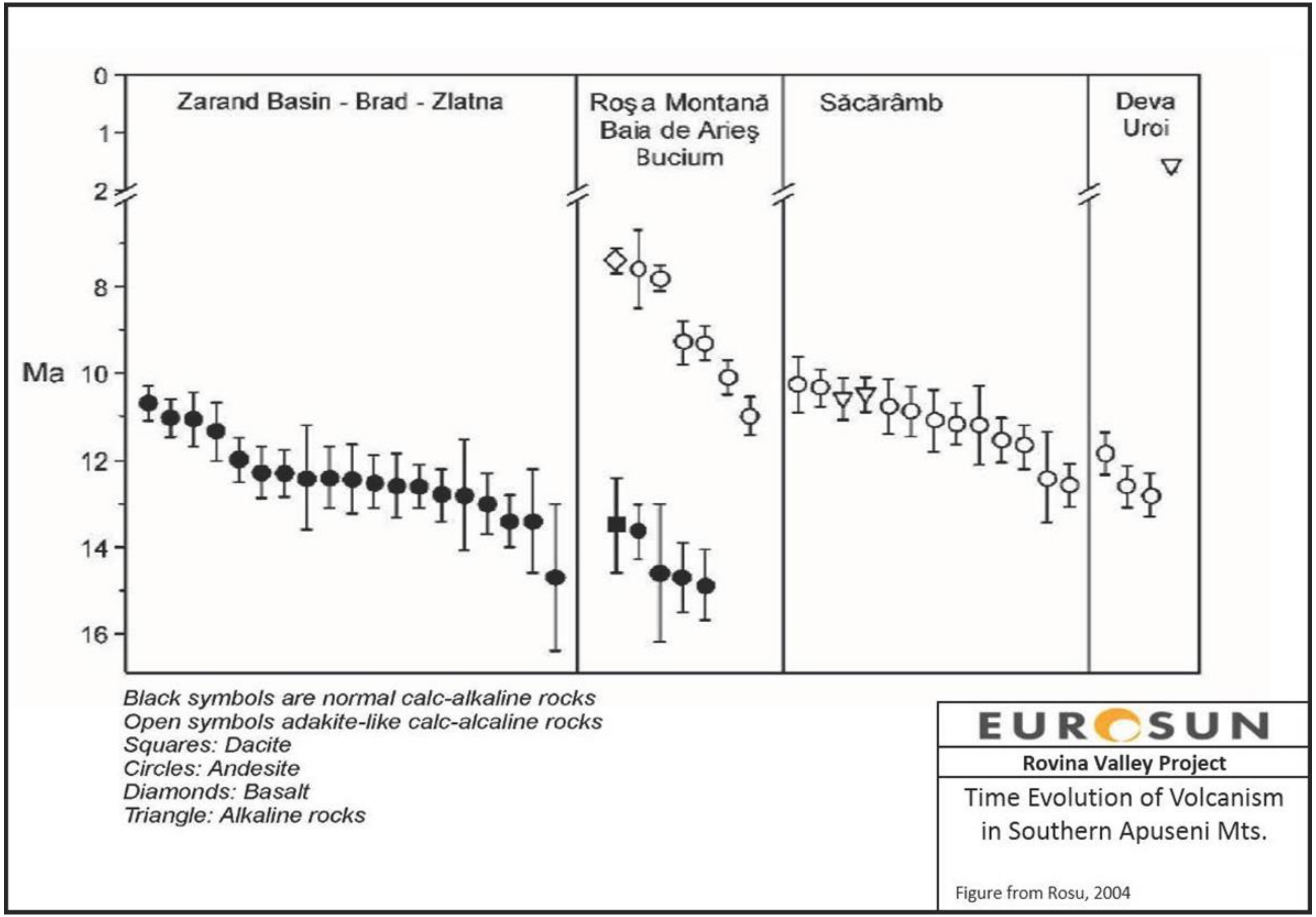


Figure 7.3: Time Evolution of Volcanic Activity in the Southern Apuseni Mountains

7.2 ROVINA VALLEY PROJECT SETTING

The Rovina property occurs within the defined Golden Quadrilateral Mining District located just east of the Brad-Barza sub-district, and near the northern end of the Sacarimb-Brad volcanic belt (see Figure 7.4). The Property covers a sequence of Neogene-aged subvolcanic intermediate intrusive rocks, which in other parts of the Golden Quadrilateral host epithermal- and porphyry-style mineralisation. A variety of mineral deposit types are present in the Golden Quadrilateral area, including porphyry copper, epithermal veins (low-sulphidation, and less commonly high-sulphidation), breccia pipes, and replacement bodies. ESM's exploration programmes have identified gold-rich porphyry systems (Colnic and Ciresata deposits) and a copper-gold porphyry (Rovina) associated with Neogene subvolcanic intrusive complexes.

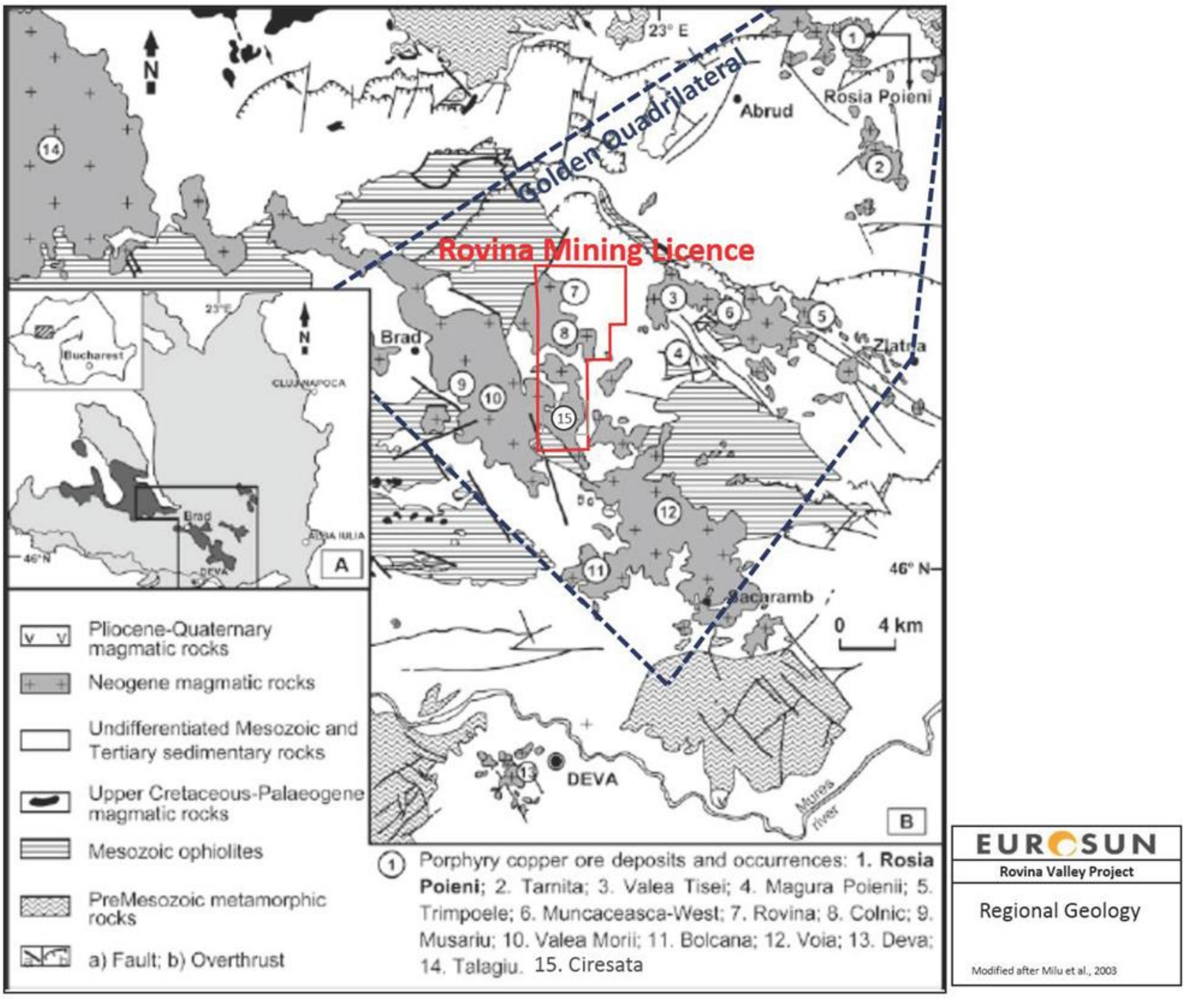


Figure 7.4: Regional Geology

The Rovina and Colnic porphyry deposits lie within a north-eastern volcanic outlier of the 8 km to 10 km diameter, Neogene-aged, Brad-Barza volcanic field. The Brad-Barza volcanic field is well known for hosting high-grade gold veins with an estimated historical gold production of 11.2 Moz, dating back to the Roman period (ca. 2,000 years ago) (Vlad and

Orlandea, 2004). The Ciresata porphyry, 4.5 km south of Colnic, lies within the eastern part of the Brad-Barza volcanic field.

The basement stratigraphy and volcanic rocks of the Rovina-Colnic-Ciresata area are regionally similar to those of the Brad-Barza volcanic field (see Figure 7.4 and Figure 7.5). The basal sequence comprises an upper Cretaceous flysch, locally termed the "Strate de Piriul Izvorului". This unit consists of tightly folded siltstones, wackes, and thin interbeds of shale. The regional trend of fold axes is northwest–southeast, with some folds being overturned, resulting in southwest-dipping axial planes (Romania State Geology Map, 1:50,000 scale).

Unconformably overlying this unit are a series of Neogene-aged, intermediate composition, sub horizontal volcaniclastics and flows which have limited aerial extent. Common in the area are isolated bodies of massive plagioclase–amphibole ± pyroxene ± biotite andesites (the "Andesite tip Barza") which are mapped as subvolcanic intrusives (see Figure 7.5).

Lava flows and subvolcanic intrusives from the Rovina-Colnic area cover an arcuate-shaped belt that is approximately 7 km long and 2 km to 3 km wide (see Figure 7.5). The porphyry mineralisation at Rovina and Colnic is hosted by porphyritic hornblende-plagioclase ± quartz ± pyroxene diorites (Damian, 2006, in Ruff, 2006); these are interpreted as subvolcanic intrusives. At Ciresata, porphyry mineralisation is hosted in hornblende-plagioclase ± quartz diorites and hornfelsed Cretaceous sediments (Damian, 2008).

Mapped phyllic alteration halos at Rovina, Colnic, and Ciresata occur in subvolcanic hornblende feldspar porphyries, volcaniclastic units, and locally in Cretaceous sediments.

The Valea Morii Cu-Au porphyry deposit within the Barza magmatic complex, located just outside the Rovina Exploration Licence (see Figure 7.4), has age dates ranging from 11.41 Ma to 11.30 Ma (Kouzmanov et al., 2006). Age dates on proximal barren andesite intrusions cover a wider range, both pre- and post-dating mineralisation, at 12.44 Ma, 11.87 Ma, and 10.95 Ma (Kouzmanov et al., 2006).

Geological mapping coupled with ground magnetic surveys by ESM in the Rovina-Colnic area indicate the presence of late-stage hornblende-feldspar porphyries, with primary magnetite occurring within the alteration halo of the Colnic porphyry system at the Cornetel Peak. These are interpreted to represent post-mineralisation intrusives. Thus, based on present data, mineralisation is bracketed by the basal volcaniclastic unit at Rovina and the post-mineral subvolcanic intrusives at Colnic, and may be of a similar age to the Valea Morii porphyry mineralisation (i.e. about 11.4 Ma).

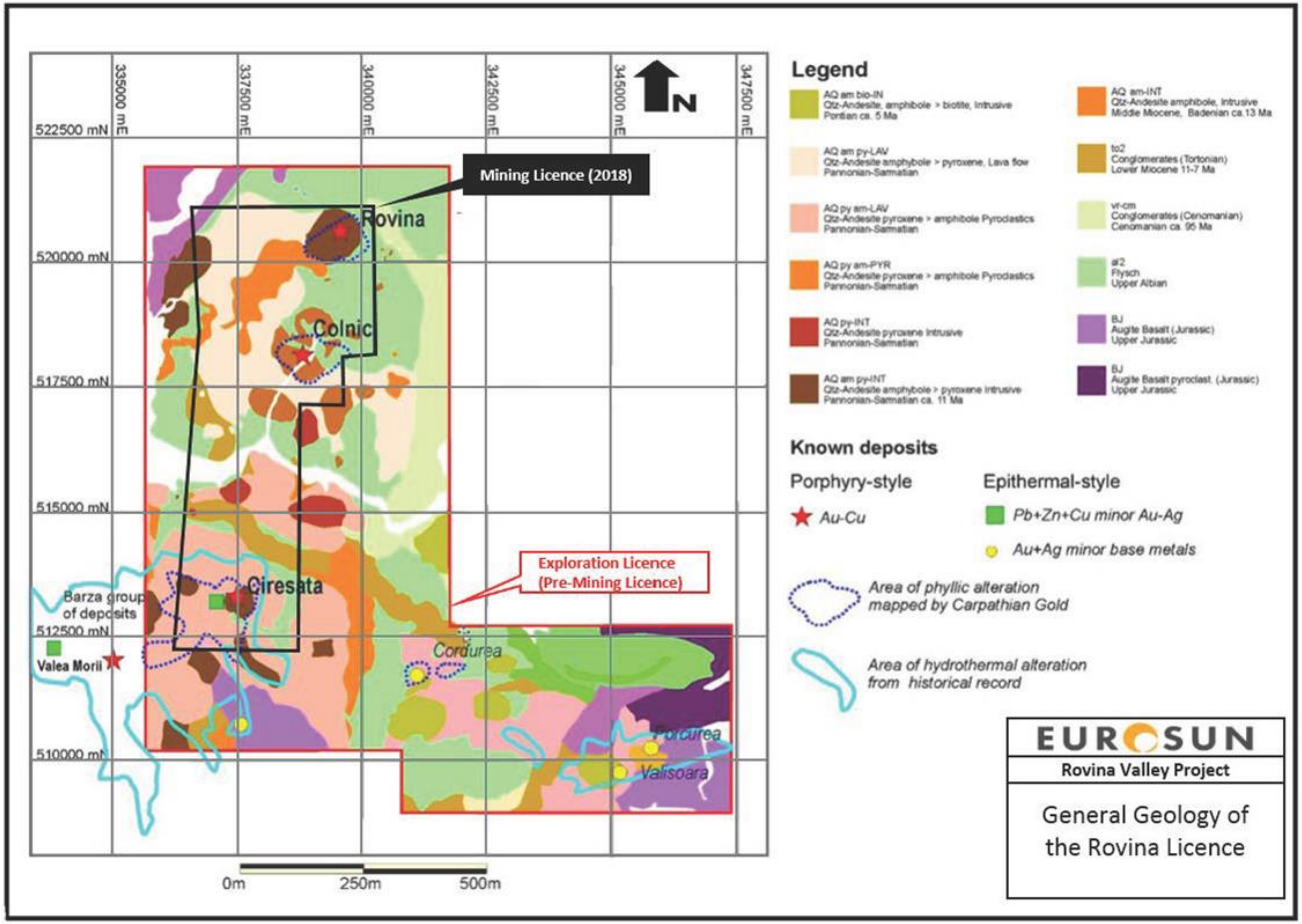


Figure 7.5: Geology of the Rovina Licence

7.3 DEPOSIT GEOLOGY

7.3.1 Introduction

Each of the Rovina, Colnic, and Ciresata porphyries share many basic geologic-mineralisation attributes. These include association with both subvolcanic intrusives of similar composition and similar alteration suites. The mineralised porphyries at Rovina, Colnic, and Ciresata display moderate to intense potassic hydrothermal altered cores, and strong quartz stockwork veining. The Au-Cu mineralisation manifests as stockwork veining and disseminations of pyrite and chalcopyrite, centred on porphyritic, subvolcanic-intrusive complexes of hornblende-plagioclase diorites. The Colnic and Ciresata porphyries classify as gold-rich, with the Rovina porphyry falling within the Cu-Au subtype. All three porphyries contain many of the features common in gold-rich porphyries [i.e. dioritic, calc-alkaline stock associated and abundant magnetite alteration (Sillitoe, 2000)].

Geometry of the mineralisation and host porphyries is different for each of the deposits. At Rovina, the host porphyries are generally cylindrical and vertical. At Colnic, the porphyries are lobate, with mineralisation decreasing with depth and a phyllic-altered cap locally preserved. Both Rovina and Colnic porphyries intrude extensive igneous-magmatic breccia carapaces, whereas Ciresata mineralisation is centred on a relatively narrow subvolcanic “neck” with a significant amount of mineralisation hosted in adjacent hornfelsed sediments.

No significant porphyry-related epithermal or skarn mineralisation has been identified, with only minor occurrences of gold ± silver, lead, and zinc in narrow epithermal veins within the phyllic alteration halo at Colnic. In addition, no significant weathering oxidation of hypogene sulphides is observed. Detailed description of the logged lithologies, alteration, and mineralisation for each deposit is described in the following sections.

7.3.2 Rovina Deposit Geology

7.3.2.1 Rovina Geology Summary

Copper-gold mineralisation at Rovina is hosted in multiple composite plagioclase-hornblende porphyritic subvolcanic intrusives. This mineralisation reaches the surface and is exposed in one location as outcrops in the Baroc valley drainage over approximately 300 m (see Figure 7.6). The remaining sparse and scattered outcrops are phyllic-altered fragmental volcanics and porphyritic volcanics, which comprise a mapped phyllic alteration halo of 1,000 m × 600 m. The mineralised porphyries are cylindrical and vertical, with mineralisation extending up to 600 m below surface. At least three mineralised porphyries are recognised. The main porphyry (Rovina porphyry) intrudes (or is surrounded by) a brecciated porphyritic unit. This breccia unit is locally mineralised and is interpreted as an intrusive magmatic breccia (IMB) carapace to an upper-level intrusive. The last, post-mineral stage of intrusive activity is the emplacement of a phreatomagmatic breccia complex, which cuts earlier porphyry units and is grade destructive. Surface geology (see Figure 7.6) and two cross-sections through the Rovina porphyry (see Figure 7.7 and see Figure 7.8) are shown below. Plan map locations for the cross-sections are shown in Figure 7.6.

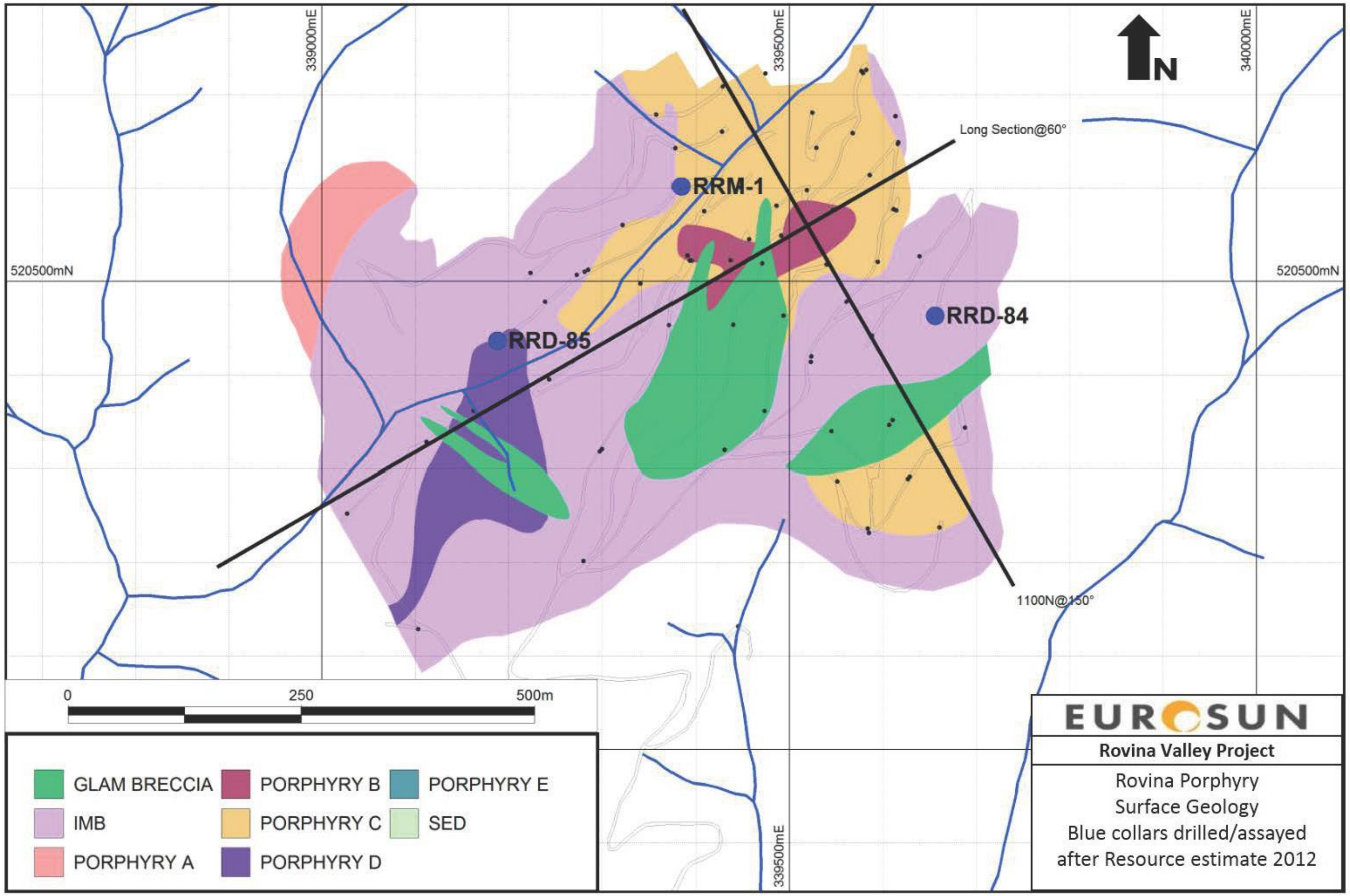


Figure 7.6: Rovina Porphyry Surface Geology

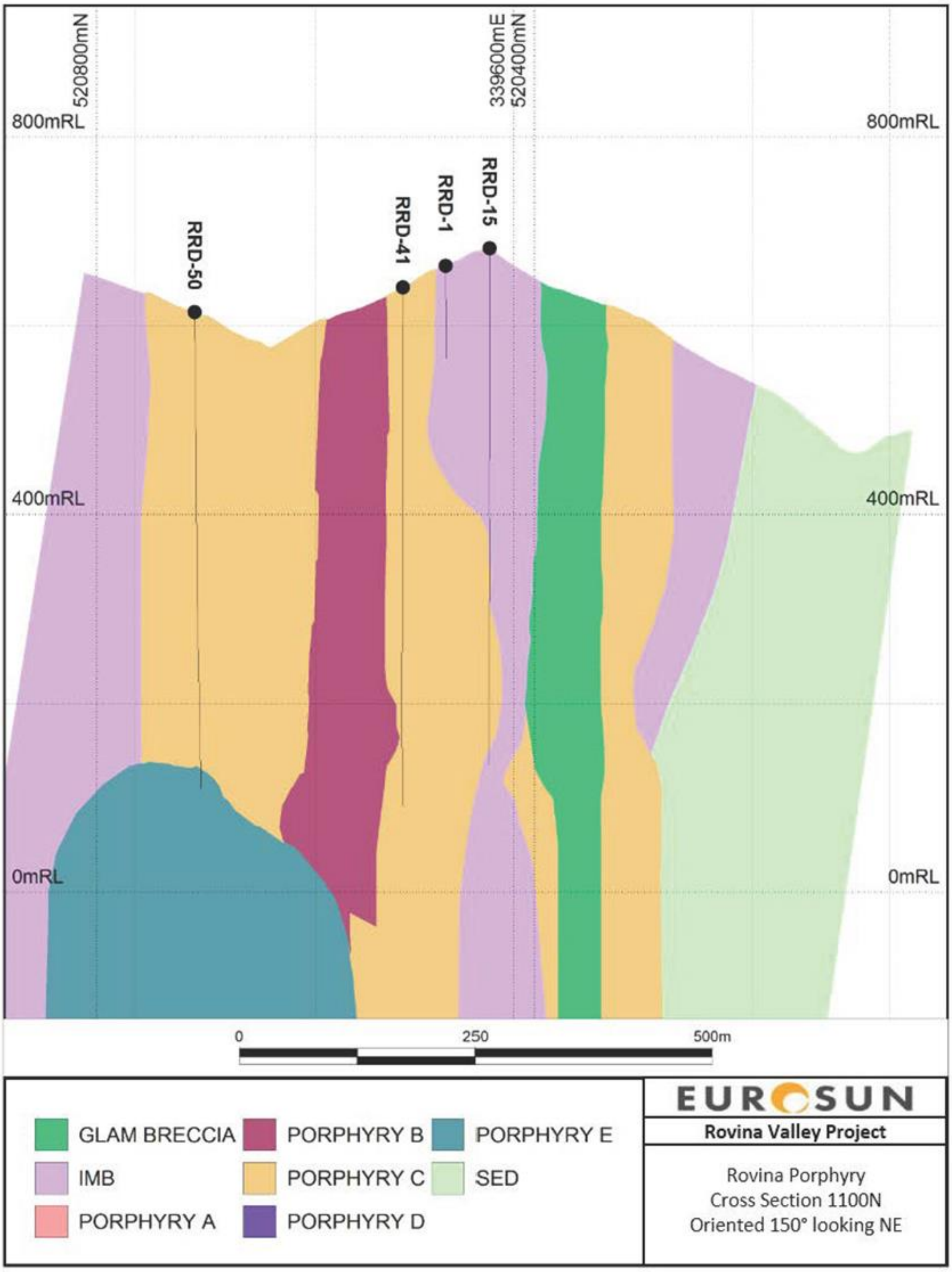


Figure 7.7: Cross-Section of Rovina Porphyry with Major Lithologic Units

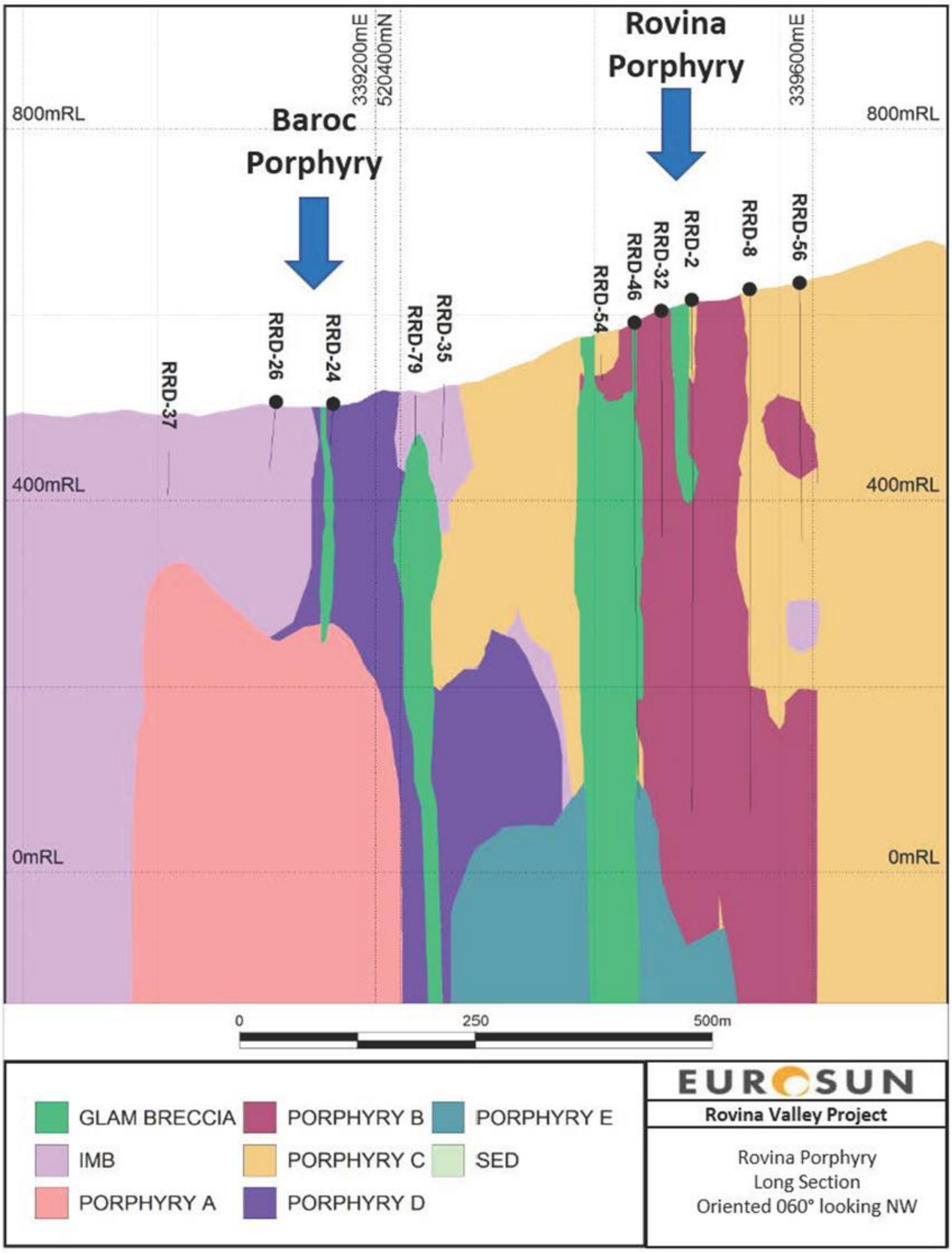


Figure 7.8: Long Section of Rovina Porphyry with Major Lithologic Units

Multiple inter-mineral intrusive phases into the Rovina porphyry have been recognised with features such as breccia clasts of intense stockworked porphyry within later, less intense stockwork porphyry. An example is Porphyry B intruding the axial position of the Rovina porphyry, as shown in Figure 7.9, clearly illustrating a contact between the younger inter-mineral Porphyry B and the older Porphyry C (Rovina porphyry). Intense stockwork veining in Porphyry C contrasts against the moderate stockwork veining in Porphyry B in the upper section of the core. These multiple events lead to complex and overprinting stockwork veining and alteration features, and likely resulted in grade enhancements.

Figure 7.9: Photograph of Drill Core from RRD-45 Contact between Porphyry C and Porphyry B

Recognised alteration types associated with mineralisation include an early potassic (biotite ± K-spar), magnetite (several events), and magnetite-propylitic (termed MACE, with magnetite, chlorite, and epidote). The MACE alteration often overprints the earlier potassic alteration. Phyllic alteration occurs around the margins of the mineralised porphyries.

Petrographic description samples from the Rovina porphyry indicate that a variably potassic and magnetite-altered plagioclase-hornblende porphyry is host to disseminated and veinlet-controlled chalcopyrite (see Table 7.1).

Table 7.1: Petrographic Description of Mineralised Rovina Porphyry

Sample	Petrographic Summary Description
RRD-9-1 (DH RRD-9 at 384.57 m)	Altered plagioclase-amphibole porphyry (dioritic?), with disseminated and fracture-controlled chalcopyrite mineralisation associated mainly with potassic alteration (biotite-K-feldspar); early quartz-amphibole-magnetite±carbonate-chalcopyrite-pyrite veinlets are cross-cut by later quartz-chalcopyrite-carbonate-magnetite-pyrite-Au veinlets.
RRD-13-2 (DH RRD-13 at 184.78 m)	Altered plagioclase-amphibole porphyry (dioritic?), with disseminated and fracture-controlled chalcopyrite mineralisation associated mainly with potassic alteration (K-feldspar-biotite); magnetite-rich fractures are common, with the assemblage quartz-magnetite-calcite±amphibole-K-feldspar-chalcopyrite.
RRD-18-3 (DH RRD-18 at 165.76 m)	Altered plagioclase-amphibole porphyry (dioritic?), with disseminated chalcopyrite mineralisation associated mainly with potassic alteration (biotite±K-feldspar); overprinted by minor quartz-magnetite-biotite-plagioclase-scapolite(?)-chalcopyrite-pyrite veinlets and later quartz-magnetite±pyrite stringers.

7.3.2.2 Rovina Lithology and Logging Unit Descriptions

The detailed logging protocol of ESM has differentiated the major lithologic units into subunits, based primarily on textural features and on cross-cutting mineralisation features in the case of inter-mineral porphyries. Five different porphyries have been interpreted at Rovina and a list of major lithologic units and logging codes is shown in Table 7.2.

Table 7.2: Major Lithologic Units and Logging Codes for Rovina

	Code	Name	Description/Location
Rovina composite hornblende-plagioclase porphyries	64	Rovina Por. (POC: early-mineral)	Principal mineralised porphyry
	65	Inter-mineral Por. (POB: inter-mineral)	Inter-mineral axial intrusion to POC
	63	Baroc Valley Por. (POD: late-inter-mineral)	Satellite porphyry body
	62	Potassic-silicic Por. (POE: late-mineral)	Deep occurrence
	66	Northwest Por. (POA: late-mineral)	Minor importance
	69	Breccias BX undifferentiated	Minor importance, localised orthomagmatic breccias within Por. units
Rovina Wall Rocks	67	IMB Complex	Heterogenous fragmental hornblende-feldspar Por. with similar composition groundmass; predominantly wall rock, locally mineralised
	68	Flysch sediment (SED)	Folded Cretaceous sediments, localised occurrence in southeast, unmineralised
Rovina post-mineral units	61	Glamm Breccia (GBX)	Green rock-flour matrix with polylithic clasts locally derived; phreatomagmatic breccia complex with several facies; grade destructive.

Overburden (OB – 600): The completely oxidised colluvium, alluvium, residual soil, and weathered rock occurring at the top of most drillholes.

Glamm Breccia (GBX – 610): The glamm breccia (phreatomagmatic breccia) tends to have distinct, sharp cross-cutting contacts with the surrounding wall rocks. This unit is generally polylithic, containing sand to cobble scale, rounded to angular clasts of mineralised and barren porphyry, previously formed breccia phases, and vein fragments. This unit appears to have formed post-mineral, and aside from minor locally present carbonate ± base metal mineralisation, it has not been cross-cut by mineralised quartz or sulphide veins or overprinted by earlier potassic or MACE alteration. Most of the gold and copper mineralisation occurring within the glamm breccia is restricted to clasts of strongly veined porphyritic wall rock that have been incorporated into this unit close to the margins of the breccia pipe. The matrix of the glamm breccia is generally chlorite-clay enriched, giving the unit a characteristic pistachio green to light grey (bleached) colour. The margins of clasts have also been affected by this alteration, resulting in bleached rims. Four texturally distinct subcategories of glamm breccia have been recognised in the drill core, described below.

Crystal Rich Glamm (XGBX – 611): This rock type was previously logged as a late (post-mineral) fine- to medium-grained feldspar-amphibole porphyry because of its homogeneous, porphyritic appearance. It has been interpreted here as an early, crystal-rich phase of the glamm breccia unit, due to its ubiquitous occurrence within this unit, similarities in the style and intensity of chlorite-clay alteration, and complete lack of mineralised veining and potassic/MACE alteration. Crystal-rich glamm generally contains 5–7 modal per cent 2–3 mm feldspar, and 5–7 modal per cent 2–3 mm mafic phenocrysts, within a much finer grained chlorite-clay enriched matrix. This unit is generally cross-cut by thin glamm breccia shoots, and clasts of crystal-rich glamm are common within younger glamm breccia phases. Locally, a weakly developed lamination has been observed that may indicate a genetic relationship with the more strongly banded laminated glamm breccia described below. This rock type is generally unmineralised.

Laminated Glamm (LGBX – 612): This rock type is characterised by a pervasive flow banding consisting of sub-parallel, commonly irregular laminations defined by composition and grain size variation. Previously, laminated glamm breccia was included as part of the matrix-rich glamm. It has been distinguished here due to its porphyritic appearance, which resembles the crystal-rich glamm described above. This unit may also be cross-cut by matrix- or clast-rich glamm breccia shoots, and clasts of this rock type are common within younger glamm breccia phases. This rock type is generally unmineralised.

Clast-Rich Glamm (CGBX – 613): This rock type is generally poorly sorted, containing angular to sub-rounded clasts of strongly altered/mineralised/veined porphyry, barren porphyry, previously formed breccia phases, and vein fragments, within a pistachio green to light grey (bleached) chlorite-clay enriched matrix. Close to the margins of the breccia pipe, clast-rich glamm breccia may contain significant copper/gold mineralisation within clasts of mineralised wall rock that has been incorporated into the breccia unit. Where mineralised clasts do not occur, clast-rich glamm breccias are generally unmineralised. Clast-dominated glamm breccias grade into, and are interfingered with, matrix-dominated breccias, suggesting that these two rock types may be related. The Glamm breccias possibly represent the marginal facies to matrix-dominated breccia shoots.

Matrix-Rich Glamm (MGBX – 614): The matrix-dominated glamm breccia is finer grained, better sorted, and more comminuted than the clast-rich glamms described above. The rock is generally macroscopically structureless and has a pistachio green to light grey (bleached) colour due to pervasive chlorite-clay enrichment. Where mineralised clasts do not occur, this rock type is generally unmineralised.

Amphibole-Feldspar Porphyry (Porphyritic Diorite – 620, 630, 640, 650, and 660): The majority of the veining, mineralisation, and alteration in the Rovina deposit are hosted by a series of medium- to coarse-grained amphibole-feldspar porphyry stocks. These units generally contain approximately 7 to 10 modal per cent 3 to 12 mm black green euhedral prismatic hornblende porphyrocrysts, and 15 to 20 modal per cent 1 to 4 mm white/colourless subhedral-blocky prismatic, zoned plagioclase phenocrysts within fine, granular to aphanitic groundmass. The various porphyry stocks at Rovina (POA, POB, POC, POD, and POE) were identified as separate intrusive phases using a combination of observations, including the alignment of phenocrysts adjacent to intrusive margins, truncation of quartz veins within older porphyry phases cross-cut by younger ones, and the incorporation of quartz vein fragments and quartz veined xenoliths within younger porphyry phases that have been emplaced into older, previously veined phases. Decrease in phenocryst size and sudden changes in grade and alteration type were also considered when distinguishing between earlier and later porphyry intrusions. Within each porphyry unit, finer grained (621, 631, 641, 651, and 661) and coarser grained (622, 632, 642, 652, and 662) end-members were identified during logging, as well as intervals displaying magmatic foliation or lamination (623, 633, 643, 653, and 663).

In total, five amphibole-feldspar porphyry stocks were identified at Rovina, namely (from youngest to oldest):

1 Porphyry A (660) – Northwest (Baroc Valley) Porphyry (late mineral)
2 Porphyry E (620) – Potassic-Silicic Porphyry (late mineral)
3 Porphyry D (630) – Baroc Valley Porphyry (late inter-mineral)
4 Porphyry B (650) – Inter-Mineral Porphyry (inter-mineral)
5 Porphyry C (640) – Rovina Porphyry (early mineral)

Intrusive Magmatic Breccia (IMB – 670): The intrusive magmatic breccia is a pre-mineral clastic unit that appears to form the host rocks to the mineralised porphyry stocks at Rovina. This unit generally has a complex texture and contains sub-angular to rounded igneous and sedimentary rock fragments within a fine-grained, granular to porphyritic matrix. This unit is affected by early potassic and MACE alteration and is cross-cut by mineralised quartz and sulphide veining. Where this unit is strongly altered, margins of clasts are poorly defined, giving the rock a clotty/spotted appearance. Intervals displaying well-developed magmatic foliation are common within this unit, suggesting that it may be part of a pre-mineral diatreme breccia or subvolcanic intrusion that subsequently became the host to the mineralised porphyry stocks at Rovina. In addition to the rock type described above, two texturally distinct subcategories of intrusive magmatic breccia have been recognised in the drill core.

Laminated Intrusive Magmatic Breccia (LIMB – 671): This unit consists of restricted intervals of moderately to strongly laminated intrusive magmatic breccia. Lamination is generally defined by clotty/discontinuous sub-parallel layers of magnetite-amphibole-chlorite ±

sulphide alteration within a much finer grained matrix. These zones generally grade into and out of more typical intrusive magmatic breccia.

Xenolith Rich Porphyry (XIMB – 672): Intervals of clast-poor intrusive magmatic breccia with a spotty/porphyritic texture also occurs locally. These intervals generally grade into and out of more typical intrusive magmatic breccia.

Flysch Sediment (SED – 680): A locally laminated unit of very fine-grained mudstone to fine-grained sandstone was intersected to the south of the Rovina porphyry in RRD-38. This rock type appears to host the intrusive magmatic breccia unit described above. Adjacent to the contact with the intrusive magmatic breccia, this unit may be strongly fractured and biotite ± sericite/clay-enriched, becoming dark brown to pale grey/beige in colour. Fracture controlled pyrite ± silica may also occur within altered intervals.

Breccia (BX – 690): Poorly defined intervals of early to syn-mineral (possibly orthomagmatic) brecciation occur within all of the described porphyry phases at Rovina. These breccias included zones of monolithic (692) wall rock breccia and polylithic (693) brecciation that may have formed in response to magmatic fluid discharge from inter-mineral or late mineral porphyry phases. Intrusive breccias that appear to form a sheath around inter-mineral and late-mineral porphyry stocks have also been observed. In general, these units are affected by similar alteration and stockwork intensity as the surrounding rocks.

Post Mineral Dyke (LD – 700): Weakly altered, unmineralised late intrusions (dykes) have also been identified locally within the Rovina deposit. However, overall, post-mineral dykes are not common.

7.3.2.3 Rovina Alteration and Logging Unit Descriptions

Several alteration types are recognised with mineralisation mainly associated with the early potassic and locally overprinting MACE alteration. A list of alteration types and logging codes is shown in Table 7.3 followed by the description of the logged alteration style.

Table 7.3: Major Alteration Units and Logging Codes for Rovina

Alteration Style	Logging Code	Description
Potassic	PT (31)	Widespread “biotisation” of groundmass, magnetite stringers, minor microscopic K-spar replacements
Magnetite-propylitic	MACE (30)	Common magnetite dissemination and in stringers, quartz, chlorite, epidote, amphibole, and carbonate
Potassic-silicic (bio-K-spar-sil)	PTSI (3139)	At depth, intense “biotisation” of groundmass with silicification, minor microscopic K-spar replacements
Silicic (sil)	SIL (39)	Localised silicification
Phyllic	PH (33)	Broad wall rock halo of sericite, pyrite, quartz
Transitional phyllic	TRPH (43)	Phyllic overprint of earlier, mostly potassic and MACE alteration; not prevalent
Argillic	A (36)	White clays and pyrite typically associated with late fracture zones
Carbonate	CC (37)	Calcite and clays associated with rare late-stage galena-sphalerite breccia fills
Propylitic	PR (42)	Rare chlorite, carbonate, pyrite in late dykes
Propylitic w/in glamm breccia	GM (35)	Pervasive in rock-flour matrix, chlorite, carbonate, pyrite
Weathering oxidation	Fe-Ox (41)	Localised and limited to upper few to tens of metres limonite, goethite, and red-brown hematite with white clays

Oxidation (Fe-Ox – 41): Rusty orange-brown limonite, goethite, red-brown hematite, and white clay-enriched oxidation.

Propylitic Alteration (PR – 42): Propylitic alteration within the Rovina deposit is not widespread and is generally restricted to late intrusive dykes and post-mineral breccias (i.e. glamm breccias). Propylitic alteration is generally light green to pistachio green in colour and characterised by a mineral assemblage containing abundant chlorite-carbonate-clay and pyrite. Aside from minor locally present carbonate ± basemetal ± gold-bearing veins, propylitically altered rocks are generally poorly mineralised.

Propylitic Alteration within Glamm Breccia (GM – 35): The matrix of clast-rich and matrix-rich glamm breccias is generally light green to pistachio green in colour and characterised by a propylitic mineral assemblage containing abundant chlorite-clay-carbonate. Propylitic alteration has also resulted in bleaching of the margins of wall rock clasts occurring within this unit.

Clay-Sericite-Pyrite-Quartz-Carbonate Alteration (CSPQ – 44): To simplify the current alteration model, phyllic alteration (PH – 33) and transitional phyllic alteration (TRPH – 43) occurring peripheral to mineralised parts of the Rovina deposit, and less significant structurally controlled argillic alteration (A – 36) and base metal-carbonate alteration (CC – 37) occurring along lithological contacts and within fractured/faulted zones throughout the deposit, were correlated together as clay-sericite-pyrite-quartz carbonate alteration (CSPQ). CSPQ alteration is characterised by a dominant assemblage of clay-sericite-quartz-pyrite that is generally consistent with the low-grade peripheral parts of the Rovina deposit. Approximately 5 % to 15 % pyrite by volume generally occurs within this alteration type as 3 m to 7 mm clusters of fine-grained aggregates, disseminated grains, and within millimetre-scale fractures. CSPQ alteration is typically light grey, greenish grey, to pinkish grey in colour.

MACE Alteration (MACE – 30w, 30m, and 30i): Magnetite-amphibole-chlorite-epidote-rich alteration (MACE), also containing K-feldspar, quartz, titanite, anhydrite, and carbonate, appears to be associated with the highest-grade intervals within the Rovina deposit. It is common for rocks within the Rovina deposit to have been potassic altered prior to becoming overprinted by MACE alteration. Therefore, where MACE alteration is less intense, the rock generally has a brownish-green to pinkish-green colour and a mineral assemblage consisting of biotite-amphibole-chlorite-magnetite-quartz-pink K-feldspar and sulphides (cpy-py). Pink, fine-grained K-feldspar is more common in this alteration type than in purely potassic alteration. Where MACE alteration is pervasive, it is texturally destructive, generally giving the rock a dark green to light green colour. The MACE alteration assemblage is typically associated with magnetite stringers and clots, a well-developed quartz-sulphide vein stockwork, and significant fine-grained disseminated cpy-py-mt mineralisation.

Potassic Alteration (PT – 31w, 31m, and 31i): Potassic alteration is the most widespread alteration type occurring within the Rovina deposit. Potassic alteration is generally associated with a mineral assemblage that includes biotite-magnetite-quartz-pink K-feldspar ± trace green amphibole and chlorite. The alteration minerals occur as finely disseminated grains in the groundmass of the rock and as granular aggregates of “ratty” biotite-magnetite ± sulphides (cpy-py). Very fine, irregular stringers of magnetite are also associated with this alteration type. The amount of pink-coloured K-feldspar and amphibole/chlorite can vary greatly. Within the Rovina deposit, it is common for potassic alteration to grade into or become overprinted by MACE alteration through a gradual increase of the amount of amphibole, chlorite, and epidote present in the alteration assemblage.

Potassic-Silicic Alteration (PTSI – 3139): Potassic-silicic alteration is characterised by a lack of magnetite and an intense reddish-brown to pale pinkish-brown colour, which distinguishes it from the darker brown colour of potassic alteration. At Rovina, this alteration type is defined by the assemblage of reddish-brown coloured biotite-pink K-feldspar-quartz-epidote-chlorite and sulphides (py-po-mo-cpy). Sulphides occur as fine disseminated grains within the groundmass and within quartz-sulphide veins. The potassic-silicic alteration at Rovina is similar to the deep-seated Colnic potassic core and is characteristically of a low grade.

Silicic Alteration (SIL – 39): Silicification is characterised by pervasive bleaching and quartz enrichment ± minor epidote-chlorite-clay ± pyrite. Silicification generally gives the rock a light grey colour and increased hardness. Pyrite occurs as fine disseminated grains and grain aggregates within the groundmass and within quartz veins. In some places, this alteration type appears to be related to the potassic-silicic alteration described above.

7.3.2.4 Rovina Discussion

Within the Rovina deposit, it is common to observe more than one overprinting alteration phase within a given interval of drill core. In the drill logs, the mineral assemblage and intensity of each alteration type has been recorded, highlighting the dominant alteration type within each logged interval. The dominant alteration type was then assigned to one of 18 logging codes, which were used to create the current alteration model. Considering this, small and/or isolated intervals of alteration occurring within an otherwise homogeneous zone of a different dominant alteration type were not broken out as separate units during interpretation.

To simplify the current alteration model, argillic, phyllic, transitional phyllic, and carbonate-clay alteration were grouped together as clay-sericite-pyrite-quartz alteration (CSPQ). All these alteration types appear to have formed late, locally overprinting earlier alteration phases. Aside from localised copper and gold mineralisation occurring within some carbonate-base metal veins, these alteration types are not associated with significant mineralisation.

The initial intrusion of the Rovina porphyry (POC) into the IMB was followed by widespread potassic alteration within the Rovina porphyry and pervasive MACE alteration within the surrounding breccia package. Peripheral to the Rovina porphyry, MACE and potassic alteration were replaced and locally overprinted by phyllic alteration, forming an extensive quartz-sericite-pyrite halo.

Intrusion of the inter-mineral porphyry (POB) into the Rovina porphyry (POC) was followed by weak MACE alteration within the inter-mineral porphyry and more intense overprinting MACE alteration within the hosting Rovina porphyry. MACE alteration appears to be most strongly developed within the Rovina porphyry adjacent to the northern margin of the inter-mineral porphyry. In this strongly altered zone, overprinting MACE alteration is texturally destructive, and pre-existing potassic alteration minerals are rarely preserved. Further from the intrusive centre, overprinting MACE alteration decreases in intensity, eventually becoming secondary to the underlying potassic alteration.

Intrusion of the potassic-silicic porphyry (POE) was followed by pervasive potassic-silicic alteration within this intrusion. However, the potassic-silicic alteration occurring within POE appears to have had little visible effect on the surrounding wall rocks.

At Rovina, all intrusive rock types have been cross-cut by post-mineral glamm breccias. Subsequent to their formation, the glamm breccia dykes appear to have focused late, low-temperature fluids, resulting in pervasive propylitic alteration of crystal-rich and laminated end-members. Propylitic alteration within the glamm breccia has also affected the matrix to clast-rich and matrix-rich glamms and the margins of wall rock clasts.

Tectonic shattering and brecciation adjacent to intrusive contacts and glamm breccia shoots is often associated with structurally-controlled argillic alteration and/or minor carbonate-base metal veining. Late, low-temperature argillic alteration has also affected the upper parts of glamm breccia dykes and the intrusive magmatic breccia occurring to the south of the Rovina porphyry.

In the southwest, the Baroc Valley porphyry (POD) is associated with a MACE-altered core, grading into potassic alteration closer to the porphyry's margins. The Northwest (Baroc Valley) porphyry (POA) is associated with a very weak, propylitic to unaltered core, grading into potassic and potassic-silicic alteration closer to its margins. Overprinting potassic-silicic alteration and silicification occurs within the Baroc Valley porphyry and the intrusive magmatic breccia package adjacent to the eastern and southern margins of the POA.

The upper parts and margins of the Baroc Valley porphyry and the Northwest (Baroc Valley) porphyry have been overprinted by retrograde phyllic and/or argillic alteration, which becomes more extensive to the southwest. Intrusive magmatic breccia occurring in this area has also been pervasively clay-sericite-pyrite-quartz enriched.

Alteration styles for the Rovina deposit are listed in Table 7.3, with inferred paragenesis and timing of the alteration assemblages shown in Table 7.4.

7.3.2.5 Rovina Mineralisation Descriptions

Gold-copper mineralisation is associated with pyrite-chalcopyrite-magnetite occurring in veinlet stockworks and as finely disseminated grains. Oxidation is restricted to the uppermost few metres, except for a small area in Baroc Valley at the Rovina porphyry where weathering oxidation is 15 m to 25 m deep within the copper-gold mineralisation. In this area, secondary copper minerals malachite and chrysocolla are observed in the weathering zone, and minor occurrences of supergene copper minerals (chalcocite) occur below the weathering zone, typically associated with short drillhole intervals of elevated copper grades.

Deposit-scale controls to mineralisation are the localisation of the principal hornblende-plagioclase porphyry intrusion (Rovina porphyry POC), which is elongated in a north-westerly direction, measuring approximately 600 m northwest × 350 m northeast. This porphyry has vertical contacts over at least 600 m in depth, and apparently terminates northward in the northeast trending Baroc Valley zone. Lower-grade copper-gold mineralisation extends down the Baroc Valley zone to the southwest, to include the Baroc Valley porphyry as a satellite to the main Rovina porphyry. This intrusive geometry suggests possible northwest structural control for emplacement of the Rovina porphyry intersecting a northeast structural zone controlling emplacement of the Baroc Valley Porphyry; see Table

7.5 for the geology codes and Figure 9.2 for the Baroc Valley grade distribution. A similarly oriented structural intersection is interpreted for the Colnic deposit.

At Rovina, two early-stage magmatic-fluid alteration events are recognised (PT, MACE, and a locally occurring magnetite-only alteration). Higher grades of gold-copper mineralisation are best developed and associated with broad zones of intense quartz-sulphide stockwork veining (up to 70 % of rock mass). Stockwork veining intensity typically correlates with alteration intensity, and in higher-grade zones, such as in the Baroc Valley area, intense stockwork veining with overprinting MACE alteration obscures all primary rock textures. The earliest copper-bearing assemblage is observed in both early magnetite-bearing veinlets/stringers and disseminated in the rock mass and consists of magnetite + chalcopyrite + bornite + minor pyrite. Cross-cutting veinlets indicated multiple fracturing and hydrothermal pulses. Seventeen vein types have been recognised, with five types most common with gold-copper mineralisation (see Table 7.4). These five vein types are hairline magnetite stringers, quartz veins, quartz-magnetite-sulphide veins, quartz-sulphide veins, and banded quartz-sulphide veins.

Intense MACE alteration is common in the higher-grade zones with a pervasive disseminated mineral assemblage; quartz>>>magnetite>cpy>py> amphibole and/or chlorite ± epidote, ± K-feldspar. Most of the high-grade intervals are hosted in the early porphyry (Rovina porphyry POC), generally coincident along the vertical margins of late inter-mineral stocks (Porphyry B) which themselves are also host to mineralisation, but to a lesser extent (see Figure 7.9).

In comparison with Colnic, stockwork intensity at Rovina more consistently correlates with copper-gold grade and is coincident with intense MACE alteration. In addition, Rovina does not show an intensive phyllic overprint related to late-stage quartz stockwork, as seen on the upper part of the Colnic porphyry.

Molybdenum mineralisation is rarely observed in the drill core within quartz-molybdenite veinlets. Worldwide, other gold and gold-copper porphyries tend to be deficient in molybdenum; however, when present, it tends to concentrate as a halo to the copper-gold core (Sillitoe, 2000). ESM has assayed on a limited basis for molybdenum at Rovina, with assay results to date being insignificant (averaging < 5 ppm to 20 ppm Mo); however, a localised enrichment of molybdenum cannot be ruled out pending further assay checks. Silver has been regularly assayed, and grades to date are generally exceptionally low throughout (averaging <1 ppm to 2 ppm) and do not constitute economic mineralisation.

Table 7.4: Alteration Paragenesis, Rovina

Alteration Phase	Timing - ►	Relative Importance	Sulphide Assemblage	Comment
Potassic (PT)	______	xxxx	cp, py, (mo)	Deposit-wide, pervasive; some minor late-inter-mineral intrusives lack this alteration
Magnetite-Propylitic (MACE)	_______	xxxxxxxx	py, cp (bn)	Deposit-wide, patches
Potassic-Silicic (PTSI)	_____	xxx	py, cp,	Occurs only at depth
Silicification (S)	_________ ________	xx	py	Common in MACE
Transitional Phyllic (TRPH)	_______		py	Minor occurrences
Phyllic (Ph)	______________________________		py	Broad halo, no mineralisation
Propylitic (P)	______		py	Associated with post-mineral phreatomagmatic breccias (glamm breccia)
Carbonate (CC)	---______		py, gal, sph	Associated with minor late-stage epithermal fracture-breccia fill
Argillic (A)	______		py	Minor fracture controlled

cp chalcopyrite
py pyrite
bn bornite
mo molybdenite
sph sphalerite
gal galena
( ) rare

Table 7.5: Vein Types and Logging Codes, Rovina

Vein-Type Code	Relative Importance/Occurrence	Description
mt-st	High	Magnetite stringers
mts-st	High	Magnetite-sulphide-quartz stringers
qs-vn	High	Quartz-sulphide veins
cp-st	High	Chalcopyrite stringers
bq-vn	Medium	Banded quartz-sulphide veins
py-st	Medium	Pyrite stringers
qcs-vn	Medium	Quartz-carbonate-sulphides ± base metals ± barite ± fluorite
dq-vn	Medium-Low	Dark grey quartz vein
q-vn	Medium-Low	Quartz veins
bi-vn	Low	Biotite ± K-feldspar veins
cl-st	Low	Chlorite/amphibole/epidote ± sulphides ± magnetite stringers
kqz-vn	Low	K-feldspar-quartz veins
wq-vn	Low	Irregular early quartz veins, randomly oriented, often discontinuous
po-st	Rare	Pyrrhotite stringers
ca-vn	Low	Carbonate-quartz veins/stringers
tr-st	Low	Tourmaline stringers
anh-vn	Not important	Anhydrite veins
gyp-vn	Not important	Gypsum (after anhydrite) veins
z-vn	Not important	Zeolite (chabazite/stilbite) veins

7.3.3 Colnic Deposit Geology

7.3.3.1 Colnic Geology Summary

Gold-copper mineralisation at Colnic is hosted in multiple composite plagioclase-hornblende porphyritic subvolcanic intrusives. This mineralisation reaches the surface in the Rovina Valley and is exposed in outcrops and road-cuts in the valley bottom for approximately 400 m. The remaining sparse and scattered outcrops are phyllic-altered porphyritic volcanics and Cretaceous sediments, and propylitic-altered hornblende andesites. The Colnic deposit has a large phyllic alteration halo covering 2,000 m × 1,700 m. Two mineralised porphyry-centres comprise the bulk of the Colnic deposit: one occurring in the Rovina Valley (Colnic porphyry) which partially outcrops, and a second centred approximately 200 m southeast on F-2 Hill (F-2 Hill Porphyry). The mineralised porphyries are lobate, with a wider horizontal dimension than vertical extent. These bodies intrude mostly older pre-mineral intrusives, and locally in the northeast Cretaceous flysch sediments. At Colnic, and especially for the F-2 Hill Porphyry, much of the wall rock is a clastic unit of igneous composition with several facies, ranging from flow-laminated silty textures to zenolithic porphyry, which is interpreted to be an IMB complex related to the emplacement of the porphyry as a marginal carapace. Locally, this unit is altered and mineralised close to its contact with the porphyry, which is shallowly east-dipping on F-2 Hill.

The Colnic porphyry is more intensely mineralised than the F-2 Hill porphyry and is complicated by an interpreted series of northeast-striking and subvertical inter-mineral dykes and breccias. These inter-mineral dykes and breccias may have been important for grade enhancements in this area, and the Rovina Valley porphyry is interpreted to be older than the F-2 Hill porphyry. Surface geology (see Figure 7.10) and two cross-sections through the Rovina porphyry (see Figure 7.11 and Figure 7.12) are shown below.

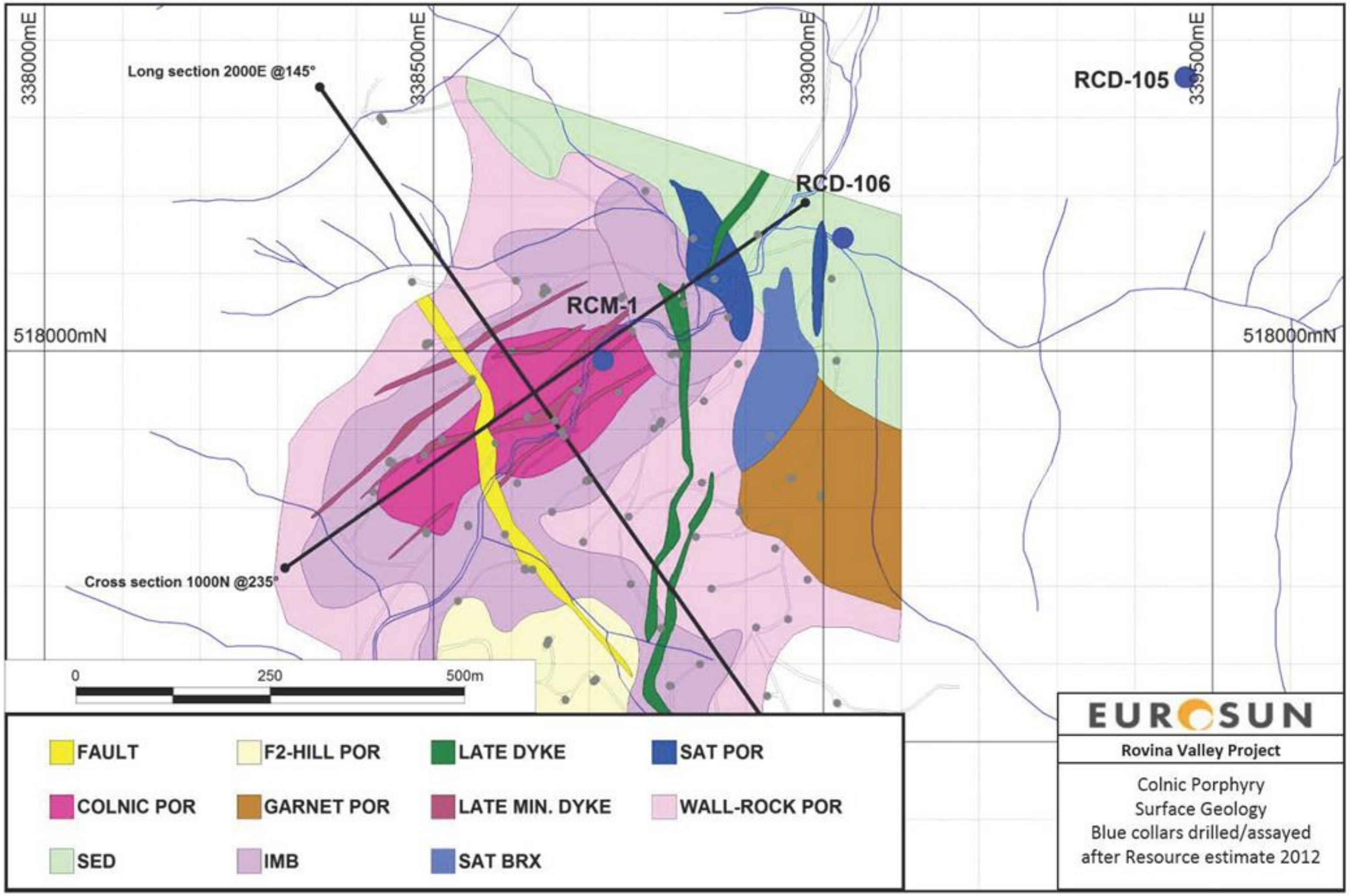


Figure 7.10: Surface Geology Map of the Colnic Porphyry

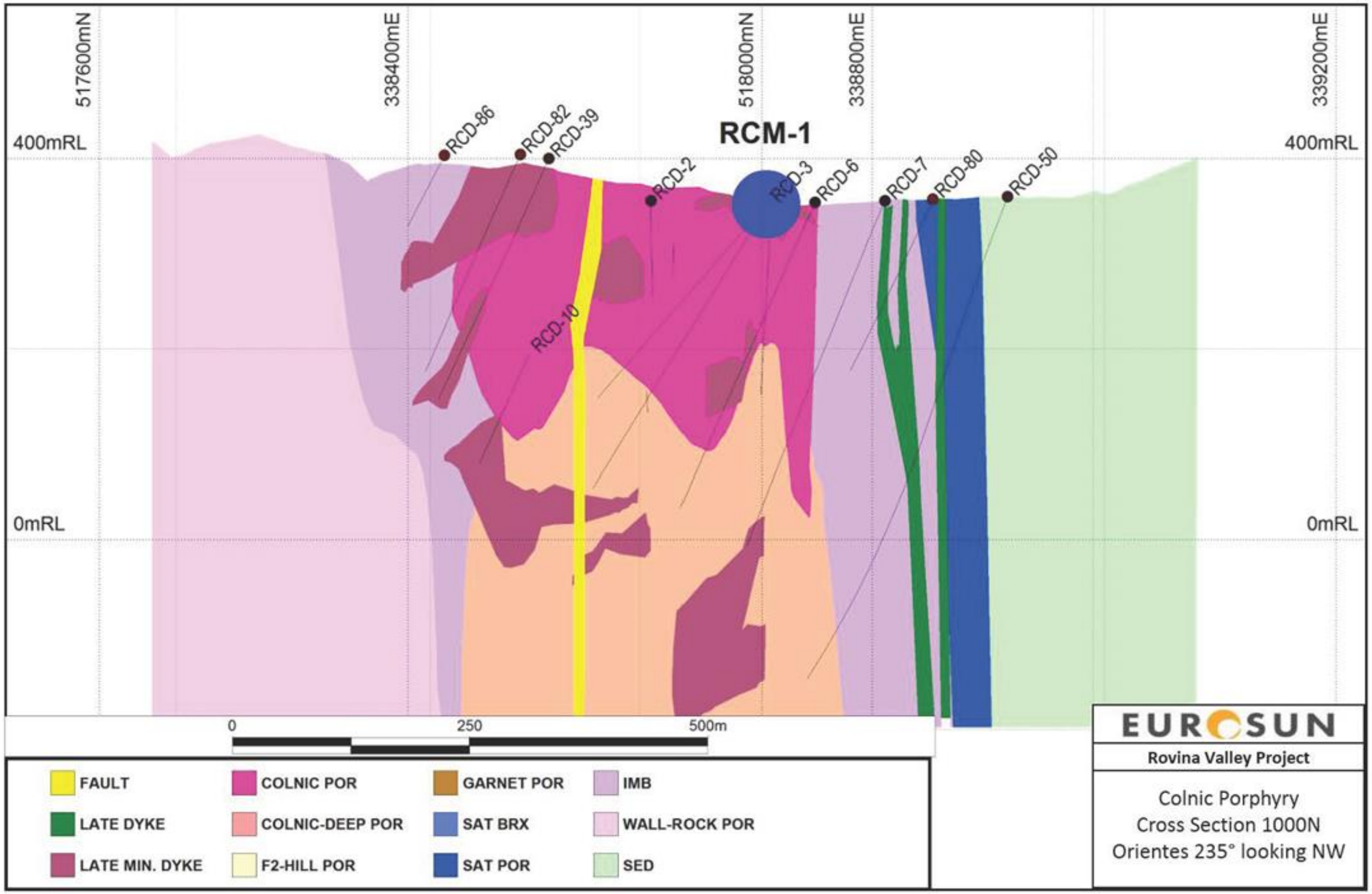


Figure 7.11: Cross-Section through the Colnic Porphyry with Major Lithologic Units

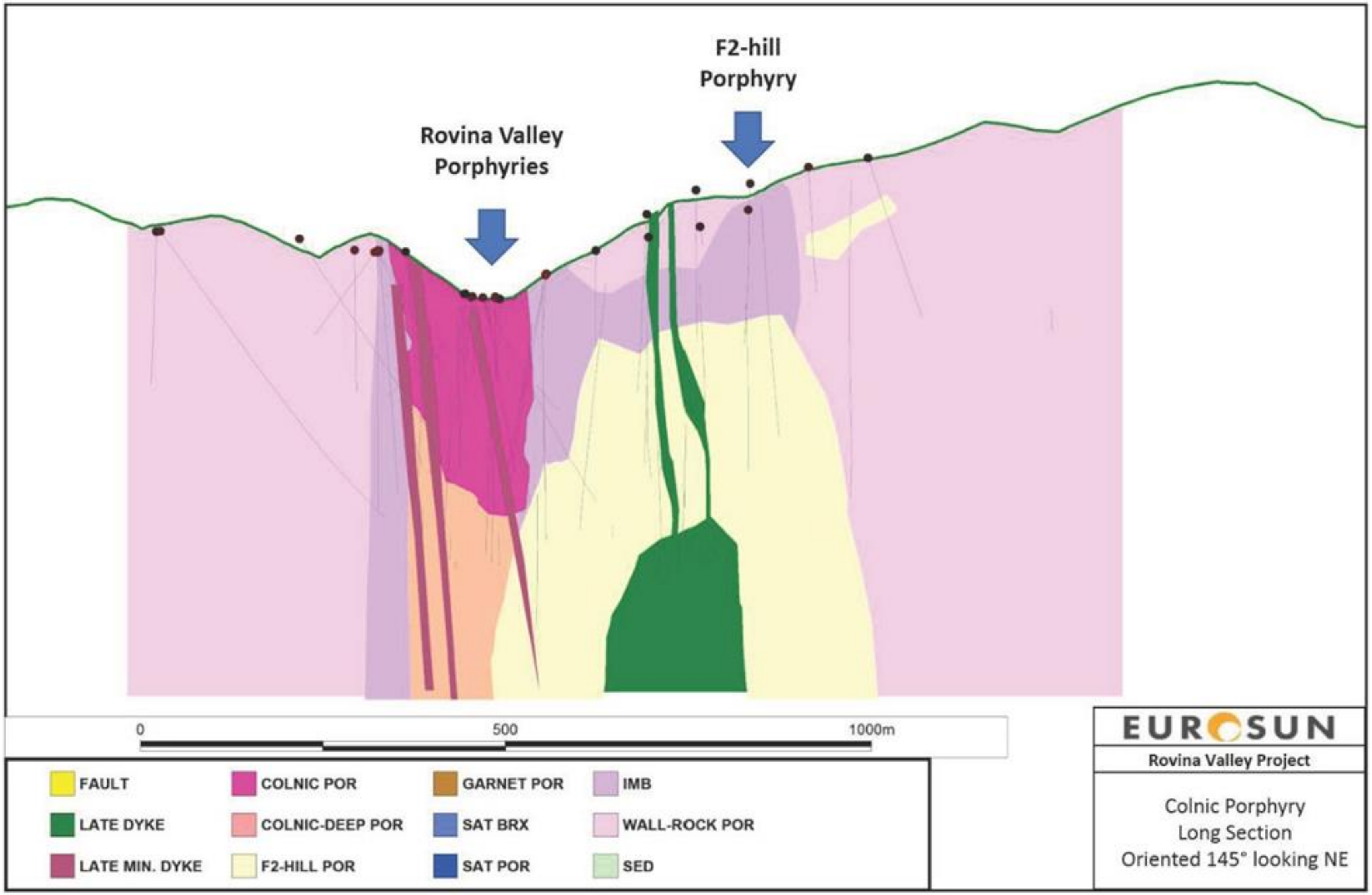


Figure 7.12: Long Section through the Colnic Porphyry with Major Lithologic Units

At Colnic, overprinting alteration is a quite common feature. Phyllic alteration overprint on gold-copper mineralised potassic alteration is prevalent, especially in the Colnic porphyry. Recognised alteration types that are associated with mineralisation include an early potassic (biotite ± K-spar), magnetite (several events), and magnetite-propylitic (or MACE, with magnetite, chlorite, epidote). Phyllic alteration occurs around the margins of the mineralised porphyries.

Petrographic description of samples from the Colnic porphyry indicates a variably potassic- and magnetite-altered plagioclase-hornblende porphyry, commonly overprinted by phyllic alteration, which is host to disseminated and veinlet-controlled chalcopyrite (see Table 7.6). In addition, minor occurrences of sphalerite and pyrrhotite are observed in core and petrography.

Table 7.6: Summary Petrographic Description of Mineralised Colnic Porphyry

Sample	Petrographic Summary Description
DH RCD-3 at 313.3 m	Intensely altered quartz-feldspar-hornblende(?) porphyry, with potassic alteration of K-feldspar-biotite and minor disseminated magnetite, overprinted by vein selvages of phyllic alteration (bleaching of biotite and presence of clays). Early vein set of quartz-pyrite-chalcopyrite-pyrrhotite apparently cut by later pyrite-chalcopyrite veins with phyllic selvage.
DH RCD-10 at 31.7 m	Intensely altered quartz-felpspar-possible mafic relics porphyry; quartz replacement dominant in groundmass. Relics of potassic alteration overprinted by strong phyllic alteration (sericite, clays) and carbonate, with likely silicification of groundmass. Several cross-cutting vein sets with (oldest to youngest): 1) aplitic quartz-pyrite-chalcopyrite; 2) comb quartz-pyrite-chalcopyrite-sphalerite; 3) coarse calcite; 4) fine-grained calcite.
DH RCD-16 at 203.10 m	Intensely altered plagioclase-mafic(?) porphyry with possible secondary quartz in matrix. Brown and green banding resultant from localised overprinting of propylitic assemblage on potassic alteration assemblage. Potassic alteration of biotite alteration of groundmass with magnetite, with minor overprinting of sericite and clays and minor disseminated sphalerite and pyrite. Green bands show extensive replacement of groundmass by green coloured clays and epidote. These are associated with much more intense development of magnetite, followed by pyrite, and then sphalerite plus chalcopyrite.

7.3.3.2 Colnic Lithology and Logging Unit Descriptions

Four different porphyries have been interpreted at Colnic. Major lithological units are described in detail below. The lithologic units and logging codes are summarised in Table 7.7.

Basement Sediment (SED–13): Sedimentary rocks occur along the north-eastern margin of the Colnic deposit, hosting the satellite porphyry and breccia package, and the garnet-bearing porphyry described below. This unit is made up of laminated, very fine-grained mudstone to fine-grained sandstone. Adjacent to the satellite porphyry, the sediments may be strongly fractured and biotite and/or sericite-clay enriched, becoming dark brown to pale grey/beige in colour. Quartz stockwork veining and minor copper mineralisation may be associated with locally altered sediments.

Cornitel (Wall Rock) Porphyry (WR-POR–14): A coarse-grained porphyry containing 7 to 12 modal per cent 5 to 10 mm black-green amphibole phenocrysts and 17 to 26 modal per cent 2 mm to 3 mm white to colourless plagioclase phenocrysts within an aphanitic groundmass occurs to the north, south, and west of the Colnic deposit. This unit is generally not veined or mineralised and is overprinted by pervasive phyllic to transitional alteration.

Table 7.7: Major Lithologic Units and Logging Codes for Colnic

	Code	Name	Description/Location
Colnic composite hornblende-plagioclase porphyries	5	Colnic Por. C (early mineral)	Higher-grade upper-part of Colnic Por. In Rovina Valley, often brecciated
	6	Deep Colnic Por. CD (Early inter-mineral)	Deep coherent part of Colnic Por.
	7	Por. Dykes and breccias LM-POR (inter-mineral)	Parallel set, structurally controlled along trend of Rovina Valley, principally in upper Colnic Por. C.
	4	F-2 Hill Por. and breccias F-2-POR (inter-mineral)	Southwest of Colnic Por. on F-2 Hill; moderate grade
	12	Garnet bearing Por. G-POR (late-mineral)	Satellite Por, low-grade, minor importance
	11, 9, 10	Satellite Por. and breccia package S-POR, S-IMB, S-BX (late-mineral)	Satellite Por, low-grade, minor importance
Colnic wall rocks	8	IMB Complex	Heterogenous fragmental hornblende-feldspar Por. with similar composition groundmass, highly variable textures (from igneous zenolithic to silty fluidised banding), predominant wall rock, locally mineralised
	14	Cornitel Wall Rock Por. WR-POR	Older, pre-mineral, and mostly barren
	13	Flysch (basement) sediment SED	Folded Cretaceous sediments, localised occurrence northeast area; barren
Colnic post-mineral units	3	Post-Mineral Por. Stock PM-POR	Deep axial-core intrusive within F-2 Hill Por.
	1	Post-Mineral Dykes LD-1	Principally on F-2 hill, likely related to Post-Mineral Por. Stock.
Faults	2	Chubby's Fault FLT	Northwest-striking, subvertical fault-fracture zone; post-mineral

Intrusive Magmatic Breccia (IMB–8): At Colnic, the mineralised porphyry stocks are enveloped by a heterogeneous package of clast-rich, laminated, and brecciated porphyry, similar to the IMB unit described in the Rovina deposit. This unit generally has a complex texture and contains sub-angular to rounded igneous rock fragments within a fine grained granular to porphyritic matrix. Intervals displaying well developed magmatic foliation are also common, especially adjacent to the margins of the F-2-Hill porphyry. At Colnic, the IMB unit is believed to represent a pre-mineral subvolcanic intrusion that subsequently became host to mineralised porphyry stocks.

Amphibole-Feldspar Porphyry: Much of the veining, mineralisation, and alteration at Colnic is hosted by a series of fine- to coarse-grained amphibole-feldspar porphyry stocks and dykes. However, correlation of the various early-mineral, inter-mineral, late-mineral, and post-mineral intrusions is complicated by the fact that cross-cutting porphyry phases are similar in composition, internally heterogeneous, and can be extensively brecciated, especially at their margins. Alteration associated with subsequent intrusions also masks intrusive contacts. However, using a combination of abrupt changes in geochemistry and alteration intensity and contact relationships observed while logging and re-logging drill core, eight distinct intrusive phases have been identified at Colnic. In general, the coherent porphyry stocks contain 7 to 12 modal per cent 1 mm to 6 mm black-green amphibole phenocrysts, and 17 to 26 modal per cent 1 mm to 2 mm (3 mm max.) white to colourless plagioclase phenocrysts within an aphanitic groundmass; however, within each unit variations in phenocryst content and grain size are common.

The following eight phases of amphibole-feldspar porphyry intrusion and brecciation have been identified at Colnic (from oldest to youngest):

- Colnic Porphyry and Breccia Package – C-POR-5 (early-mineral)
- Deep Coherent Colnic Porphyry Stock – CD-POR-6 (early inter-mineral)
- Late-Mineral Porphyry Dykes and Breccias – LM-POR-7 (inter-mineral)
- F-2-Hill Porphyry and Breccia Package – F-2-POR-4 (inter-mineral)
- Garnet Bearing Porphyry – G-POR-12 (late-mineral)
- Satellite Porphyry and Breccia Package S-POR-11; S-IMB-9; S-BX-10 (late-mineral)
- Post-Mineral Porphyry Stock – PM-POR-3 (post-mineral)
- Post-Mineral Porphyry Dykes – LD-1 (post-mineral)

<u>Colnic Porphyry and Breccia Package (C-POR–5)</u>: This unit represents the earliest intrusive phase at Colnic and is the most important host to quartz stockwork veining, alteration, and mineralisation in this area. A significant component of the Colnic porphyry package is made up of breccia (orthomagmatic and/or intrusion related). The remainder appears to consist of a heterogeneous package of medium-grained feldspar-amphibole porphyry; however, several episodes of overprinting alteration and intense quartz stockwork veining make identification of primary textures within this unit difficult.

In drillholes RCD-4, RCD-9, RCD-10, RCD-39, RCD-51, RCD-73, and RCD-82, intervals of micro-diorite have been identified within the Colnic porphyry unit. In general, these zones are fine-grained, weakly altered, and lack the pervasive quartz stockwork veining characteristic of the rest of the Colnic porphyry package. In general, the micro-diorite shoots are extensively brecciated. Where coherent bodies of micro-diorite occur, it is generally dark grey to green/brown in colour and contains 15 to 17 modal per cent ±1 mm amphibole, and 25 to 30 modal per cent 0.2 mm to 0.6 mm plagioclase crystals within an aphanitic groundmass. In RCD-10, RCD-51, and RCD-73, the micro-diorite is cross-cut by a series of sheeted quartz-sulphide veins that may be related to a moderate increase in the copper and gold grades within this unit.

Within the Colnic porphyry package, all drillholes containing micro-diorite, with the exception of RCD-4, occur on cross-section 1000N. In the current model, these intervals are believed to represent a volumetrically restricted, steeply dipping shoot of finer-grained igneous rock, emplaced into the Colnic porphyry package at a late stage in its development. Although the grade within these micro-diorite shoots is typically lower than in the surrounding rocks, they do contain moderate copper and gold mineralisation. For simplicity in the current interpretation, the micro-diorite has not been modelled separately from the Colnic porphyry package.

<u>Deep Coherent Colnic Porphyry Stock (CD-POR–6)</u>: This unit occurs in the deep parts of cross-sections 940N, 1000N, and 1050N, and consists of a coherent, medium-grained feldspar-amphibole porphyry intrusion that has been overprinted by pervasive reddish-brown potassic K3 alteration. Intrusive breccias are commonly associated with the margins of this

unit. The deep coherent Colnic porphyry appears to have been emplaced into the older Colnic porphyry package sometime after its formation and has subsequently been cross-cut by the northeast-to-southwest-striking late-mineral dykes and breccias described below.

<u>Late-Mineral Porphyry Dykes and Breccias (LM-POR–7)</u>: Several thin (generally 5 m to 15 m thick), north-northeast to northeast-striking, steeply southeast-dipping (65° to 85°) amphibole-feldspar porphyry dykes and breccia zones have been identified within the Colnic porphyry package, outcropping on the northern side of the Rovina Valley, and in holes drilled on cross-sections 940N, 1000N, and 1050N. The dykes and breccia zones are believed to have formed at a late stage in development of the Colnic deposit within a structural corridor running parallel to the Rovina valley. As a consequence, the late dykes and breccias cross-cut both the Colnic porphyry package and the deep coherent porphyry and are responsible for many of the sharp grade breaks occurring in holes drilled in this area. In addition to having unique geochemistry, the late-mineral dykes have been identified in the drill core using a number of observations, including the alignment of phenocrysts adjacent to dyke margins, the truncation of quartz veins, the presence of quartz vein fragments and quartz veined xenoliths within the dykes, and a general decrease in phenocryst size, quartz stockwork veining, and alteration intensity within the dykes relative to the surrounding rocks. Intrusion-related brecciation is also commonly associated with the margins of these units.

<u>F-2-Hill Porphyry and Breccia Package (F-2-POR–4)</u>: The F-2-Hill porphyry package consists of a heterogeneous assemblage of fine- to medium-grained feldspar-amphibole porphyry, micro-diorite shoots, and orthomagmatic and intrusion-related breccias that occur to the south of the Colnic porphyry package beneath F-2 Hill. These rock types appear to be related, and together form the host to the alteration, quartz stockwork veining, and mineralisation in this area.

<u>Garnet-Bearing Porphyry (G-POR–12)</u>: A laminated, garnet-bearing, medium-grained feldspar-amphibole porphyry stock occurs to the east of the F-2-Hill porphyry, in contact with the basement sediments. Locally weak to moderate quartz veining and K3 alteration occurs within this unit; however, it is not mineralised.

<u>Satellite Porphyry and Breccia Package (S-POR–11; S-IMB–9; and S-BX–10)</u>: A coherent medium- to coarse-grained feldspar-amphibole porphyry stock (11) associated with moderate quartz stockwork veining, K3 alteration, and weak to moderate copper mineralisation occurs to the northeast of the Colnic porphyry package. The position of this unit appears to be controlled by the northwest-trending contact between the intrusive magmatic breccia and basement sediments. A package of intrusive breccia (similar to the IMB unit described above) is closely associated with the emplacement of the satellite porphyry stock and has been modelled as S-IMB (9). Cross-cutting zones of matrix- to clast-supported breccia containing strongly veined K3+A (potassic overprinted by argillic) altered sediment and porphyry fragments have also been identified in association with the satellite porphyry stock and modelled separately as S-BX (10). These three units may, however, be related.

<u>Post-Mineral Porphyry Stock (PM-POR–3):</u> A post-mineral, propylitically altered, medium-grained porphyry stock has been identified within the F-2-Hill porphyry on cross-sections 670N and 740N.

Post-Mineral Porphyry Dykes (LD–1): Post-mineral, propylitically altered, medium-grained feldspar-amphibole porphyry dykes have also been identified within the Colnic deposit. These dykes may be related to the post-mineral stock described above.

Chubby's Fault (FLT–2): Chubby's Fault is a roughly north-northwest to northwest-trending zone of fractured rock, gouge, and fault breccia generally associated with intense argillic alteration.

7.3.3.3 Colnic Alteration and Logging Unit Descriptions

Several alteration types are recognised, with mineralisation mainly associated with the early potassic and locally overprinting MACE alteration. A list of alteration types and logging codes is shown in Table 7.8.

Table 7.8: Major Alteration Units and Logging Codes For Colnic

Alteration Style	Logging Code	Description
Potassic	K3	Widespread "biotisation" of groundmass, magnetite stringers, minor microscopic K-spar replacements; in parts, a brownish to pinkish coloration
Magnetite-propylitic	K2	Common magnetite diss. and in stringers, quartz, chlorite, epidote, amphibole, and carbonate, resulting in greenish-black colour and masking rock texture.
K-Alteration	K	Mixed assemblage of potassic (K3), with patchy overprinting magnetite-propylitic (K2), and overprinted by a structurally controlled (quartz-stockwork associated) phyllic alteration of variable intensity. Occurs mainly in the higher-grade upper Colnic Por.
Phyllic	PH	Broad wall rock halo of sericite, pyrite, quartz, pyrite; note typically, without quartz veinlets
Transitional Phyllic	TRPH	Phyllic overprint of earlier, mostly potassic and magnetite-propylitic alteration; extensive in the F-2 hill area.
Argillic	A	White clays and pyrite typically associated with late fracture zones. Restricted to Chubby's Fault zone
Propylitic	PR	Rare chlorite, carbonate, and pyrite in late dykes and post-mineral intrusives on F-2 Hill.
Potassic + Argillic	K3+A	Mixed early potassic with argillic along contact-zone fractures, exclusively in the Satellite por. package; of minor importance.

K3 Alteration: At Colnic, potassic (K3) alteration is the most common alteration type identified in the drill core and thin section. K3 alteration is generally associated with pervasive biotite or K-feldspar alteration of the groundmass, accompanied by quartz, magnetite, and disseminated sulphides (pyrite, pyrrhotite, and chalcopyrite). In porphyritic rocks, mafic phenocrysts are generally replaced by granular aggregates of "ratty" biotite, magnetite, and sulphides, while K-feldspar commonly replaces plagioclase phenocrysts. Depending on whether biotite or K-feldspar is more abundant, K3-altered rocks will have a dark brown or reddish-brown to pink colour, respectively.

Within the Colnic deposit, potassic alteration is best developed within the F-2-Hill porphyry and the deep coherent Colnic porphyry units. The late-mineral dykes are also commonly associated with potassic alteration.

K2 Alteration: At Colnic, it is common for K3 alteration to grade into or become overprinted by K2 alteration through a gradual increase of the amount of chlorite, magnetite, epidote, and actinolite present in the alteration assemblage. K2 alteration is generally most strongly developed adjacent to micro-fractures and veins containing magnetite-quartz-chlorite±actinolite and sulphides (pyrite, chalcopyrite). K2 alteration is generally associated with chloritisation of biotite, the replacement of plagioclase by epidote and calcite, and a much more intense development of magnetite, followed by pyrite and chalcopyrite. Where K2 alteration is pervasive, it is generally texturally destructive, giving the rock a dark green to light green colour. Where K2 alteration is less intense, the rock has a brownish-green to pinkish-green colour, and a mineral assemblage consisting of biotite-chlorite-magnetite-K feldspar-quartz-epidote-calcite±actinolite and sulphides (cpy-py).

K Alteration: The alteration occurring within the Colnic porphyry package is characteristically complex, consisting of patchy zones of K3 overprinted by K2 alteration, which in turn is overprinted by structurally controlled phyllic alteration. In addition, rocks occurring adjacent to late- and post-mineral breccias, faults, and fractured zones have been affected by low-temperature argillic alteration. The general term K is used here to describe an alteration package consisting of variable amounts of K2, K3, phyllic, and argillic alteration, typically associated with intense quartz stockwork veining.

TRPH Alteration (Transitional Phyllic): Transitional phyllic alteration is used to describe intervals where phyllic alteration partially overprints K2 or K3 alteration. Transitional phyllic alteration is most observed in the F-2 Hill area.

PH Alteration (Phyllic): Phyllic alteration is characterised by a dominant assemblage of sericite-quartz-pyrite±clay that is generally consistent with the low-grade peripheral parts of the Colnic deposit. Approximately 5 to 15 modal per cent pyrite generally occurs within this alteration type as 3 mm to 7 mm clusters of fine-grained aggregates, disseminated grains, and within mm-scale fractures. Phyllic alteration is typically pale grey to pinkish grey in colour.

P Alteration (Propylitic): Propylitic alteration is characterised by the assemblage of chlorite-carbonate-epidote. This alteration type is most observed within the post-mineral stock and dykes and as more pervasive alteration within the wall rocks outside of the phyllic alteration halo.

K3+A Alteration (Potassic-Argillic): Potassic-argillic alteration has been used to describe alteration associated with the brecciated parts (10) of the satellite porphyry package in which the matrix and margins of K3-altered sediment and porphyry clasts have been overprinted by argillic clays.

Alteration (Argillic): Argillic alteration generally occurs in association with late fractured/faulted zones, such as Chubby's Fault, and overprints all earlier alteration types. Argillic alteration is characterised by an assemblage of white clays (illite-smectite), quartz, and pyrite. Pyrite generally occurs as coarse, euhedral crystals.

7.3.3.4 Colnic Mineralisation Descriptions

Gold-copper mineralisation is associated with pyrite-chalcopyrite-magnetite occurring in veinlet stockworks and as disseminated grains. Oxidation is restricted to the uppermost few metres of the prospect, and no significant oxide cap or supergene-enriched horizons have been encountered to date.

Deposit-scale controls to mineralisation consist of the localisation of two hornblende-plagioclase porphyry centres: the Colnic porphyry and the F-2 Hill porphyry. The Colnic porphyry occurs in the Rovina Valley, elongated parallel to the northeast-trending valley over an area approximately 400 m long × 200 m wide. This is interpreted as the older porphyry, and its upper part contains the highest grades at Colnic. The centre of the F-2 Hill porphyry complex occurs approximately 150 m southeast of the Colnic porphyry. Interpreted structural controls on the emplacement of these porphyries are the northeast-trending Rovina Valley (as suggested by an inter-mineral dyke and breccia swarm in the upper part of the Colnic porphyry) and the northwest-striking Chubby's Fault/fracture zone (a brittle, post-mineral structure; however, may be a re-activated older structure, as evidenced by a spatial mineralisation association at depth). See Table 7.6, Table 7.7, and Table 7.8 for geology map and two cross-sections.

At Colnic, three early-stage magmatic-fluid alteration events are recognised: K3, K2, and a locally-occurring magnetite-only alteration. Gold-copper mineralisation at the Colnic porphyry is best developed within K2 and K3 potassic alteration of quartz diorite porphyry, particular where multidirectional stockwork vein intensity is highest. The earliest copper-bearing assemblage is observed both in early quartz veins and disseminated in the rock mass. It consists of magnetite+chalcopyrite+bornite+minor pyrite. Cross-cutting veinlets indicated multiple fracturing and hydrothermal pulses. Sixteen vein types have been recognised from detailed core logging, with five principal types associated with gold-copper mineralisation (see Table 7.9). Overprinting alteration events can obscure earlier events, particularly noted in parts of the higher-grade upper Colnic porphyry (in Rovina Valley) where phyllic alteration is associated with intense late-stage quartz±pyrite stockwork veining. In some cases, gold-copper mineralisation occurs in K2 alteration (and rarely in K3 alteration) without any apparent stockworking.

Table 7.9: Colnic Vein Types and Logging Codes

Vein Type Code	Relative Importance or Occurrence	Description
Bi-vn	Low	Biotite±K-feldspar veins
Mt-st	High	Magnetite stringers (M-vns)
Mts-st	High	Magnetite-sulphide ± quartz stringers
Cl-st	Low	Chlorite-carbonate ± sulphide ± magnetite stringers
Wq-vn	Low	Irregular early quartz veins, randomly oriented and often discontinuous
Dq-vn	Medium-low	Dark grey quartz veins
q-vn	Medium-low	Quartz veins
Bq-vns	Medium	"Banded" quartz ± sulphide veins
Qs-vn	High	Quartz-sulphide veins
Py-st	Medium	Pyrite stringers
Cp-st	High	Chalcopyrite stringers

Vein Type Code	Relative Importance or Occurrence	Description
Po-st	Low	Pyrrhotite stringers
Qcs-vn	Medium	Quartz-carbonate ± sulphide veins; low temperature (epithermal)
Ca-vn	Low	Carbonate/quartz veins/stringers
Tr-st	Low	Tourmaline stringers
z-vn	Minor	Zeolite veins

Locally, veins occur as sheeted zones; however, the predominant occurrence and appearance in core is as multidirectional stockworks. To evaluate mineralisation controls in the upper Colnic porphyry, which is elongated in the northeast direction and parallel to the angled drillholes of ESM, seven orthogonally angled drillholes were completed in late 2007. This drilling was instrumental in defining the late-mineral dyke and breccia swarm parallel to the northeast-trending Rovina Valley. As part of this orthogonal drilling programme, a structural study on oriented core was undertaken on four of these orthogonal drillholes. Drill core was oriented using the spear technique for marking bottom of core. Confidence limits were assigned based on precision of core marks between runs where core breaks could be aligned (as core quality was good, this was quite common). A total of 1,140 confident structural measurements were taken, of which 639 represent mineralisation-related veinlets. Results indicated a general multidirectional orientation of veinlets, with a slight bias to a northwest-southeast strike.

ESM routinely assays for zinc, lead, and silver in addition to gold and copper. At Colnic, the gold-copper mineralisation contains anomalous zinc ranging from 150 ppm to 600 ppm, with an approximate average of 300 ppm. A zone of elevated zinc + gold mineralisation has developed predominantly in or proximal to the transitional phyllic alteration zone (TRPH). Grades in this zone range from approximately 0.1 g/t Au to 3 g/t Au and 300 ppm Zn to 5,000 ppm Zn. The zinc-gold zone is interpreted to represent deposition of remobilised zinc and gold from a collapsing phyllic–potassic alteration front. The more acidic, H^2S-bearing fluids associated with the phyllic alteration (PH) may have dissolved gold and re-precipitated the element at the rock-composition redox contact with the magnetite-rich porphyry-style mineralisation (K2 and K3). In some cases, the contact zone between the phyllic and mineralised potassic zone grades from pyrite to pyrrhotite to magnetite, representing a possible sulphidation front. Proximal to and within Chubby's Fault/fracture zone, zinc mineralisation appears to be related to late-stage quartz–carbonate veinlets.

Molybdenum mineralisation is rarely observed in the drill core within quartz-molybdenite veinlets, typically in the deeper zones of K3 alteration. ESM has assayed on a limited basis for molybdenum, including some samples selected by visual observation of molybdenite in the drill core. Worldwide, other gold and gold-copper porphyries tend to be deficient in molybdenum; however, when present, it tends to concentrate as a halo to the copper-gold core (Sillitoe, 2000). Molybdenum assay results to date at Colnic have been insignificant (averaging < 5 ppm to 20 ppm Mo); however, a localised enrichment of molybdenum cannot be ruled out pending further molybdenum assay checks. Silver has been regularly assayed, and grades to date are typical of porphyries worldwide; values are generally very low throughout (averaging <1 ppm to 2 ppm), and do not constitute economic mineralisation. Table 7.10 shows an interpreted alteration-mineralisation paragenesis.

Table 7.10: Colnic Alteration Paragenesis

Alteration Phase	Timing ▶	Relative Importance	Sulphide Assemblage	Comment
Potassic (K3)		xxxxx	cp, po, py, (mo)	Deposit-wide, pervasive; some minor late inter-mineral intrusives lack this alteration
Magnetite-Propylitic (K2)		xxxxxxxx	py, cp, (sph)	Deposit-wide, patches
K-Alteration (K)		xxxxxxxx	py, cp, (sph, gal, Au)	Localised overprint associated with quartz stockwork-related phyllic
Silicification (S)		xx	py	Pervasive in K2, present in K3
Transitional Phyllic (TRPH)		Au–Zn	py, sph, po	Overprints outer margin of potassic zones
Phyllic (Ph)			py, (po)	Broad halo, no mineralisation
Propylitic (P)			py	Associated with post-mineral dykes
Argillic (A)			py, sph, gal, (cp, Au)	Chubby’s Fault zone and rare epithermal veinlets in phyllic halo

cp chalcopyrite
py pyrite
po pyrrhotite
mo molybdenite
sph sphalertite
gal galena
Au native gold
( ) rare

7.3.4 Ciresata Deposit Geology

7.3.4.1 Ciresata Geology Summary

Ciresata contains the highest average gold grades in the RVP, with gold-copper mineralisation hosted sub-equally in a Neogene subvolcanic “neck” and adjacent hornfelsed Cretaceous sediments. The subvolcanic intrusion is a relatively coarse-grained hornblende-plagioclase porphyry (Early Mineral Porphyry), with a narrow vertical feeder zone and “ballooning” at the dipping planar contact between the hornfelsed Cretaceous sediments and an older subvolcanic intrusion (Host Rock Porphyry), approximately 250 m below the present surface. Subsequent inter-mineral and late-mineral dyke-like intrusives commonly occur at depth and appear to be related to the occurrence of monolithic and polylithic breccias of surrounding host rock. Locally, some of the late- to post-mineral dykes, which cut mineralisation, intrude zones with more intense stockwork veining.

Mineralisation occurs as a broad quartz-pyrite-magnetite-chalcopyrite stockwork zone centred on the subvolcanic intrusive “neck”, and does not reach the present surface, occurring at 50 m to 150 m depth. Deep drilling in 2012 targeting the roots of the porphyry intersected gold-copper mineralisation 500 m below previous drilling, indicating a vertical extent of mineralisation of approximately 1,000 m; however, the presently known mineralisation extent indicates the widest lateral mineralisation dimension is in the upper part of the deposit, approximately 400 m below the surface, where the Early Mineral Porphyry expands spatially. The older capping porphyry (Host Rock Porphyry) preserves a barren altered litho-cap over the porphyry mineralisation, and is intensely phyllic-altered, overprinting an early magnetite alteration.

Surface mapping by ESM recognised a zoned suite of porphyry-style alteration, ranging from magnetite alteration (magnetite stringers) overprinted by phyllic in the litho-cap, outward to potassic, phyllic, and propylitic in a predominant cover of volcanic rocks. The dimensions of the mapped phyllic halo are approximately 1,400 m × 1,000 m. The magnetite alteration zone results in a positive magnetic anomaly. Results from outcrop sampling showed weakly anomalous to nil gold and copper mineralisation. Figure 7.13 shows the surface geology, and a cross-section through the Ciresata porphyry is shown in Figure 7.14.

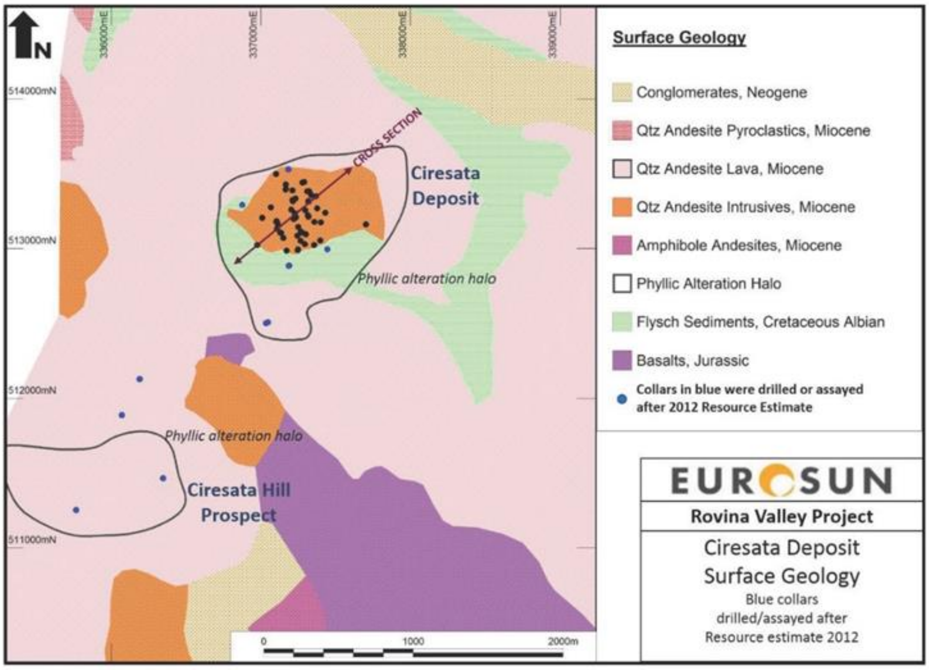


Figure 7.13: Surface Geology of the Ciresata Porphyry

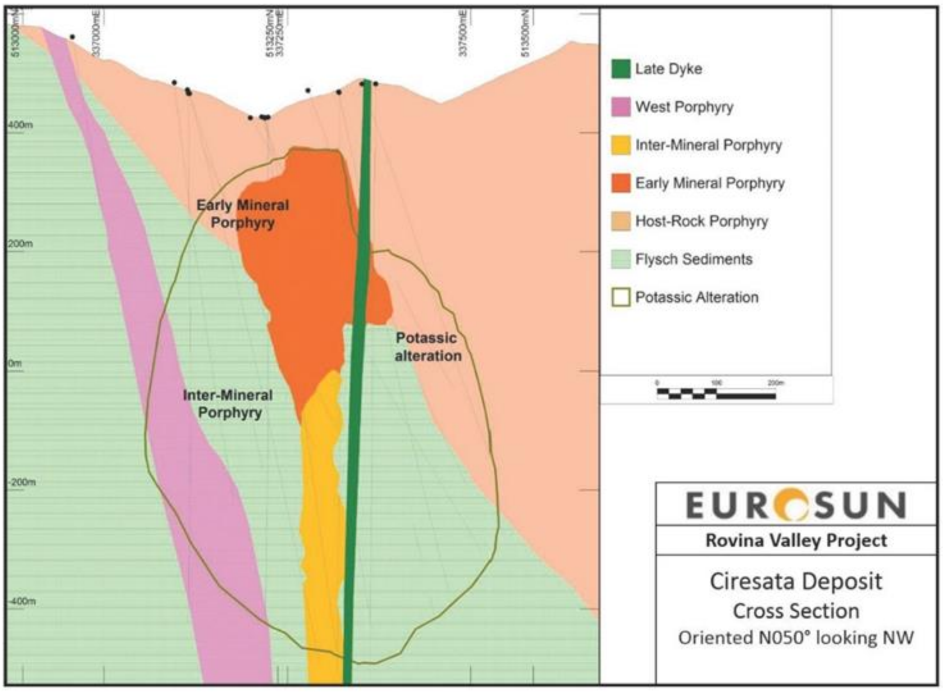


Figure 7.14: Cross-Section through the Ciresata porphyry

Alteration types associated with mineralisation include incipient biotite-potassium feldspar ± magnetite. Amphibole is commonly observed replacing pyroxene. This is interpreted to be an early-stage potassic alteration, and occurs in the Early Mineral Porphyry, Hornfelsed Sediment, and weakly in the Late Mineral Porphyry. A common alteration assemblage associated with higher gold-copper grades (chalcopyrite-pyrite) is quartz-magnetite-chlorite. This magnetite-chlorite alteration occurs in the Early Mineral Porphyry and in the Hornfelsed Sediment and is interpreted to be associated with the main stage mineralisation. In the deeper parts of the deposit, some veins have albite alteration selvages, and could represent sodic-calcic alteration (Ruff, et al. 2012).

A late-stage overprinting alteration of quartz-sericite-pyrite (Phyllic) occurs pervasively at the top of the deposit, and at depth, it occurs in fractures with white sericite halos. A late stage but widespread occurrence of weak, incipient replacement of rock-forming minerals by carbonate and kaolinite may be related to infiltration of groundwater during the final cooling of the system (Ruff et al. 2012).

Petrographic description of samples from the Early Mineral porphyry and mineralised Hornfelsed Sediment indicates secondary magnetite is prevalent in both the disseminated potassic alteration and in stockwork veining of quartz-magnetite-chlorite-pyrite-chalcopyrite (see Table 7.11).

Table 7.11: Petrographic Description of Mineralised Ciresata Porphyry and Hornfels

Sample	Petrographic Summary Description
RGD-8 (DH RGD-8 at 510.55 m)	Altered porphyritic microdiorite with cross-cutting millimetric veinlets. Rock-forming minerals are mostly replaced with biotite-K-spar-quartz, with carbonates and sericite forming a potassic alteration assemblage. Opaque minerals are represented by fine-grained disseminated pyrite and chalcopyrite and coarser-grained chalcopyrite-magnetite-pyrite in veinlets
RGD-7 (DH RGD-7 at 396.05 m)	Altered albite-biotite hornfels after clay or siltstone cut by cross-cutting veinlets. Albite-biotite are of contact metamorphic origin, with hydrothermal biotite observed associated with magnetite veins. Opaque minerals include common disseminated magnetite and chalcopyrite. Fissures and veinlets contain magnetite and chalcopyrite and are cross-cut by wider quartz+minor pyrite veins.
RGD-17 (DH RGD-17 at 449.10 m	Altered amphibole-plagioclase microdiorite. Amphibole (up to 3 mm) and plagioclase phenocryst crystals are replaced by sericite and secondary biotite. The crystalline micro-granular groundmass contains abundant disseminated secondary biotite, sericite, and quartz. Disseminated opaque minerals include pyrite-chalcopyrite-magnetite, which are cut by microfractures containing carbonate and quartz. Quartz veinlets contain abundant magnetite, chalcopyrite, and minor pyrite.

The Ciresata geologic model is relatively simpler than the Rovina and Colnic models, due to less extensive early and inter-mineral stage porphyries. The Early Mineral Porphyry is the principal mineralising porphyry, with the later Inter-Mineral Porphyry intruding the root feeder zone, which has mineralised the hornfelsed wall rock.

7.3.4.2 Ciresata Lithology and Logging Unit Descriptions

The Ciresata deposit exhibits an apparently simple intrusive history and fewer lithologic logging and modelling units have been defined compared with Colnic and Rovina. Descriptions of these units are provided below, and a list of major lithologic units and logging codes is shown in Table 7.12.

Table 7.12: Ciresata Major Lithologic Units and Logging Codes

	Code	Name	Description/location
Ciresata mineralised porphyries	64	Early Mineral Por. EM-P	Principal mineralised porphyry; coarse-grained hornblende-plagioclase microdiorite porphyry; narrow dyke-like root zone expanding at the wall rock contact between the hornfelsed sediment and the overlying Host Rock Porphyry.
	65	Inter-Mineral Porphyry (IM-P)	Weakly mineralised amphibole-plagioclase microdiorite porphyry; intrudes the root zone of the EM-P.
Ciresata wall rocks	68	Basement Sediment (SED)	Folded Cretaceous sediments, hornfelsed near contact zone with EMP, where it becomes an important host to mineralisation.
	71	Host Rock Porphyry (HR-P)	Highly altered, older capping porphyry, with heterogenous primary textures displaying auto-brecciation and locally fluidised zones.
Ciresata late-post-mineral intrusives	62	Late-Mineral Porphyritic Dykes (LM-P)	Amphibole-plagioclase porphyry occurring as < 40 m thick dyke intrusives; weak to unmineralised, though wall rock contact zones can have intense pre-existing veining.
	70	West Porphyry (W-P)	Very weakly mineralised dyke-like amphibole-plagioclase porphyritic intrusion west of the deposit

Overburden (OB – 60): The completely oxidised colluvium, alluvium, residual soil, and weathered rock occurring at the top of most drillholes.

Basement Sediment (SED – 68): The basement sediments at Ciresata are made up of massive, compact, very fine-grained mudstone and minor fine-grained sandstone that have undergone thermal contact metamorphism prior to an overprinting hydrothermal alteration, which tends to obscure original textures. Altered sedimentary rocks have been intersected in most of the drillholes completed at Ciresata and host the early- and late-mineral intrusions that make up the Ciresata porphyry complex. These intrusive contacts with the SED unit are easily observed in the drill core. The SED unit occurring adjacent to the intrusive bodies generally exhibit strong stockwork veining of biotite-magnetite-quartz-pyrite-chalcopyrite and are black in colour. These altered and veined sediments contain some of the most intense veining and highest gold-copper grades at Ciresata. Distal to the intrusions and at depth, the sediments become alternating light grey/beige in colour, with recognisable thin interbeds of sandstone, grading outward into fresh, less compact, and soft sediments.

Host Rock Porphyry (HR-P – 71): Above the Ciresata deposit and representing the surface expression, is an intensely phyllic-altered, porphyritic volcanic unit with a heterogenous primary texture. The general appearance is of a “clotty” porphyry, related to alteration of mafic phenocrysts within a rock fabric that includes coherent clasts within an aphanitic, microgranular, and porphyritic matrix suggestive of auto-brecciation. Locally, the matrix material exhibits lamination textures suggestive of fluidised flow during emplacement. Petrographically, the modal mineralogy of this rock differs from the subjacent EM-P, due to the occurrence of primary quartz in the groundmass. The HR-P is weakly- to moderately-mineralised immediately above the EM-P contact and displays extensive early-stage magnetite veinlet alteration that has survived an intense phyllic alteration overprint and is exposed on the present surface above the blind gold-copper mineralisation. From surface mapping and drillhole logging, the HR-P has been interpreted to represent the oldest and largest intrusive unit in the Ciresata deposit area, intruding upwards and westward along bedding dip planes of the Cretaceous sediments.

Early-Mineral Porphyry (EM-P – 64): The Early-Mineral Porphyry at Ciresata is a coarse-grained porphyritic intrusion containing 7 to 12 modal per cent 5 mm to 10 mm black-green hornblende phenocrysts, and 17 to 26 modal per cent 2 mm to 3 mm white to colourless plagioclase phenocrysts within a microcrystalline groundmass. Petrography classifies this rock as a porphyritic microdiorite. This intrusive unit has a narrow dyke feeder zone expanding in volume at the wall rock contact between the SED unit and the capping HR-P unit. The EM-P is commonly associated with intense quartz stockwork veining, pervasive potassic alteration, and significant gold-copper grades, and represents the centre of the Ciresata mineralising system. A strong phyllic alteration overprint in the upper part of the deposit can locally obscure the contact between the EM-P and the older, capping HR-P.

Inter-Mineral Porphyry (IM-P -65): A hornblende-plagioclase porphyry, of similar phenocryst and groundmass modal mineralogy to the EM-P except for primary quartz identified in the groundmass, as determined through petrography. The IM-P is relatively weakly altered, with low to moderate intensity stockwork veining and a visible intrusive contact with the EM-P in the drill core. In addition, the IM-P is observed to contain refractory quartz vein fragments and clasts of stockwork-mineralised EM-P. The IM-P has a restricted occurrence intruding the deeper dyke root zone of the EM-P and hosts low-grade gold-copper mineralisation.

West Porphyry (W-P 70): A hornblende-plagioclase porphyritic dyke-like intruded into the SED unit, west of the Ciresata mineralisation and outside of the gold-copper mineralisation halo. This unit is very weakly mineralised, with an incipient potassic alteration interpreted to be metosomatic, and a propylitic alteration overprint.

Late-Mineral Porphyry (LM-P – 62): Locally, sub-vertically dipping, north-south to northwest-striking amphibole-feldspar porphyry dykes and associated intrusion-related breccia zones have been identified within the Ciresata porphyry assemblage. In general, these dykes are < 40 m thick and are barren or very weakly mineralised. The dykes have a similar texture and mafic content to the EM-P but appear to contain a higher percentage (typically > 30 modal per cent) of 1 mm to 3 mm plagioclase phenocrysts, and contain primary modal quartz in the groundmass, based on petrographic observations. The dykes and breccia zones are late- to post-mineral stage and form a relatively small volume within the Ciresata deposit. Near the surface, the late-mineral dykes are generally light green to grey in colour, and chlorite-clay-magnetite-epidote (propylitic) or clay-sericite-pyrite-quartz (phyllic or argillic) altered; however, at deeper elevations the late-mineral dykes are overprinted by pervasive potassic alteration. Locally, significant gold-copper stockwork mineralisation occurs in the SED wall rocks adjacent to the dykes' margins, with the dykes themselves having inclusions of mineralised wall rock clasts; however, the late-mineral dykes in general are weakly mineralised to unmineralised.

Breccias: Breccias occur locally marginal to the Ciresata mineralisation core. These breccias include zones of monolithic wall rock breccia and polylithic brecciation that may have formed in response to magmatic fluid discharge from late- to post-mineral porphyry phases. In general, these units are affected by the similar alteration and stockwork intensity as the surrounding rocks and were not modelled separately.

7.3.4.3 Ciresata Alteration and Logging Unit Descriptions

Alteration associated with mineralisation is predominantly potassic+magnetite and magnetite-chlorite (MACE). A list of alteration types and logging codes is shown in Table 7.13.

Table 7.13: Major Alteration Units and Logging Codes for Ciresata

Alteration Style	Logging Code	Description
Potassic	PT (31)	Widespread “biotisation” of groundmass; magnetite prevalent as stringers and disseminated; minor microscopic K-spar replacements in both EMP and hornfels. Most important alteration and correlates with gold-copper mineralisation.
Magnetite-Chlorite	MACE (30)	Common magnetite-chlorite disseminated and in quartz veinlets, ± epidote, amphibole. Patchy widespread occurrences associated with higher-grade mineralisation.
Phyllic	PH	Broad wall rock halo of sericite, pyrite, quartz
Transitional Phyllic	TRPH (43)	Phyllic overprint common in the porphyry cover rocks; appears to overprint a magnetite-rich propylitic alteration in the HR-P.
Argillic	A	White clay alteration halos to pyrite + quartz-bearing micro-fractures; occurs pervasively on a deposit scale, though volumetrically minor (intermediate argillic).
Propylitic	P	Chlorite, carbonate, sericite, pyrite in upper part of late dyke intrusives and the EM-P; grades into potassic alteration at depth
Weathering oxidation	TOX (41)	Localised and limited to upper few- to-tens of metres, limonite, goethite, and red-brown hematite with white clays

Total Oxidation (TOX – 41): Related to surface weathering and comprising rusty orange-brown limonite, goethite, red-brown hematite, and white clay enriched oxidation, located within a few to tens of metres from the surface, and do not affect the deeper gold-copper mineralisation.

Potassic Alteration (PT – 31): Potassic alteration is the most common alteration type occurring within the Ciresata deposit, and effectively outlines the maximum extent of gold-copper mineralisation in this area. Potassic alteration is generally associated with a mineral assemblage that includes biotite-quartz±K-feldspar±magnetite and pyrite>chalcopyrite. The alteration minerals occur as finely disseminated grains in the groundmass of porphyritic and sedimentary rocks, and less pervasively as granular aggregates of “ratty” biotite-magnetite±sulphides after mafic phenocrysts in the porphyries. This mineral assemblage gives the rock a dark brown to reddish-brown colour.

Within the Ciresata deposit, elevated gold-copper grades are closely associated with biotite-magnetite enrichment, intensely developed quartz-magnetite-pyrite-chalcopyrite stockwork veining within the early-mineral porphyry, and in a broad halo extending into the adjacent hornfelsed sedimentary rocks.

MACE Alteration (MACE – 30): MACE alteration at the RVP is a logging term used to describe a magnetite-amphibole-chlorite-epidote alteration imparting a greenish appearance to the drill core, which is commonly associated with gold-copper mineralisation. At the Ciresata deposit, the MACE alteration is comprised of quartz-magnetite-chlorite±epidote with pyrite+chalcopyrite. It is common for rocks to have been potassically altered prior to becoming overprinted by MACE alteration. Therefore, where MACE alteration is less intense, the rock generally has a brownish-black colour and a predominant alteration mineral assemblage of biotite and magnetite, whereas the localised MACE alteration consisting of chlorite-magnetite-quartz with associated pyrite-chalcopyrite gives the rock a dark green to light green colour. The MACE alteration assemblage is typically associated with magnetite stringers and clots and well-developed quartz-sulphide vein stockwork, with significant fine-grained disseminated cpy-py-mt mineralisation. At Ciresata, the MACE alteration is "patchy" in its recognised distribution, and thus the PT is a better deposit-wide guide to gold-copper mineralisation.

Transitional Alteration (TRPH – 43): Transitional phyllic alteration is the most common alteration type effecting porphyritic rocks in the upper parts of the Ciresata deposit. This alteration type is characterised by the selective replacement of mafic phenocrysts by clotty zones of chlorite-magnetite-epidote-carbonate and a more pervasive clay-sericite-quartz-pyrite enrichment of the groundmass. Magnetite stringers and clots and weakly developed quartz-sulphide veining are also commonly associated with this alteration type; however, gold-copper grades are generally low.

Alteration (Argillic): Argillic alteration occurs as white alteration selvages (ranging from 1 cm to 2 cm wide to 10 m wide illite-smectite) to planar fractures and breccia-fracture zones. These fractures contain sparse pyrite±quartz/carbonate. This alteration should more appropriately be termed Intermediate Argillic to be consistent with alteration terminology (Seedorff et al, 2005). These fractures and alteration are late- to post-mineral, locally overprint earlier mineralisation/alteration, and in general do not have any effect on gold-copper grade, though in some cases this alteration may have decreased the grade. This fracture-controlled alteration is common throughout the deposit, typically spaced 20 m to 30 m apart.

PH Alteration (Phyllic): Phyllic alteration is characterised by a dominant assemblage of sericite-quartz-pyrite±microcrystalline clay minerals that is generally consistent with the low-grade peripheral parts of the Ciresata deposit. Approximately 5 to 15 modal per cent pyrite generally occurs within this alteration type as 3 mm to 7 mm clusters of fine-grained aggregates, disseminated grains, and within mm-scale stringers. Phyllic alteration typically has a pale grey to pinkish grey colour within porphyritic rocks, and a pale grey-beige colour within sedimentary rocks. In the upper parts of the Ciresata deposit, the phyllic alteration is intense, forming a white rock, and overprints earlier alteration types.

To simplify the current interpretation, phyllic alteration and argillic alteration (Intermediate Argillic) were modelled together as quartz-sericite-pyrite±clay alteration.

P Alteration (Propylitic): Propylitic alteration is characterised by the assemblage of chlorite-carbonate-sericite±epidote. This alteration type is most commonly observed within and adjacent to the upper parts of the late-mineral dykes.

Sodic Alteration: In some deep drillholes at Ciresata, minor occurrences of sodic alteration have been observed. This alteration occurs as narrow (1 cm to 5 cm wide) milky-white selvages, to early quartz veins with "wobbly", or curviplanar boundaries. The white alteration appears to be a replacement of rock minerals which are interpreted to be microcrystalline albite. Owing to the rare occurrence of this alteration and its apparently negligible effect on gold-copper mineralisation, it has not been modelled.

7.3.4.4 Ciresata Discussion

From the observations above, the gold-copper mineralisation at Ciresata appears to be bracketed by the host rock porphyry emplacement and the late porphyry dyke emplacement. An interpreted sequence of Neogene geologic events is as follows:

- Cretaceous sediments (SED) intruded by the host rock porphyry (HR-P), with intrusive contact largely controlled by northeast-dipping bedding planes
- Thermal contact metamorphism of SED unit grading outward from the HR-P contact, resulting in hornfels
- Vertical emplacement of the narrow, pipe-like early mineral porphyry (EMP) within the SED, expanding at the SED/HR-P contact
- Multiple stockwork fracturing events of the EMP-SED and overlying HR-P, with associated alteration and mineralisation
- Intrusion of the inter-mineral porphyry (IM-P) in the root zone of the EM-P
- Post- to very late-stage mineral intrusion of dykes, including the late-mineral porphyry and the west porphyry
- Subsequent cooling and un-roofing of the mineral system

This interpreted sequence of geologic events at the Ciresata deposit with both pre- and post-mineral intrusions is commonly observed in porphyry deposits (Sillitoe, 2000, and Seedorff et al, 2005). Recognition of inter-mineral porphyry intrusions is important due to their effects on grade but can be difficult to identify due to similar compositions and alteration/mineralisation overprints (Sillitoe, 2000). Late- to post-mineral intrusions result in sharp grade breaks. Gold-copper mineralisation is presented in the following section.

7.3.4.5 Ciresata Mineralisation Descriptions

Gold-copper mineralisation at Ciresata is associated with magnetite-pyrite-chalcopyrite occurring in veinlet stockworks and as finely disseminated grains over a wide area of approximately 450 m (NW-SE) by 300 m (NE-SW) and narrowing with depth. Recent deep drilling has intersected mineralisation 500 m below previous drilling, suggesting approximately 1,000 vertical metres of mineralisation. This mineralisation is centred on the early mineral porphyry (EM-P), with approximately 65 % hosted in the hornfels sediments (SED) and 35 % in the EM-P. In general, grade decreases as a function of distance away from the EM-P-to-SED contact. In the southwest sector of the deposit, SED is the only host, and with the local occurrence of higher grades, a deeper mineral porphyry has been postulated but not yet confirmed by drilling.

Gold correlates positively with copper grade and the gold:copper ratio is relatively constant throughout the deposit. Chalcopyrite is the only copper mineral present. Petrography and gold deportment studies show that gold is fine-grained, and associated with grains of chalcopyrite, pyrite, quartz, and rarely with magnetite (Damian, 2011; Wang and Prout, 2008; Sliwinski, 2012). In addition, scanning electron microscope (SEM) analysis of gold grains from Ciresata indicates they are native gold, with an average composition of 94.7 % Au and 5.2 % Ag. On a deposit scale, marginal to the gold-copper mineralisation, anomalous zinc (up to 400 ppm) and lead (up to 40 ppm) occur and appear to be associated with the phyllic alteration halo. Within the mineralisation, zinc is anomalous, with an approximate average value of 170 ppm. Limited analysis for molybdenum (670 samples) showed assays ranging from 2.5 ppm to 121 ppm, with an average of 20 ppm. Weathering oxidation is restricted to the uppermost tens of metres, and thus does not affect the gold-copper mineralisation.

An early-stage magmatic-fluid alteration event, potassic (PT, biotite+magnetite) is recognised with less extensive overprinting magnetite-chlorite (MACE) alteration. The broad outline of PT alteration correlates well with the outer limit of mineralisation in both the SED and porphyry units. Magnetite is a predominant alteration mineral, and the abundance of magnetite coupled with stockwork vein intensity generally correlates with gold-copper grade.

Higher grades of gold-copper mineralisation, in the core of the porphyry body and near contacts with dykes and SED, are associated with broad zones of intense quartz-magnetite-pyrite-chalcopyrite stockwork veining (up to 80 % of rock volume). Other important vein types recognised are thick-banded (1 cm to 10 cm) quartz-pyrite-chalcopyrite veins (commonly associated with higher gold grades) and magnetite-chalcopyrite stringers. Cross-cutting veinlets indicated multiple fracturing and hydrothermal pulses. Vein types and logging codes are shown in Table 7.14.

Table 7.14: Vein Types and Logging Codes, Ciresata

Vein-Type Code	Relative Importance or Occurrence	Description
Bi-vn	Low	Biotite-magnetite±K-feldspar veinlets
Mt-st	High	Magnetite stringers (M-vns)
Mts-st	High	Quartz-magnetite–sulphide veinlets
Wq-vn	Low	Irregular early quartz veinlets; randomly oriented and often discontinuous
Dq-vn	Medium-low	Dark grey quartz veinlets
q-vn	Medium-low	Quartz±sulphides veinlets
Bq-vn	High	Thick-banded quartz-magnetite-sulphide veins
Qs-vn	High	Quartz-sulphide veins
Mq-vn	Low	Thick, irregularly shaped milky quartz veins
Py-st	Low	Pyrite stringers
Cp-st	High	Chalcopyrite stringers
Po-st	Low	Pyrrhotite stringers
Qcs-vn	Low	Quartz-carbonate±sulphide veins; open space, low temperature (epithermal)
Ca-vn	Low	Carbonate/quartz veins/stringers
Tr-st	Rare	Tourmaline stringers
M-vn	Rare	Quartz-Mo veinlet

The Ciresata gold-copper mineralisation and associated alteration assemblages correspond to described porphyry models (Sillitoe, 2000, and Seedorff et al, 2005). Early-stage magmatic-hydrothermal fluids introduced potassium and iron in the form of magnetite (PT alteration), and gold-copper. Distal the core of mineralisation, these fluids evolved, resulting in hydrolysis reactions with the calc-silicate wall rocks, and forming a phyllic alteration assemblage (PH). With time and cooling, these distal fluids may collapse on the potassic altered core and overprint with a phyllic assemblage (TRPH), and infiltrate the potassic core through brittle fractures, forming the fracture-controlled intermediate-argillic (A) to phyllic alteration assemblages (PH). Ciresata does not have an epithermal overprint, apart from very sparse open-space-filling quartz-calcite±galena-sphalerite. A timeline alteration paragenesis is shown in Table 7.15.

Table 7.15: Ciresata Alteration Paragenesis

Alteration Phase	Timing →	Relative Importance	Sulphide Assemblage	Comment
Potassic (PT)		xxxx	cp, py, (mo)	Deposit-wide, pervasive
Magnetite-Chlorite		xxxxxxxx	py, cp	Deposit-wide, patches
Sodic (Na)		x	py	Occurs only at depth
Silicification (S)		xx	py	Common in Mag-Chl alt.
Phyllic (Ph)			py	Broad halo, no mineralization
Propylitic (P)			py	Late Mineral porphyry
Carbonate (CC)			py,	Pervasive in vein halos
Argillic (A)			py	Fracture controlled

It is CCIC MinRes's opinion that the level of understanding of the geology, structure, and mineralisation of the RVP has advanced to an adequate level to support mineral resource estimation.

8 DEPOSIT TYPES

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

8.1 GENERAL

The principal targets on the RVP are related to the porphyry copper-gold mineral deposit model. Porphyry deposits are generally large, low- to medium-grade deposits in which primary (hypogene) sulphide minerals are dominantly structurally controlled, and which are spatially and genetically related to felsic to intermediate porphyritic intrusions (Seedorff et al., 2005). The large size and structural control (e.g. veins, vein sets, stockworks, fractures, 'crackled zones', and breccia pipes) serve to distinguish porphyry deposits from a variety of deposits that may be peripherally associated, including skarns, high-temperature mantos, breccia pipes, peripheral mesothermal veins, and epithermal precious metal deposits. Secondary minerals may be developed in supergene-enriched zones in porphyry Cu deposits by weathering of primary sulphides. Such zones typically have significantly higher Cu grades, thereby enhancing the potential for economic exploitation (Sinclair, 2006).

Porphyry deposits occur throughout the world in a series of extensive, relatively narrow, linear metallogenic provinces. They are predominantly associated with Mesozoic to Cenozoic orogenic collisional belts in western North and South America and around the western margin of the Pacific Basin, particularly within the Southeast Asian Archipelago. However, major deposits also occur within Palaeozoic orogens in Central Asia and eastern North America and, to a lesser extent, within Precambrian terranes (Sinclair, 2006 and Seedorff et al., 2005).

Porphyry deposits are large and the world's most important source of Cu, Mo, and Re, and are major sources of Au, Ag, Sn and significant by-product metals include W, In, Pt, Pd, and Se. They account for approximately 50 % to 60 % of the world Cu production (Sinclair, 2006). Grades for the different metals vary considerably but generally average less than 1 %. In porphyry Cu deposits, Cu grades range from 0.2 % to more than 1 % Cu; Mo content ranges from approximately 0.005 % Mo to 0.03% Mo; Au contents range from 0.004 g/t Au to 0.35 g/t Au; and Ag content ranges from 0.2 g/t Ag to 5 g/t Ag. Re is also a significant by product from some porphyry Cu deposits. Some Au-rich porphyry Cu deposits have relatively high contents of Pt-group elements (PGE) (Mutschler and Mooney, 1995; Tarkian and Stribrny, 1999 in Sinclair 2006).

8.2 GOLD (± COPPER) PORPHYRIES

Copper grades in porphyry Au + Cu deposits range from negligible to comparable to those of the porphyry Cu + Au subtype, but Au contents tend to be consistently higher, averaging between 0.2 g/t Au and 2.0 g/t Au. Because of the apparent independent relationship between the Cu and Au content of porphyry deposits, Sillitoe (2000) suggested that porphyry deposits should contain > 0.4 g Au/t to be called Au-rich. Other workers have suggested using an Au to Cu ratio instead of absolute grade to determine an Au-Cu Porphyry subtype (Seedorff et al., 2005, and Murakami, et al., 2009). Sillitoe (2000) concludes that the geologic features of Au-rich porphyries are very similar to the Cu-Au porphyry subtype. The Au endowment in porphyry deposits is the topic of much of the current porphyry deposit research which is considering tectonic settings and magmatic genesis, basement rock compositions, and emplacement fluid geochemistry processes (Seedorff et al. 2005, Murakami et al. 2010, Halter et al. 2002).

Most Cu-Au porphyry intrusive complexes consist of a series of both pre- and post-mineralisation intrusions. The pre-mineralisation intrusions are generally equigranular in texture and genetically related to the porphyry stock, and often intrude along the shoulders of the pre mineralisation intrusion. Post-mineralisation dykes and plugs and diatremes are also commonly associated. Various hydrothermal breccias occur as early orthomagmatic (strong K-silicate altered) and/or late phreatic and phreatomagmatic varieties (Sillitoe, 2000, and Seedorff et al, 2005).

Copper and gold grades in the early orthomagmatic breccias may be substantially higher than in the surrounding porphyry rocks, while later breccia types are generally of sub-economic grade. Large (> 0.5 km wide) low-grade or barren diatreme breccias and minor pebble dykes often conclude the evolution of gold-rich porphyry systems (Sillitoe, 2000).

Most Cu-Au porphyry systems consist of varying quantities of six principal alteration types, namely Ca–Na silicate, K-silicate (potassic), propylitic, intermediate argillic (sericite–clay–chlorite), sericitic, and advanced argillic. Colnic, Rovina, and Ciresata all display characteristics of several of these alteration types.

Gold-rich porphyry deposits are typically associated with abundant magnetite in the early K-silicate alteration phase and also in the intermediate mineralisation stages (Sillitoe, 2000). Economically mineralised zones commonly form upright cylinders or bell-shaped zones. Intermediate, sericitic, and advanced argillic zones can also host economic grades of copper and gold, but less frequently than the K-silicate zone. The mid-parts of many porphyry deposits correlate with the highest gold grades, often as high as double that in upper or lateral margin parts. Gold is generally fine grained (< 20 µm, and often < 100 µm). In pyrite-rich Au porphyry deposits, gold strongly correlates with the pyrite; while in pyrite-poor deposits, gold is commonly associated with chalcopyrite or bornite (Sillitoe, 2000).

In AGP's opinion, the deposits of the RVP area are considered examples of porphyry deposits due to association with the porphyritic intrusive complexes, alteration assemblage, and mineralisation style described above. According to Sillitoe, porphyry deposits with average gold grades > 0.4 g/t Au can be generalised as “gold-rich”. Gold grades returned from the drilling at Colnic and Ciresata tend to average in the range of 0.8 g/t Au to 1.2 g/t Au, with copper grades averaging around 0.1 % Cu to 0.2 % Cu. Gold grades at Colnic and Ciresata are > 0.4 g/t Au and Au/Cu ratio (g/t Au / % Cu) of 5, which falls under the classification of a true Au-Cu porphyry subtype while Rovina, with an Au/Cu ratio of one, falls under the Cu-Au porphyry subtype.

9 EXPLORATION

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

Most of the exploration on the property has been performed by three companies: Minexfor between 1974 and 1998, and again in 2001, Rio Tinto from 1999 to 2000, and ESM since 2004. In September 2011, Barrick Gold and ESM formed an exploration collaboration group to evaluate further exploration targets on the Rovina licence. Early-stage exploration focused on property-wide target generation and was dominated by soil and stream sediment geochemical surveys, and regional airborne geophysical programmes, undertaken in conjunction with surface and underground geological mapping, trench sampling, and detailed ground geophysical programmes. Specific details of previous early-stage programmes (Minexfor and Rio Tinto) are not well documented. ESM has purchased available documentation for the Minexfor and Rio Tinto work stages.

Various exploration techniques have been utilised during the exploration stages and are described below.

9.1 COORDINATES AND DATUM

Coordinates used by ESM, Rio Tinto, and Minexfor are in the "Stereo70" grid system, which is the official coordinate system used in Romania. The exploration licences registered with NAMR are also in this grid system. ESM utilises the Stereo 70 system, which is compatible with standard GIS software packages. When GPS surveys are used, UTM Datum WGS 84 Zone 35 coordinates are converted to Stereo 70.

9.2 MINEXFOR (1975 TO 2000)

9.2.1 Geographic/Grid Control

There is no documentation of the grids constructed by Minexfor; however, it is believed that all maps, sample locations, and drillhole collars are registered and reported in the Stereo 70 coordinate system.

9.2.2 Topography

There is no reference in the available documents defining the source for the topographic base used by Minexfor. Presented topographic maps are presumed to be from standard government published maps at 1:5,000 scale.

9.2.3 Geological Mapping and Related Studies

Geologic mapping completed by previous exploration groups on the property is limited or not well documented. From Minexfor, some prospect-scale geology maps have been obtained by ESM.

9.2.4 Ground Geophysics

The Romanian government had reportedly completed IP/resistivity, natural polarisation, and gamma ray geophysical surveys at Colnic in 1977–1978. Those surveys were apparently completed on ten north–south lines, spaced 200 m apart. No additional information is available on these surveys. That 30-year-old data has been superseded by recent work. Additional magnetic and gamma ray geophysical surveys were completed by Minexfor during 1983 to 1984. ESM did not receive copies of any of the results of these surveys.

9.2.5 Drill Core Sampling

As a follow-up to the early-stage work, Minexfor tested various geophysical anomalies and surface defined targets at the Colnic Deposit area (14 core holes totalling 4,740 m) and the Rovina Deposit (34 holes totalling 23,119 m). They also tested the Ciresata prospect, located approximately 4.5 km south of Colnic with six wide-spaced core holes totalling 1,200 m.

Details regarding Minexfor drill core sampling programmes are very poorly documented and are not discussed in this report. None of the Minexfor core is in a usable format and most of it has been dumped in heaps adjacent to the Rovina deposit. ESM's purchased historical data package includes hard copies of drill logs with coordinates and hand-written assays for many of the completed holes.

9.3 RIO TINTO (1999 TO 2000)

9.3.1 Geographic/Grid Control

Rio Tinto established grids for soil surveys over four separate prospects within the present ESM property as shown in Table 9.1.

All grids were oriented north–south and were designed mainly for soil geochemical sampling. There is no documentation in the data regarding how the grids were surveyed and whether they were physically marked on the ground. All maps, sample locations, and drillhole collars are registered and reported in Stereo 70 coordinate system.

Table 9.1: Rio Tinto Grids

Licence	Prospect	Area (ha)	Grid Spacing	Purpose	Range of Au Values (g/t)
Rovina	Colnic	220	200 m × 200 m and 100 m × 100 m in northwest	164 soil samples	0 to 0.69
	Ciresata	1,120	200 m × 100 m and 200 m × 200 m in southeast	500 soil samples	0 to 0.11
NOTE: Licence is the present ESM property; Rio Tinto had a much larger licence					

9.3.2 Topography

There is no reference in the available documents defining the source for the topographic base used by Rio Tinto. Presented topographic maps are presumed to be from standard government published maps at 1:5,000 scale.

9.3.3 Geological Mapping and Related Studies

Exploration reports from Rio Tinto indicate no geologic mapping was completed beyond reconnaissance-style investigations. Work by other groups on the property, prior to ESM, is not well documented.

9.3.4 Airborne Magnetic/Radiometric Geophysical Surveys

In 1999, Rio Tinto completed a helicopter-borne magnetic/radiometric survey (flown by Fugro Airborne Corp. out of Canada) over an area approximately 24 km by 30 km. The area covered by the survey included both the Colnic and Rovina porphyries, which were in the north-eastern portion of the Rio Tinto permit.

The airborne survey was completed on east–west-oriented, 150 m-spaced lines using a helicopter elevation of 60 m and a sensor elevation of approximately 35 m. Total survey length was 3,995-line km. In 2005, ESM purchased the digital TIFF-format images for this survey from the NAMR. Seven images were provided, including potassium, thorium, uranium, total magnetic field, reduced-to-pole, analytical signal, and first vertical derivative, but no original raw data. The reduced-to-pole magnetic image revealed a 5 km long arc shaped zone of magnetic highs and lows, bordered to the west by a strong magnetic low anomaly. The low corresponded to the outer western ring of a circular feature with an 8 km radius.

9.3.5 Geochemistry – Stream Sediment Sampling

Rio Tinto reportedly completed a programme of reconnaissance stream sediment sampling over several drainage basins, results of which are shown in Table 9.2.

Table 9.2: Rio Tinto Stream Sediment Sampling

Prospect	Number of Samples	Range of Grades (g/t Au)
Colnic	24	0 to 1.04
Ciresata	13	0 to 0.43
Total	**37**	

No details were provided regarding sample collection methodology, sample size, or preparation. Rio Tinto followed up this first-stage work with soil geochemistry and rock-chip sampling in all these drainage basins. Further work by ESM has superseded these results.

9.3.6 Soil Geochemical Sampling

Rio Tinto reportedly completed a programme of grid soil sampling over three separate grids. Few details were provided regarding sample collection methodology, sample size or preparation.

At the Ciresata Prospect, a soil geochemistry survey covering 11.2 km^2 was completed at a spacing of 200 m × 100 m, widening to 200 m × 200 m on the southeast part of the grid. Results from this survey defined partly-coincident Au (> 10 ppb), Cu (> 25 ppm), and Mo (> 2 ppm) anomalies. In addition, over a prominent magnetic anomaly from geophysical data, there are coincident Au–Cu–Mo soil anomalies that cover an area of approximately 1,200 m × 600 m. Within this anomaly, Minexfor reportedly drilled four vertical diamond drillholes.

Rio Tinto's soil geochemical grid at Colnic covered an area of approximately 1.2 km × 1.4 km and lines were oriented north–south, spaced 200 m apart with sampling stations every 100 m. In the western part of the grid, line spacing is 100 m. Soil samples were sieved to minus 80 mesh and analysed by OMAC Laboratory in Ireland for Au and a 45-element suite using inductively coupled plasma optical emission spectroscopy.

9.3.7 Rock Chip Sampling

Rio Tinto collected 153 rock chip samples at the Colnic and Ciresata Prospects, as shown in Table 9.3. This limited reconnaissance-level work has been superseded by surface exploration work subsequently completed by ESM.

Table 9.3: Rio Tinto Rock Chip Sampling

Prospect	Number of Samples	Range of Grades (g/t Au)
Colnic	133	0 to 1.24
Ciresata	14	0.01 to 0.21
Total	**147**	

9.3.8 Drill Core Sampling

Rio Tinto did not report any drilling activities in their exploration reports filed with the NAMR and subsequently purchased by ESM.

9.4 ESM (2004 TO 2012)

9.4.1 Geographic/Grid Control

ESM established five grids as shown in Table 9.4. These grids formed the basis of soil sampling surveys and ground geophysical surveys. These grids are generally of temporary nature with only baselines marked with monuments. Subsequent geological mapping and channel sampling were located with government topographic maps, compass and tape surveys, and handheld GPS units without the utilisation of a base station.

Table 9.4: ESM Grids

Prospect	Area	Grid Spacing	Purpose
North part of Rovina Licence	24 km^2	50 m E-W × 50 m N-S	Ground magnetic survey
North part of Rovina Licence	20 km^2	200 m E-W × 100 m N-S	Soil geochemistry survey; includes 100 m × 100 m infill grids over the Rovina and Colnic Deposits
Ciresata	9 km^2	200 m E-W × 100 m N-S	Soil geochemistry
Ciresata	10 km^2	50 m E-W × 50 m N-S	Ground Magnetic survey

9.4.2 Topography

The locations and elevations of all geological mapping, channel samples, and drillhole collars are plotted on 1:5,000 scale government topographic maps with 10 m elevation contours. Locations of isolated surface samples are obtained through use of handheld GPS units with detailed surface samples and drill-hole collars located utilising compass and tape surveying. A professional contract survey company (Belevion Geo-Topo SRL) was utilised to complete and regularly update topographic surveys over Rovina, Colnic, and Ciresata using a total-station instrument. In addition, drill-hole collars and access roads are surveyed by Belevion. From 2010, ESM utilised another professional contract survey company (Topo-Geo Plus). Locations for all data are in the Romanian "Stereo70" grid coordinates.

9.4.3 Geological Mapping and Related Studies

A regional reconnaissance mapping programme at a scale of 1:5,000 was completed over most of the licence, together with a more detailed mapping at a scale of 1:1,000 over the main prospects.

A grid for the mapping and sampling programme was not physically established but existing topography maps and handheld GPS units were used for control.

The geological work carried out by ESM geologists and contractors includes the following:

- Colnic/Rovina Deposits: 1:1,000 scale geology covering an area of 3.9 km^2 focusing on the immediate Rovina and Colnic target areas. Mapping was completed by independent contractor Steve Priesmeyer from A.C.A. Howe International Ltd in 2005 and re-mapped by ESM Geologist Mr Jim Stemler in 2006.
- Ciresata Deposit: 1,000 scale geology covering an area of 4 km^2 in the Ciresata area. Mapping was completed by independent contractor Steve Priesmeyer from A.C.A. Howe International Ltd in 2005 and re-mapped by Mr Jim Stemler in 2007.
- Rovina Licence: 1:2,000 scale geology mapping targeting ground magnetic anomalies within a 24 km^2 area in the Northern part of the Rovina Licence. Mapping was completed by independent contractor Steve Priesmeyer from A.C.A. Howe International Ltd in 2006 and re-mapped by Mr Jim Stemler at 1:5,000 scale in 2007.
- Mapping of drill roads at Rovina and Colnic at 1:1,000 scale. Mapping was completed by ESM geologists in 2008.
- Mapping of lithology and alteration in the Ciresata area by Barrick gold geologists.

Much of the relevant data resulting from this work is reported in Sections 7 and 9.

9.4.4 Remote Sensing and Satellite Imagery

In July 2002, ESM purchased 1:100,000 scale Satellite LandSat TM imagery from HME Partnership Ltd, Kent, U.K., which cover the entire Golden Quadrilateral. The data is integrated into a GIS database and has been used to aid in the structural interpretation of the property and for alteration mapping.

9.4.5 Ground Geophysics

In June 2006, ESM commissioned Belevion SRL from Bucharest to complete a ground magnetic survey totalling 480-line kilometres and covering a 24 km^2 area over the Colnic and Rovina deposits. The survey consisted of east–west-oriented, 50 m spaced lines, with individual stations spaced at 50 m along each line. A summary report prepared by Belevion staff included a total field image. The raw data has been subsequently re-processed and interpreted to evaluate reduced to pole and analytical signal features for interpretation (Morris, 2006). The results of the survey are shown in Figure 9.1 and show several prominent, strongly positive, anomalies.

In October 2007, ESM commissioned Belevion SRL from Bucharest to complete a ground magnetic survey over the Ciresata area. This survey covers 10 km^2 at a 50 m × 50 m grid spacing. All the field maps provided by Belevion were utilised for interpretation.

The known porphyries of Rovina, Colnic, and Ciresata demonstrate a “bulls-eye” feature resulting from a magnetic porphyry core and a surrounding magnetic low, possibly relating to magnetite destructive retrograde phyllic alteration. Geological mapping and magnetic anomaly “ground truthing” indicated the presence of several magnetic units including lava flows and volcaniclastic rocks and subvolcanic intrusives with primary magnetite. Several anomalies are present within the coupled magnetic high and adjacent low of the “bull’s eye” pattern. ESM has completed several soil geochemistry surveys over target areas in the north part of the property. Soil geochemistry results coupled with magnetic anomalies are used for guiding field reconnaissance and mapping programmes. This method resulted in the identification of subcropping potassic alteration with associated copper mineralisation at the Zdrapti Prospect and was instrumental in discovering Ciresata. As part of an alteration mapping programme at Ciresata, the prominent high-magnetic anomaly was found to be caused by intense magnetite alteration. Subsequent work has shown this to be the barren magnetite altered cap to the Ciresata mineralisation 50 m to 150 m below the surface (see Figure 9.2). In 2010, the raw magnetic data from both the Rovina-Colnic and Ciresata grids was re-processed and interpreted by Barrick Gold (Hope et al. 2010) utilising proprietary filters to highlight porphyry targets.

In September 2006, ESM contracted Belevion SRL to complete an IP/resistivity survey over the Rovina, Colnic, and Zdrapti target areas based on magnetic signature and the occurrence of known surface mineralisation and early-stage drilling results. The objective of these surveys was to provide drill-targeting guidance for the definition of the targets.

In May 2008, ESM contracted Belevion SRL to complete a IP/resistivity survey over the Ciresata target area following encouraging initial drilling results. This survey was centred on the barren magnetitic anomaly. The surveys utilised an IPC7–2.5 kW SCINTREX transmitter, and an IPR12–SCINTREX receiver. Measurements were collected every 20 m along lines of length variable from 1,000 m to 1,500 m. The theoretical depth penetration of the surveys varies from 180 m to 310 m (see Table 9.5). The IP survey layout and interpretation to date of the results from the Rovina survey were performed by an independent consulting geophysicist, Paolo Costantini, who has visited the projects and advised on-line orientation. A 3D interpretation of the results from the Rovina and Zdrapti surveys was completed. At Colnic and Ciresata, ESM has utilised pseudo-sections of inverted data and interpreted by Belevion. The results for Ciresata were inconclusive likely due to limited depth penetration.

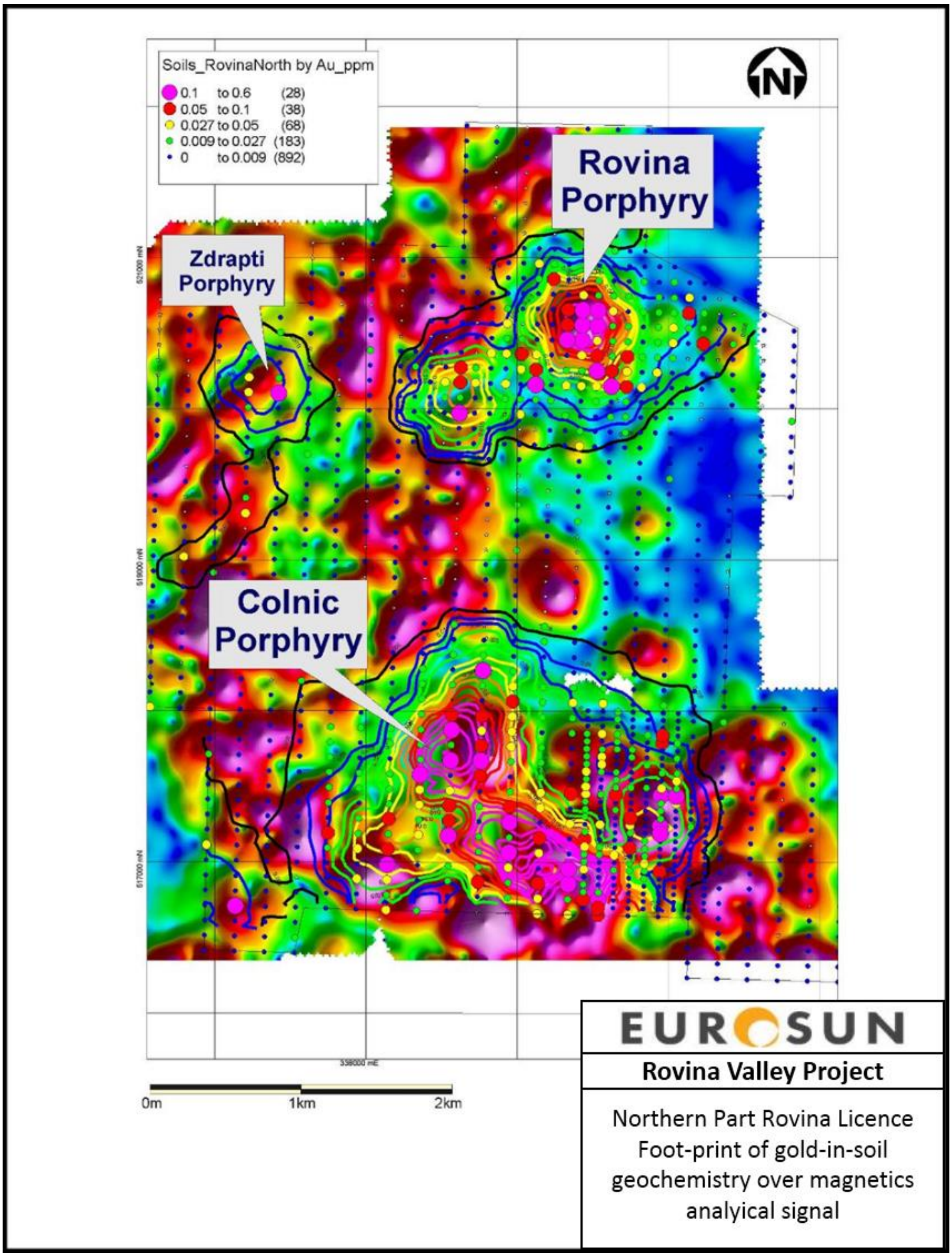


Figure 9.1: Ground Magnetic Survey and Soil Geochemistry, Northern Part of Licence

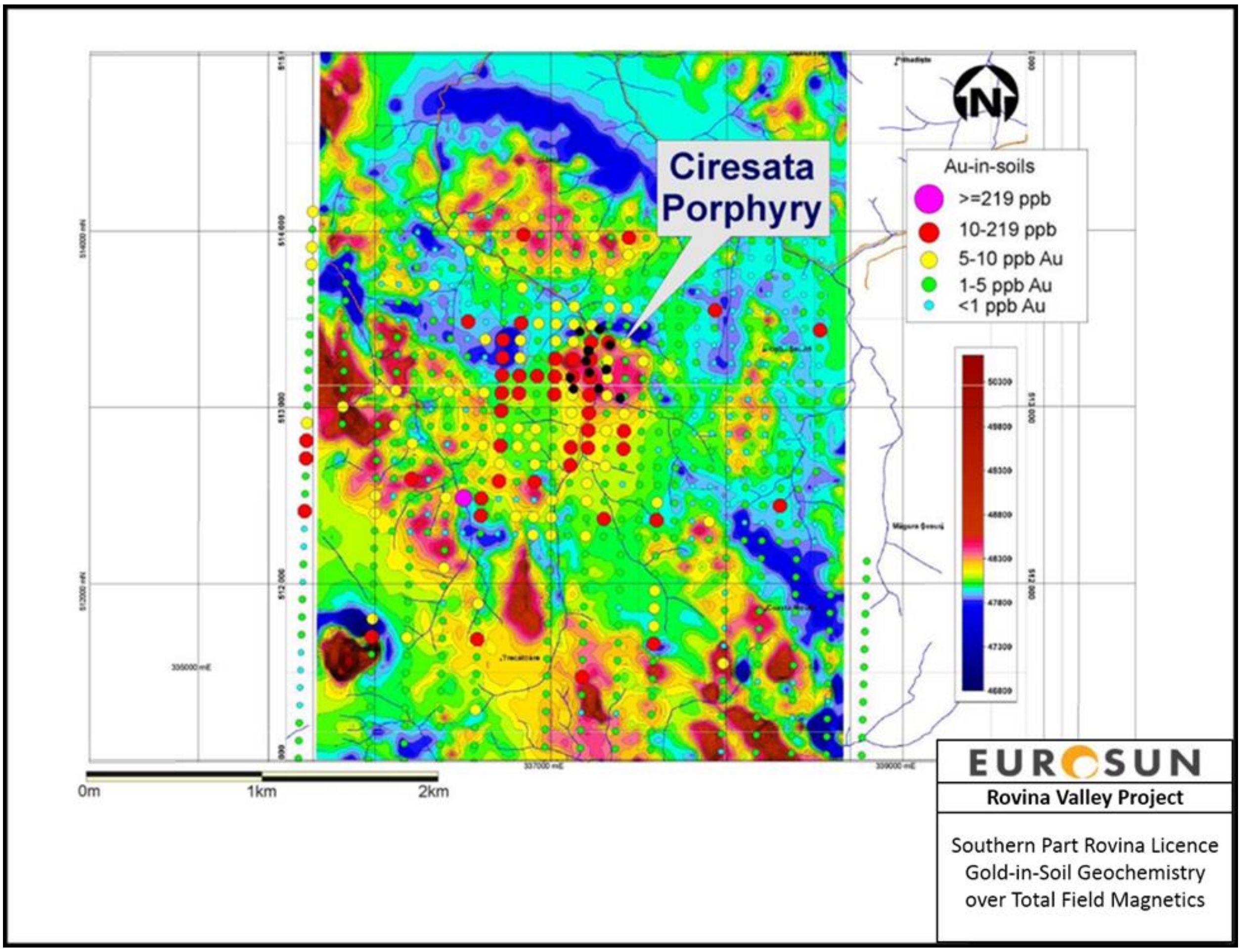


Figure 9.2: Ciresata Area Ground Magnetic Survey and Soil Geochemistry, Southern Part of Licence

Table 9.5: ESM IP/Resistivity Surveys

Prospect	Survey Dates	Amount (line m)	Depth of Penetration (m)	Comments
Zdrapti	September, 2006	4,200	180	3 lines, 1,400 m long, lines spaced 150 m apart
Colnic	Oct 16 – Nov 3, 2006	4,370	180–310	2 lines along drill sections and 1 line at 90° to sections over the Colnic deposit
Rovina	March – April, 2007	7,500	310	6 lines, 1.25 km long, lines spaced 125 m apart
Colnic	April – May, 2007	9,000	310	6 lines, 1.5 km long, spaced 300 m apart
Ciresata	May 2008	3,500	200	2 lines, 1.74 km each, in orthogonal cross-pattern
Total		**28,57**		

9.4.6 Soil Geochemical Sampling

In May 2007, ESM initiated a soil geochemical survey over the northern part of the Rovina licence. The survey grid covered an area of 20 km^2 over the Rovina, Colnic, and Zdrapti prospects on a 200 m (east–west) × 100 m (north–south) grid. In addition, infill grids were completed in selected areas with the larger grid. Results highlight coincident Au-Cu ± Mo anomalies over the known porphyry deposits of Rovina and Colnic with a series of satellite gold anomalies within the broad phyllic alteration halo at Colnic (see Figure 9.1). Additional anomalies occur at the Zdrapti prospect and other areas. ESM conducted field-follow-ups comprised of mapping and sampling of these and other magnetic anomalies within the grid.

ESM conducted a geochemistry re-survey of the Ciresata area at a closer grid spacing than the Rio Tinto survey to provide better sampling and analytical control. Results from ESM's 100 m × 100 m geochemistry survey highlight an Au + Cu anomaly extending 300 m west from the known porphyry mineralisation.

9.4.7 Rock Chip Sampling

A total of 1,538 surface rock samples were collected from the Rovina property as part of reconnaissance mapping, prospect mapping, and detailed mapping campaigns. These samples include chip-channel, chip, float, waste-dump, and discarded old core. Geologist descriptions include sample type, geology description, and map location and are entered into a GIS database. A tabulation of surface rock sampling by prospect for the Rovina property is shown in Table 9.6. This includes 294 channel-chip samples from outcrops at Colnic for a cumulative 794 m, and 83 channel-chip samples from Rovina for a cumulative 303 m collected during the period of October 2004 to May 2006. At Ciresata, rock sampling includes 109 rock samples and 34 channel samples collected in the period between 2006 and 2008, an additional 558 composited chip rock samples collected in 2010, and 126 rock samples collected by Barrick Gold geologists in 2011. Geologic description and assay results from these samples highlighted outcropping porphyry Au-Cu mineralisation that subsequent drilling has defined as the Rovina and Colnic porphyry deposits. These samples have not been used in this mineral resource estimation due to the extensive drilling completed but do provide surface evidence of the respective porphyry mineralisation.

Table 9.6: Surface Rock Sampling on the Rovina Property

Prospect	Rock Samples (from Outcrops, Subcrops, Floats, Mine Dumps)	Channel Chip Samples, Channel Samples, and Composite Chip Channel Samples	Underground Channel Samples	Total
Rovina	98	83		181
Colnic	89	294		383
Ciresata until 2008	109	34		143
Ciresata 2009-2012	126	558		684
Zdrapti	18			18

9.4.8 Drill Core Sampling

ESM has extensively drilled the deposit since 2006. Details of this drilling are discussed in Section 10 (Drilling).

9.4.9 Mineralogical and Petrographic Studies

A number of petrographic studies were commissioned in 2006, 2007, 2008 and 2011 on hand specimen and drill core samples to Dr Georghe Damian at North University in Baia Marie, to Dr Robin Armstrong of the Natural History Museum in London UK, and to Jim Clarke of Cygnus Consulting Inc. in Montreal, Canada. Jon Sliwinski from X-Strata Process Support Centre conducted high-resolution QEMSCAN analyses on drill core samples from Ciresata and Colnic. Samples were examined for lithology, alteration, paragenetic sequences, and mineralisation. Results of the studies were provided as detailed descriptions and photo-micrographs in report format (Damian, 2006 in Ruff, 2006, and Armstrong, 2006 in Ruff, 2007, Ruff et al., 2012).

9.5 ESM EXPLORATION (2012 – CURRENT)

Exploration activities on the Rovina Licence property was halted in 2012, as required by the process of conversion to a Mining Licence.

In October–December 2020, ESM conducted geotechnical drilling at the Colnic and Rovina deposits, as part of a geotechnical study for the Definitive Feasibility Study. Three holes were drilled at Colnic and one hole was drilled at the Rovina deposit and another twelve holes in the WMF and plant areas, totalling 1,311.70 m.

All drillholes were cored using a HQ-sized bit. The three inclined holes at the Rovina and Colnic deposits were drilled using a core orientation tool, plus a downhole televiewer survey. Core from these three holes was used to conduct confined and unconfined uniaxial strain tests, plus Brazilian tensile disk tests.

9.6 CCIC MINRES COMMENT

It is CCIC MinRes's opinion that the exploration efforts conducted by ESM are sufficient for the porphyry style of mineralisation. Historical drill samples by Minexfor were not used in the mineral resource estimation. Recent surface rock chip samples collected by ESM were used in guiding the exploration drilling activities; however, none of these were used in the mineral resource estimate, because ESM has drilled sufficient diamond core holes that cover the deposits.

10 DRILLING

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

Approximately 165,174 m of drilling has been completed on the property since 1975. Most of the drilling focused on the Colnic, Rovina, and Ciresata deposits (41,217 m, 64,851 m, and 56,430 m, respectively). However, smaller reconnaissance programmes have been drilled by ESM at the Zdrapti Prospect. A summary of all drilling is provided in Table 10.1.

Locations of the historical drillholes at the Colnic, Rovina, and Ciresata deposits are shown in Figure 6.1, Figure 10.1, and Figure 10.2, respectively. The ESM drilling at Rovina, Colnic and Ciresata are shown in Figure 10.4, Figure 10.6, and Figure 10.7, respectively.

Table 10.1: Summary of Drilling on the Property

Prospect	Company	Year(s)	DDH Holes	Total (m)	Core Diameter
Colnic	Minexfor	1975	1	650*	Diameter unknown
	Minexfor	1982	5	2,990*	Diameter unknown
	Minexfor	2000	8	1,100*	Diameter unknown
	ESM	2006	49	15,714	72 % HQ, 28 % NQ
	ESM	2007	39	13,635	59 % HQ, 41 % NQ
	ESM	2008	1	270	Metallurgical Drillhole HQ
	ESM	2011	18	4,645	76 % HQ, 24 % NQ2
	ESM-Barrick	2012	2	1,217	4 % PQ, 54 % HQ, 42 % NQ2
	ESM Geotech	2011-12	4	996	100 % HQ
	ESM Geotech	2020	3	798	100 % HQ
Total Colnic			**133**	**42,015**	
Rovina	Minexfor	1975–86	34	23,119	Diameter unknown
	ESM	2006	17	8,435	40 % HQ, 60 % NQ
	ESM	2007	34	15,644	40 % HQ, 60 % NQ
	ESM	2008	16	7,625	43 % HQ, 57 % NQ (Includes 1 Metallurgical Hole HQ)
	ESM	2011	4	2,113	43 % HQ, 57 % NQ2
	ESM	2012	15	5,920	43 % HQ, 57 % NQ2
	ESM-Barrick	2012	2	851	75 % HQ, 25 % NQ
	ESM Geotech	2011-12	4	1,144	95 % HQ; 5 % NQ2
	ESM Geotech	2020-21	1	351.50	100 % HQ
Total Rovina			**131**	**65,203**	
Ciresata	Minexfor	2002-03	6	1,200*	Diameter unknown
	ESM	2007	2	552	40 % HQ, 60 % NQ
	ESM	2008	14	7,183	35 % HQ, 65 % NQ (Includes 1 Metallurgical Hole HQ)

Prospect	Company	Year(s)	DDH Holes	Total (m)	Core Diameter
	ESM	2010	4	3,793	21 % HQ, 79 % NQ
	ESM (includes 3 holes ESM-Barrick)	2011	44	36,159	25 % HQ, 75 % NQ2
	ESM-Barrick	2011-12	11	7,543	6 % PQ, 36 % HQ, 5 8% NQ2
Total Ciresata			**81**	**56,430**	
Zdrapti	ESM	2007	11	2,671	(RB-57), 41 % HQ, 59 % NQ
Geotechnical (conveyor belt, WMF, plant and TMF)	ESM ESM	2011-12 2020	21 12	3019.60 162.20	100 % HQ 100 % HQ
Total Property			**358**	**169,501**	
Core diameters: PQ = 85 mm; HQ = 63. 5 mm; NQ =47.6 mm, NQ2 =50.7 mm * Historical data, may be incomplete					

10.1 HISTORICAL MINEXFOR DRILLING (1974–2003)

10.1.1 Rovina Deposit

Few details are available on the historical drill programmes completed by Minexfor at Rovina. All core samples from previous campaigns were dumped in heaps near the Rovina deposit, and, therefore, cannot be resampled. The most significant programme was completed at the Rovina deposit between 1974 and 1986 (see Table 10.2).

Table 10.2: Minexfor Historical Drilling – Rovina

Hole ID	Stereo 70 Coordinates		Elevation (masl)	Length (m)	Azimuth	Dip	Average Core Recovery (%)
	Easting	Northing					
F-1	339,395.930	520,530.373	569.9	750	0°	-90°	N/A
F-2	339,544.001	520,573.579	627.99	1,108	0°	-90°	N/A
F-3	339,278.895	520,504.938	553.27	750	0°	-90°	N/A
F-4	339,113.547	520,494.034	571.24	750	0°	-90°	N/A
F-5	339,592.436	520,283.889	635.64	750	0°	-90°	N/A
F-6	339,696.495	520,602.333	681.58	750	0°	-90°	N/A
F-23	339,454.987	520,375.845	623.71	720	0°	-90°	N/A
F-41	339,611.239	520,679.343	635.24	650	0°	-90°	N/A
F-42	339,448.855	520,602.642	580.56	550	0°	-90°	N/A
F-43	339,487.212	520,544.766	616.09	550	0°	-90°	N/A
F-44	339,390.956	520,603.637	575.21	560	0°	-90°	N/A
F-45	339,529.848	520,637.297	605.8	546	0°	-90°	N/A
F-46	339,500.432	520,471.261	638.16	550	0°	-90°	N/A

Hole	Stereo 70 Coordinates		Elevation	Length	Azimuth	Dip	Average Core
F-47	339,439.521	520,457.703	612.88	575	0°	-90°	N/A
F-48	339,558.643	520,474.819	663.89	550	0°	-90°	N/A
F-49	339,341.018	520,504.800	548.42	700	0°	-90°	N/A
F-50	339,618.911	520,586.622	653.83	650	0°	-90°	N/A
F-51	339,306.152	520,583.488	596.25	800	0°	-90°	N/A
F-52	339,351.864	520,678.107	617.39	785	0°	-90°	N/A
F-53	339,420.278	520,701.746	616	900	0°	-90°	N/A
F-54	339,479.677	520,725.688	610.96	800	0°	-90°	N/A
F-55	339,332.935	520,429.126	554.01	560	0°	-90°	N/A
F-58	339,581.110	520,727.755	612.48	800	0°	-90°	N/A
F-59	339,690.427	520,685.909	674.65	650	0°	-90°	N/A
F-60	339,641.932	520,524.957	679.76	515	0°	-90°	N/A
F-62	339,520.845	520,410.476	661.43	650	0°	-90°	N/A
F-63	339,585.283	520,438.784	682.24	650	0°	-90°	N/A
F-64	NA	NA	654.7	650	0°	-90°	N/A
F-66	339,545.557	520,335.422	634.13	650	0°	-90°	N/A
F-67	339,608.958	520,352.685	644.04	650	0°	-90°	N/A
F-68	339,676.792	520,373.739	604.98	650	0°	-90°	N/A
F-70	339,550.562	520,281.563	617.11	650	0°	-90°	N/A
F-71	339,628.186	520,289.111	617.2	650	0°	-90°	N/A
F-72	339,701.934	520,310.237	597.04	650	0°	-90°	N/A
Total				**23,119**			

Minexfor drilled 34 core holes ranging in depth between 515 m and 1,108 m (average depth 680 m), for a total of 23,119 m. All holes were collared vertically, and nominally spaced between 60 m and 90 m along seven east–northeast-oriented, 100-m spaced grid lines. Drilling was very slow and averaged about 5 m/d over a 12-year period.

ESM was provided with paper copies of simplified drill logs showing generalised lithology and alteration, and 15 m composite assays for copper, reportedly derived from 1 m assays. Gold was only sporadically reported on the drill logs at 15 m intervals, reportedly derived from composited samples averaging 5 m. The drill logs also contain sporadic results for silver, lead, zinc, molybdenum, iron, and sulphur along with occasional specific gravity measurements. The header of each log also recorded the average recovery for the hole. Collars were reported surveyed with 3-decimal accuracy.

Figure 10.1 shows the location of the drill collar and exploration galleries at the Rovina deposit. CCIC MinRes notes that no historical drillholes were used in the mineral resource estimate because the deposit has been sufficiently re-drilled by ESM.

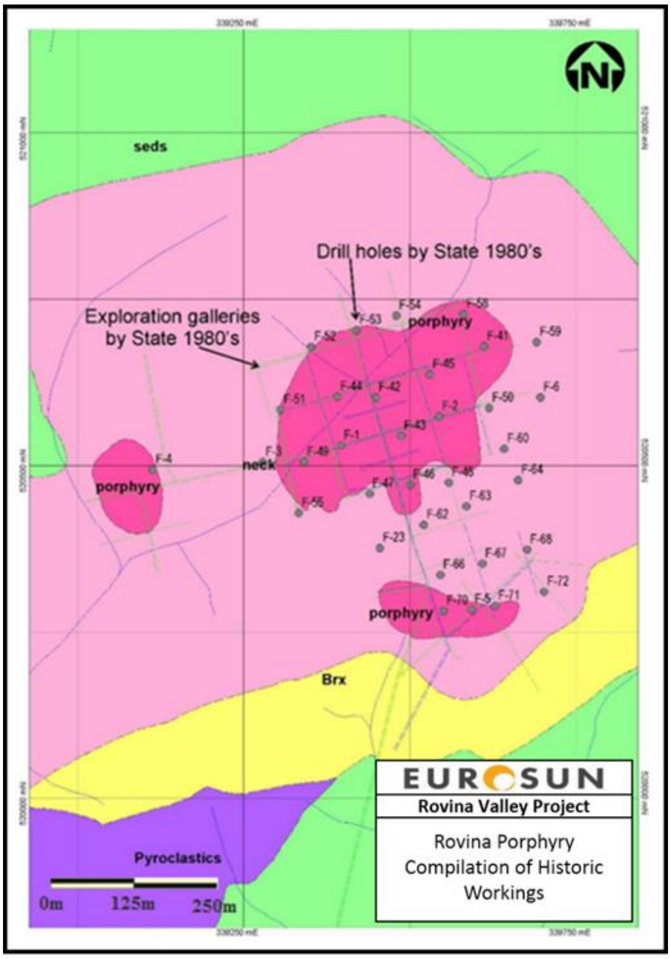


Figure 10.1: Minexfor Historical Drillholes at the Rovina

10.1.2 Colnic Deposit

Between 1975 and 2000, Minexfor drilled 14 holes totalling 4,740 m at the Colnic deposit (see Table 10.3). Few details were provided to ESM regarding this drilling campaign. Most of these holes were drilled in the same area as the ESM drilling at Colnic, and ranged in depth from 100 m to 1,200 m, averaging 340 m. Most holes were widely spaced (100 m to 500 m), and no specific grid pattern was used for the drilling. Holes were restricted to an area covering approximately 2,000 m × 800 m, targeting geophysical and surface geological targets.

The first hole drilled in the area (diamond drillhole F-2) was drilled in 1975 and was the first hole to intersect porphyry-style mineralisation at Colnic. The hole was collared vertically and continued to a depth of 650 m. This hole gives the name to the hill hosting the central-eastern part of the Colnic deposit (F-2 Hill).

Table 10.3: Minexfor Historical Drilling (1975-2003), Colnic

Hole IDr	Stereo 70 Coordinates		Elevation (masl)	Length (m)	Azimuth	Dip	Average Core Recovery (%)
	Easting	Northing					
F-2	338,737.4	517,772.3	463.34	650	0°	-90°	N/A
FA	338,974.4	518,093.1	361.13	1,200	0°	-90°	N/A
FB	339,027.5	518,320.2	367.1	650	0°	-90°	N/A
FD	339,368.1	518,077.4	390.31	490	0°	-90°	N/A
FF	338,607.7	518,273.3	367.01	650	0°	-90°	N/A
F7	338,725	518,063	358	200	255°	-5°	N/A
F8	338,571	517,885	355	200	248°	-15°	N/A
F9	338,673	517,945	350.5	200	235°	-45°	N/A
F14	338,327.61	517,300.6	338.89	100	0°	-90°	N/A
F15	338,487.29	517,598.5	343.13	100	0°	-90°	N/A
F16	338,703.86	518,172.9	366.39	100	288°	-72°	N/A
F17	339,067.88	518,027.7	373.07	100	99°	-60°	N/A
F18	338,623	517,919	353.8	100	0°	-90°	N/A
S2	?	?	?	Poorly documented			
F12	?	?	?	Poorly documented			
F13	?	?	?	Poorly documented			
			Total	**3541,2***			

Only a summary log for this Colnic drilling was provided to ESM, which documents weakly anomalous copper grades (averaging 400 ppm), between zero and 240 m depth, and stronger anomalies (averaging 1,000 ppm) from 240 m to 650 m depth. Gold was apparently not analysed.

Several years later, in the mid-1980s, Minexfor drilled core holes FA, FB, FD, and FF which were wide-spaced and located to the north and east of the ESM current drilling. These holes were collared vertically, and continued to depths ranging from 490 m to 1,200 m. Only summarised details of these holes were provided to ESM; however, long intervals of weakly to moderately anomalous copper grades (230 ppm to 710 ppm) were reportedly intersected, starting at depths generally greater than 300 m. Gold was apparently not analysed in these holes. For the holes drilled in 1975 and the early 1980s, ESM received only generalised descriptions of the drill logs.

In 2000, Minexfor completed eight additional core holes at Colnic (F7 to F9, and F14 to F18), totalling 1,100 m. These holes were drilled at angles ranging from -5° to -72° and at azimuths ranging between 235° and 288°, except for one hole drilled at an azimuth of 99°. The holes were all in the vicinity of ESM current drilling and intersected porphyry-style mineralisation. Gold values appear to be in a similar range to those reported by ESM, but copper grades were not included in the data provided. Two additional holes to the south (F14 and F15) returned only weak gold anomalies. No additional work was reported by Minexfor after this drilling. ESM purchased the drill log data with assays reported in paper format, which showed generalised lithology and alteration, and 1 m assays for gold and silver. Copper, lead, and zinc were only sporadically reported on the drill logs.

Collars were surveyed based on the 3-decimal accuracy of the collar coordinates in the database; however, the survey methodology was not documented. No holes from the Minexfor drill campaign at Colnic were used in the mineral resource estimate because the deposit has been sufficiently re-drilled by ESM.

10.1.3 Ciresata Prospect

Between 2002 and 2003, Minexfor drilled six core holes totalling 1,200 m at the Ciresata Prospect (see Table 10.4 and Figure 10.2), approximately 4.5 km south of Colnic. Few details were provided to ESM regarding the drilling campaign. No specific grid pattern was adhered to for the drilling, and holes were generally wide-spaced and presumably targeted geophysical anomalies. Hole lengths were all recorded as 200 m.

Table 10.4: Minexfor Historical Drilling (2002 to 2003), Ciresata

Hole IDr	Stereo 70 Coordinates		Elevation (masl)	Length (m)	Azimuth	Dip	Average Core Recovery (%)
	Easting	Northing					
F1	336,745.12	513,834.9	400.5	200	0	-90	N/A
F2	337,205.05	513,194.4	430.15	200	0	-90	N/A
F3	337,249.48	513,001.5	433.12	200	0	-90	N/A
F4	337,200.17	513,306.2	428.88	200	0	-90	N/A
F5	337,230.15	513,441.1	418.2	200	0	-90	N/A
F6	336,471.26	513,957.8	388.97	200	0	-90	N/A
			Total	**1,200**			

ESM purchased this data from the NAMR, and it consisted of paper drill logs of geology and reported gold assay results. ESM computerised this data for use in drillhole software. Few anomalous results were reported apart from weak scattered gold anomalies ranging between 0.25 g/t Au and 0.68 g/t Au throughout hole F4. The final 16 m of the drillhole (184 m to 200 m, the end of the hole) yielded a length-weighted average of 0.39 g/t Au. Only occasional copper analyses were completed and reported.

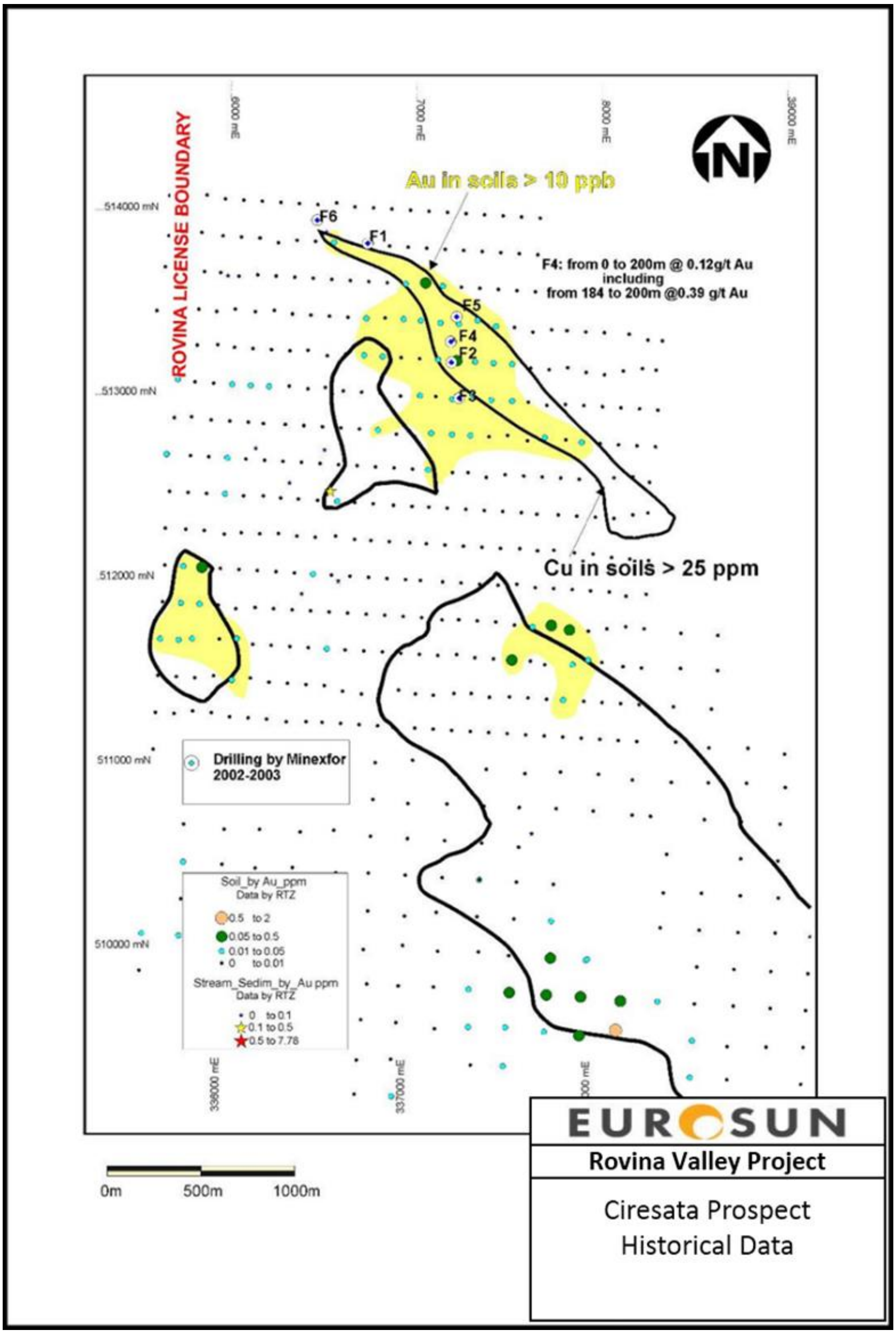


Figure 10.2: Minexfor Historical Drillholes and Rio Tinto Soil Geochemistry at Ciresata

10.2 ESM DRILLING

During site visits between 2008 and 2011, AGP observed core drilling and handling procedures and found them to be to industry standards. CCIC MinRes observed core drilling and handing operations during their November 2020 site visit. This drilling was for the geotechnical studies, by KCB consultants. CCIC MinRes also found the drilling and handling procedures to industry standards.

10.2.1 Drill Contractors

Drilling at the RVP was carried out by SC-Genfor S.R.L. (Genfor), a Romanian-based contractor. From 2006 to 2010, Genfor utilised two Warman-1000 rigs and a Longyear 45 rig, both with depth capabilities of around 600 m, and a B-57 drill rig capable of depths up to 300 m. Since Genfor was retained as drilling contractor, using a fleet of tophead drive-drill rigs on tracks from the Korean Hanjin Drilling Co. Ltd for drilling at the project. The following four rigs were used: Hanjin 4000 SD-RC (depth capability of around 500 m), Hanjin 6000SD-RC and HANJIN Doosan 6000 SD (both with depth capability of around 1,000 m) and Hanjin Doosan 7000SD (depth capability of 2,000 m).

10.2.2 Core Handling Procedures

Once drilled, core is removed from the core barrel by the drillers, washed and placed in galvanised steel core boxes. These boxes could cause some very low-level zinc contamination. No specific tests were conducted to see if this possible zinc contamination was occurring, but CCIC MinRes considers it extremely unlikely this process would induce a bias in the zinc resource model sufficient to interfere with the quality of the copper concentrates proposed for the project.

All boxes are clearly identified with the hole number, metres from/to, and box number written in permanent marker on the front. Individual drill runs are identified with small wooden blocks, where the depth (m) and hole number are recorded. Unsampled core is never left unattended at the rig.

Upon completion of drilling, holes are left open with a piece of PVC pipe is inserted into the hole, and the hole is clearly marked with a cement monument showing the hole number.

10.2.3 Collar Survey

After completion of each hole, the collars are surveyed by tape and compass method, and initially by GPS. ESM commissioned a detailed topographic survey of the property in November 2006, during which time all drillhole collars drilled prior to November were resurveyed using a total station. ESM have since contracted a surveyor to routinely survey the collar positions using a total station. CCIC MinRes comments that collar survey methodology is to industry standards.

10.2.4 Logging and Sampling

ESM owns a core shed in the town of Crişcior. This storage area is surrounded by a locked fence and monitored 24 h/d by security cameras and a guard. Core boxes are transported from the rig to this core logging facility under a geologists' supervision several times a day using a four-wheel-drive truck.

As soon as the core arrives at the core logging facility, it is checked to ensure that the core is packed properly, it is measured, logged for recovery and RQD, marked for sampling, and photographed. The core is then split with a diamond saw and both halves of the core are reinserted in the core box for sampling of the right-hand side of half-core, generally at 1 m intervals, observing important lithological boundaries. ESM has a rigorous process of double-checking all sample intervals and tags to avoid mislabelling samples. QA/QC samples are inserted in the sampling chain at that time.

The geologist logs the core using a fresh split face along with the surface generated by the drill bit. Logs are typically entered on paper that is subsequently transferred to a computerised database.

ESM geologists record the following information:

- Structure graphic log
- Veining
- Veining comprising intensity and thickness; orientation is recorded on holes that were oriented
- Lithology
- Alteration assemblage, intensity, mode of occurrence, and predominant alteration
- Mineralisation, comprising occurrences and percentages
- Magnetic susceptibility (selectively)

Once logged and sampled, the core is moved to a permanent storage facility located 1.5 km from the field office. The new core drilled under ESM supervision is in good condition. All drill cores are stored in covered core racks (see Figure 10.3) and can be easily retrieved.

Figure 10.3: Permanent Core Storage Facility

In general, core recoveries obtained by the drilling contractor have been very good, exceeding 97 %, except in localised areas of faulting or fracturing. Owing to the high average recoveries, no intervals were excluded from the database for use in exploratory data analysis (EDA), compositing, and interpolation as part of mineral resource estimation.

10.2.5 Downhole Survey

Downhole surveys are systematically conducted at approximately 50 m intervals along each hole using a Reflex EZ-Shot system. The EZ-Shot instrument uses a compass to record the azimuth of the drillhole; these types of instruments are sensitive to the presence of magnetic minerals in the rock such as magnetite and pyrrhotite. For that reason, the results of the surveys should be interpreted with care. From March 2011 onward, ESM surveyed all holes using a Reflex Gyroscope system. The Reflex Gyro instrument has the advantages of conducting surveys without being influenced by magnetic rocks and provides directional data (azimuth and dip) plus twelve parameters that are continuously recorded throughout the survey to track the path of the drillholes. ESM used a configuration that recorded directional data every 5 m downhole. Two different software programmes were used to record data according to the dip of the hole: one for holes dipping between -40° and -80°, and one for subvertical holes dipping steeper than -80°. Each hole was surveyed at collar using a total station by a contractor survey team to set the initial parameters (azimuth and dip) on which subsequent readings from the Gyro are based.

CCIC MinRes reviewed the downhole survey traces in 3D prior to check for abrupt azimuth or dip changes that might suggest the presence of false deviations from magnetic interferences or inappropriately collected readings.

ESM corrects all azimuth reading by adding 4°E to the magnetic azimuth read by the downhole instrument to account for compass magnetic declination.

10.2.6 Dry Bulk Density Measurements

ESM collected 1,125 specific gravity measurements from drill core. A total of 412 samples were collected from Rovina, 368 from Colnic, and 345 samples from Ciresata. Samples for specific gravity determination were taken at downhole intervals of between 10 m and 50 m, both in mineralised and waste rocks. The samples were sent to the ALS Laboratory at Gura Rosiei, where all samples were dried, coated in a thin layer of lacquer or shellac, then weighed in air (W1) and in water (W2). The specific gravity is calculated using the following formula:

$$\frac{W1}{(W1-W2)}$$

The volume of shellac or lacquer is too small to significantly affect the density determination, so no correction was required.

No adjustment was applied for variation in water temperature as the measurements were taken indoors at the laboratory where temperature fluctuations were minimal and, therefore, not significantly affecting the final determination.

The rock types found at RVP are generally non-porous, and CCIC MinRes, therefore, believes the specific gravity determinations are representative of the in-situ bulk density of the rock types and reliable for resource estimation purposes.

10.2.7 Rovina Deposit

ESM drilling programme at the Rovina deposit commenced on 26 May 2006. Drilling has been performed on 12, nominally southeast-oriented (150°), 55 m to 85 m spaced sections, over an area of approximately 550 m × 700 m (see Figure 10.4). Appendix A in the AGP PEA NI 43-101 2019 Report summarises the holes drilled by ESM.

Core drilled at the Rovina deposit is collared using HQ diameter and reduced to NQ or NQ2 (47.6 mm and 50.7 mm, respectively) at depths ranging between 124 m and 286 m. Currently, 40 % of the drilling is with HQ diameter and 60 % is with NQ or NQ2.

Most of the holes (60 %) were collared vertically and drilled to depths ranging between 100 m and 660 m, averaging approximately 474 m.

In 2012, as part of the ESM-Barrick exploration collaboration, two infill diamond drillholes (RRD-84 and RRD-85) were completed at Rovina, totalling 451 m (core size 75 % HQ and 25 % NQ). These holes were completed after the resource data cut-off and were not included in the current resource estimate. These holes are highlighted in blue in Figure 10.4. CCIC MinRes reviewed these two holes against the mineral resource block model for Rovina (see Figure 10.5) and is satisfied that there is no risk of overestimating the mineral resources.

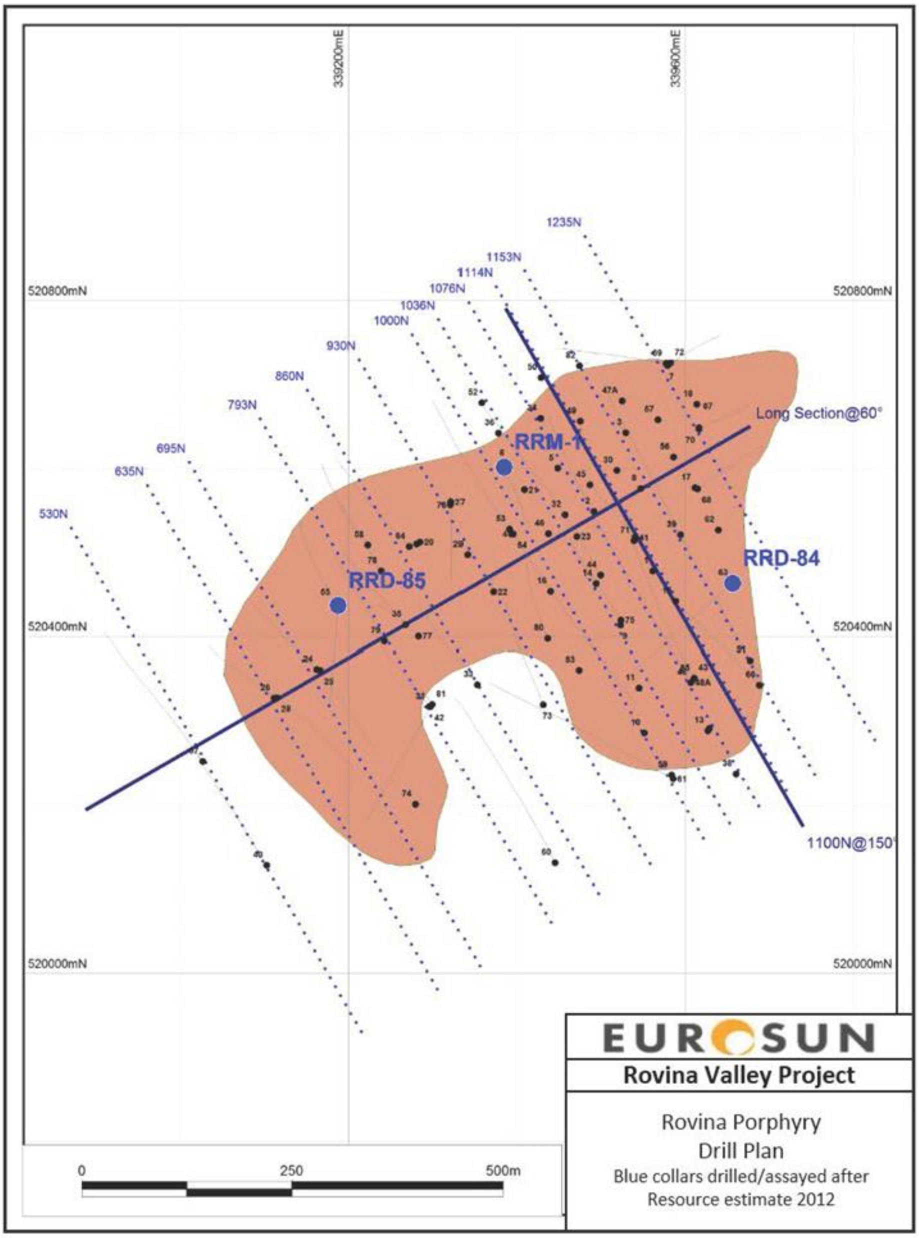


Figure 10.4: Rovina Porphyry Drill Plan

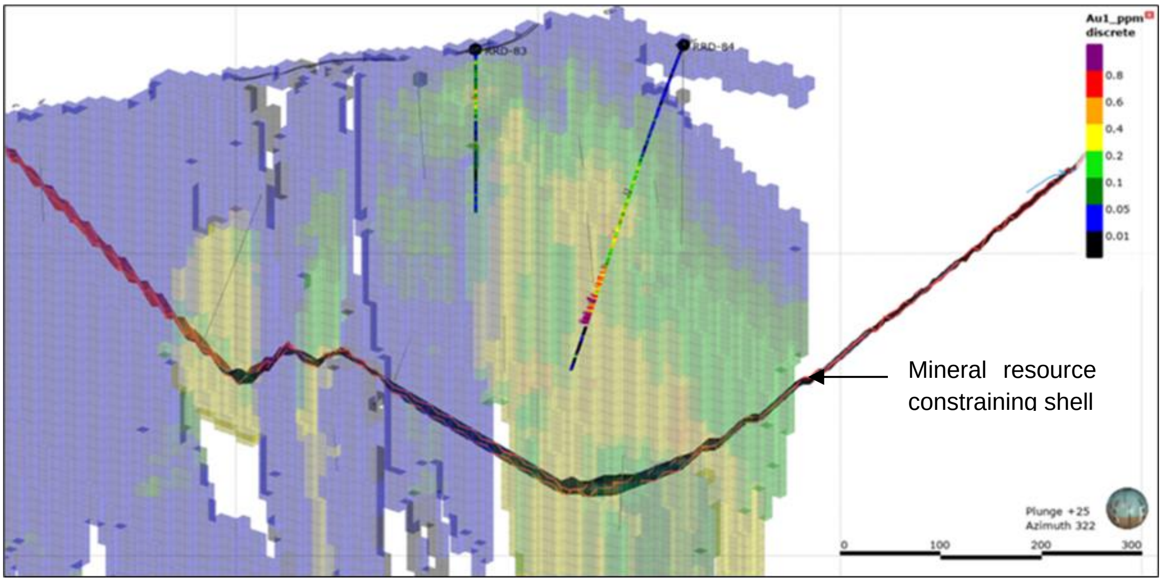


Figure 10.5: Section of Rovina Block Model showing drillholes RRD-84 & RRD-85, Coloured on Au g/t

10.2.8 Colnic Deposit

ESM commenced drilling at Colnic on 7 February 2006. As shown in Figure 10.6, drilling has been performed on 11 nominally southwest-oriented (235°), 30 m to 85 m spaced sections over an area of approximately 700 m (northwest) by 600 m (northeast). Drilling was also carried out on a perpendicular set of six sections, 60 m to 80 m spaced, nominally southeast-oriented (145°), over the same drill area. Appendix A in the AGP PEA NI 43-101 2019 Report summarises the holes drilled by ESM.

Core drilling at Colnic during the 2006 drill programme was undertaken by Genfor using track-mounted GNK 850, RB-57, and RB-58 drill rigs, each with depth capabilities of approximately 600 m. During the 2011–2012 programme, Genfor used the following four Hanjin rigs: Hanjin 4000 SD-RC, Hanjin 6000 SD-RC and Hanjin Doosan 6000 SD and Hanjin Doosan 7000 SD.

Core diameter is generally HQ (63.5 mm), and some holes continue to depths of 435 m using this diameter core. Holes are reduced to NQ or NQ2 core diameter at depths ranging between 105 m and 435 m, as required, but are typically reduced at a depth of approximately 200 m. Currently, 69 % of the drilling is with HQ diameter, and 37 % is with NQ or NQ2.

Most holes are inclined -50° to -70° and oriented at 220° to 260°. Holes have been drilled to depths ranging between 100 and 550 m, and average approximately 320 m.

In 2012, as part of the ESM-Barrick exploration collaboration, one infill diamond drillhole (RCD-106) and one exploration hole 500 m NE of the Colnic deposit (RCD-105) were completed at Colnic, totalling 1,217 m (core size: 4 % PQ, 54 % HQ, and 42 % NQ2). These holes were completed after the resource data cut-off and were not included in the current resource estimate. These holes are highlighted in blue in Figure 10.6. CCIC MinRes reviewed of these holes and established that RCD-106 was drilled around the periphery to improve geological definition and is approximately 150 m outside of the mineral resource constraining pit shell. CCIC MinRes is, therefore, satisfied that the exclusion of these holes will have negligible (if any) impact on the mineral resource estimate.

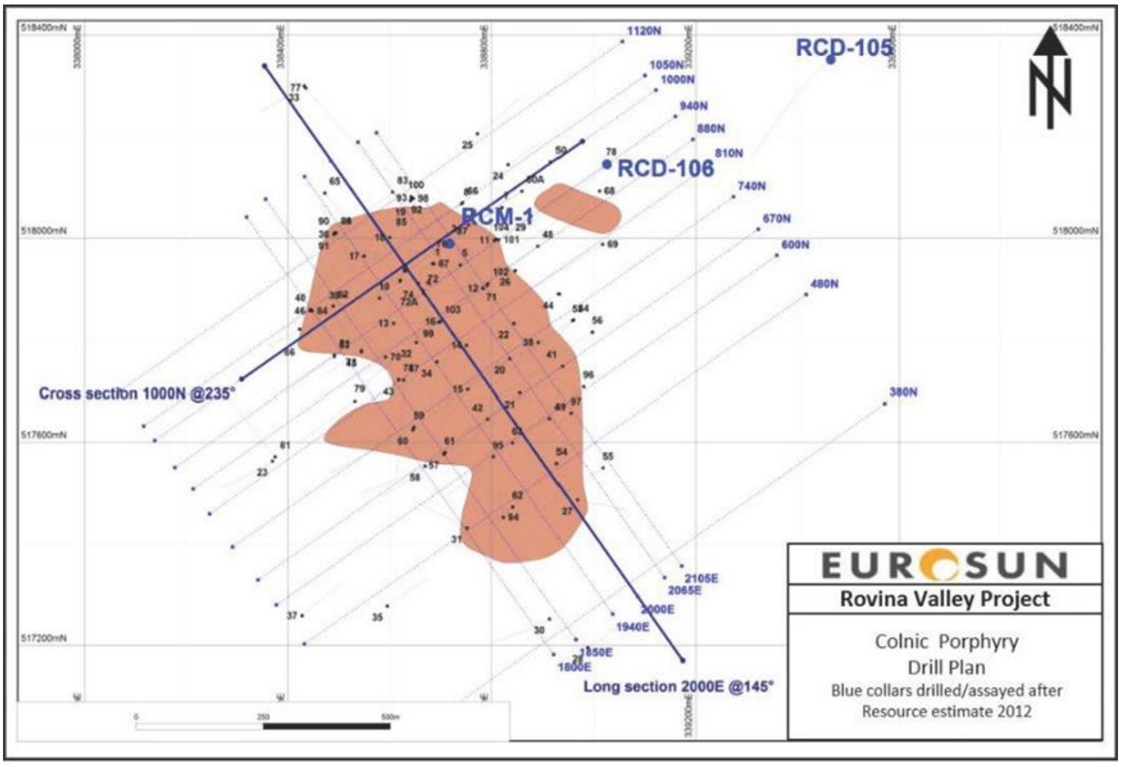


Figure 10.6: Colnic Porphyry Drill Plan

10.2.9 Ciresata Deposit

ESM drilling programme at the Ciresata deposit commenced in December 2007. Drilling has been performed on 15 nominally northeast-oriented (50°), 30 m to 130 m spaced sections (but typically 40 m apart), over an area of approximately 600 m × 600 m (see Figure 10.7).

Appendix A in the AGP PEA NI 43-101 Report summarises the holes drilled by ESM. Core drilling at the Ciresata deposit was undertaken by Genfor mainly using an RB-57 rig and a Longyear 45 rig, with depth capabilities of approximately 300 m and 600 m, respectively. During the 2010-2011 programme, Genfor used the same four Hanjin rigs.

Core drilled at the Ciresata deposit is collared using HQ diameter and reduced to NQ or NQ2 at depths ranging between 102 m and 301 m, but typically reduced at a depth of approximately 200 m. Currently, 33 % of the drilling is with HQ diameter and 67 % is with NQ or NQ2.

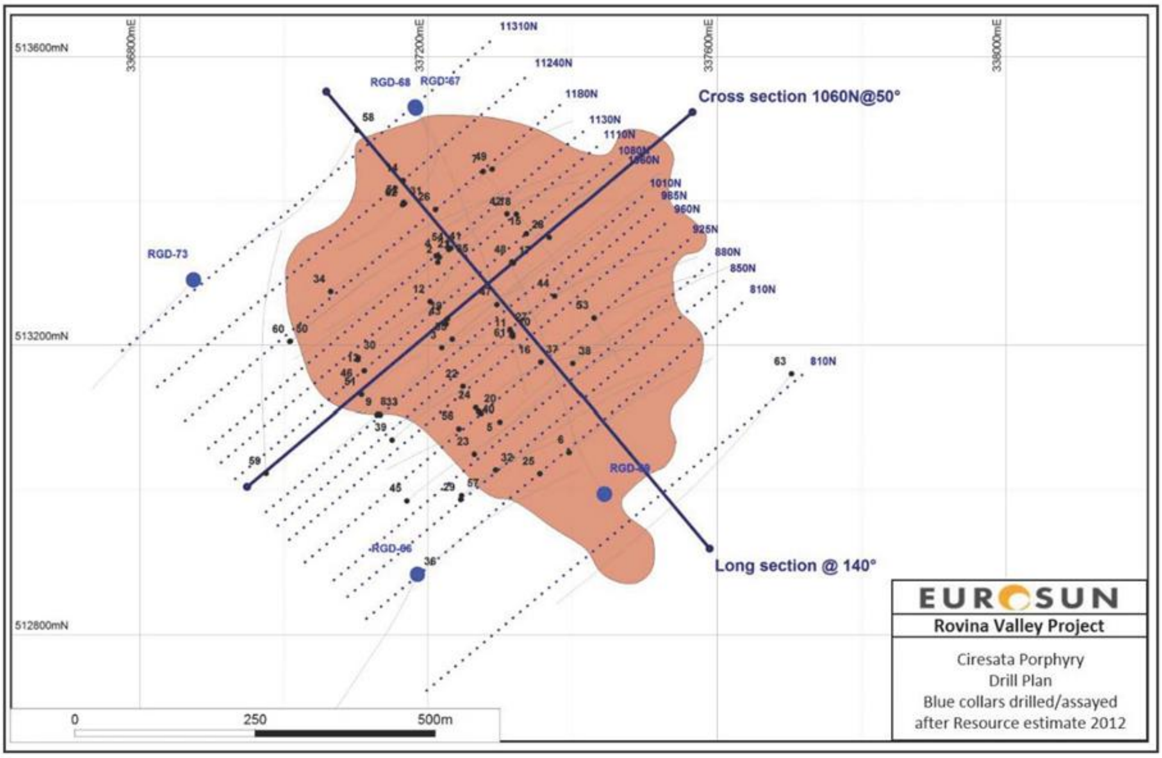


Figure 10.7: Ciresata Porphyry Drill Plan

In 2011–2012, as part of the ESM-Barrick exploration collaboration, 14 diamond drillholes were completed at and around the Ciresata deposit and at the Ciresata-hill prospect exploration site, for a total drill programme of 10,472 m. Of these 14 holes, three were incorporated in the current resource estimate (RGD-58, RGD-59, and RGD-63, a combined total of 2,930 m, and an average depth of 977 m). The other 11 holes, totalling 7,543 m in depth, were completed after the resource data cut-off and were not included in the current resource estimate. The average depth of these 11 holes was 750 m, with two holes (RGD-68 and RGD-69) testing the Ciresata deposit to an average depth of 1,500 m. Five of these holes are in and around the deposit and are highlighted in blue in Figure 10.7. The other six holes were drilled to test exploration targets in the south-west of the Ciresata porphyry at the Ciresata Hill Prospect (see Figure 10.7).

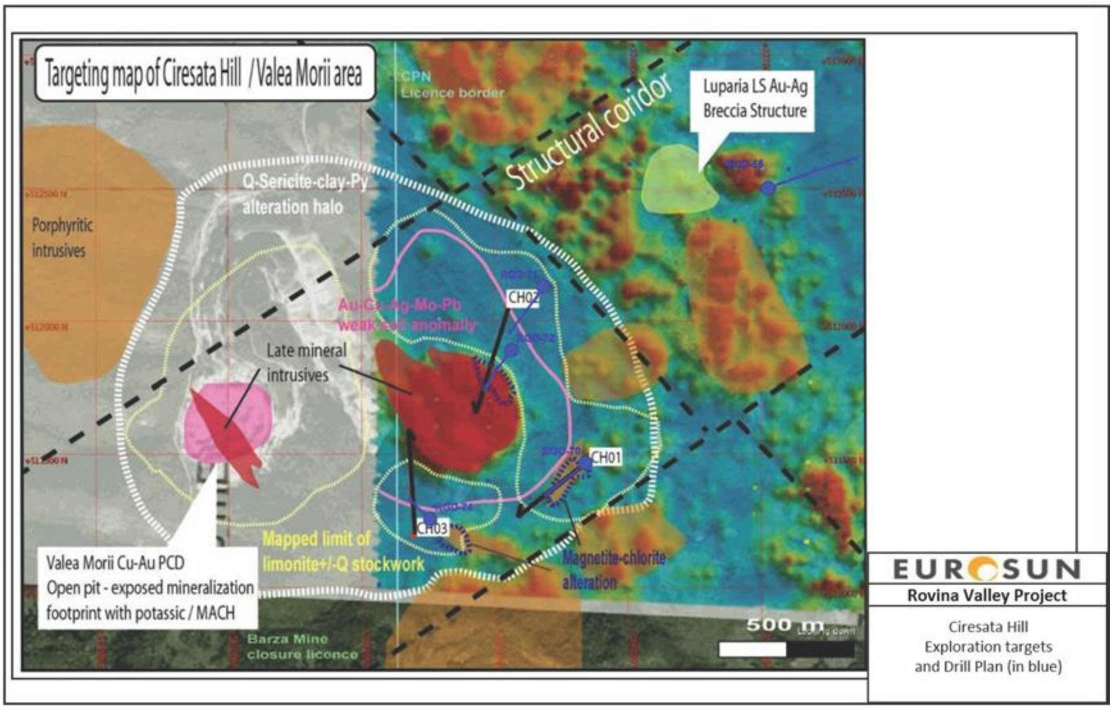


Figure 10.8: Ciresata Hill Exploration Drilling

10.2.10 Zdrapti Prospect

ESM drilling programme at the Zdrapti Prospect began in January 2007, and 11 holes totalling 2,671 m have been drilled to test sub-cropping potassic alteration with a coincident ground magnetic anomaly.

Core drilling at the Zdrapti Prospect has been carried out by Genfor using an RB-57 drill rig capable of depths up to 300 m. Core diameter is generally HQ, and some holes continue to depths of 400 m using this diameter core. Holes are reduced to NQ at depths ranging between 100 m and 110 m, as necessary. Currently, 40 % of the drilling (to the end of hole RZD-4) is with HQ diameter, and 60 % is with NQ.

Zdrapti remains a prospect, without a mineral resource estimate.

10.2.11 Geotechnical and Hydrological Drilling

Two geotechnical drilling programmes were completed, the first between October 2011 and March 2012, and the second between October and December 2020. These programmes totalled 4,331.30 m of diamond core. The programme consisted of the following:

- Five drillholes at the Rovina deposit testing the proposed pit walls, totalling 1,439.50 m. These holes were drilled to depths ranging from 268 m to 351.5 m and averaging 287.9 m.
- Seven drillholes at the Colnic deposit testing the proposed pit walls, totalling 1,809.00 m. These holes were drilled to depths ranging from 180 m to 349.50 m, and averaging 258 m.

- Two drillholes in the Bucureşci Valley above the proposed ore-conveyor gallery, for a total of 320 m; average depth is 160 m.
- Eleven drillholes in the area of the proposed tailings management facility totalling 606.60 m. These holes were drilled to depths ranging from 30 m to 100 m and average 54 m in depth. This facility and the area encompassed are not used in the March 2021 DFS.
- Twelve drillholes in the area of the proposed waste facility and plant area of the current DFS totalling 162.20 m. These holes were drilled to depths ranging from 6.20 m to 26.20 m and average 13.50 m in depth.

10.2.12 Metallurgical Drilling

In 2008, three holes totalling 1,114 m were drilled, one in each of the Rovina, Colnic, and Ciresata deposits, for potential future use in a pilot plant. At Rovina, hole RRM-1 (HQ core size) (see Figure 10.4 was drilled to a depth of 282 m. At Colnic, RCM-1 (HQ core size) (see Figure 10.6) was drilled to a depth of 270 m. At Ciresata, RGM-1 was drilled 562 m with a HQ core size to 301 m and NQ from 301 m to the bottom of the hole.

The ESM 2011–2012 metallurgical programme uses widely distributed samples from half-core from resource drilling to represent selected geometallurgical units.

10.2.13 CCIC MinRes Comment

It is CCIC MinRes's opinion that the exploration efforts conducted by ESM are sufficient for the porphyry style of mineralisation. Historical drill samples by Minexfor were not used in the mineral resource estimation. Recent surface rock chip samples collected by ESM were used in guiding the exploration drilling activities; however, none of these were used in the mineral resource estimate, because ESM has drilled sufficient diamond core holes that cover the deposits.

CCIC MinRes is of the opinion that the drilling method, drill azimuth and dips are appropriate for the porphyry style of mineralisation and the general orientation of the deposit. For the most part, the drill spacing is sufficient for mineral resource estimation at the Rovina, Colnic and Ciresata deposits. CCIC MinRes has not identified any material factors that could adversely affect the mineral resource estimate.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

All routine sample preparation and analyses of the ESM samples are performed by ALS Chemex Romania (ALS), a laboratory in the town of Gura Rosiei (where the Rosia Montana Project is located), about 45 minutes' drive northwest of the project area. Between 1997 and the end of 2005, the laboratory was operated by Analabs, an Australian-based company which at the time was part of the SGS Group. SGS is an internationally recognised organisation that operates over 320 laboratories worldwide and has an ISO 9002 certification for many of its laboratories. The Analabs Laboratory was installed as a dedicated facility for the Gabriel Resources' Rosia Montana Project during the resource definition stage. At the start of 2006, SGS pulled out of the management role of the laboratory due to decreased activity. The name was then changed to ROM Analize SRL (ROM Analize), and for the first half of 2006 was run as a privately operated independent laboratory, managed by Hidi Francisc.

In September 2006, ALS Chemex purchased ROM Analize and now manages the operation of the laboratory. There was a transition period until ALS Chemex analytical methods were adopted, and the change from ROM Analize to ALS Chemex is formally considered in this document as of 28 November 2006. ALS Chemex is an internationally recognised organisation, independent of the issuer, which operates an integrated network of 70 laboratories worldwide, and has ISO 9002 certification for many of its laboratories, including the laboratory in Romania where ESM samples are processed. In May 2012, the ALS Chemex Romanian laboratory was accredited by the Standard Council of Canada with an ISO 17025 certification, which is the highest level of accreditation a laboratory can obtain. All rock and drill samples since ESM acquired the Rovina Property have been submitted to this laboratory, either under the management of Analabs, ROM Analize, or ALS Chemex.

11.1 SAMPLE PREPARATION

11.1.1 Sample Preparation for Soil Samples

Soil samples from the ESM Rovina Property soil programme were prepared by the laboratory at Gura Rosiei. The samples were logged, weighed, and dried before sieving to -80 mesh. Both size fractions were retained. Subsequently the sieved -80 mesh fraction was pulverised to 85 % passing 75 µm or better. A 100 g portion of the final prepared samples was then shipped to the ALS Chemex laboratory in Vancouver, Canada.

11.1.2 Sample Preparation of Rock Chip and Drill Core Samples

All rock chip and core samples are prepared and analysed at the ALS Chemex laboratory, in Gura Rosiei. CCIC MinRes visited this laboratory in November 2020, to review the sample receiving, preparation procedures. All systems and procedures were compliant with ALS Chemex international operating standards, and CCIC MinRes deemed the sample preparation procedures to be to industry standards.

The laboratory can process up to 50,000 samples per month from various clients across Europe, Middle East, and North Africa. At the sample receiving terminal, all samples are weighed, bar-coded, and scanned into their electronic sample tracking system. This system improves turnaround time and reliability of results, within the need for manual capturing.

The laboratory has two large drying ovens with automatic temperature control, two jaw crushers, and six 5 kg capacity LM2 pulverisers. Each of these has forced air extraction and compressed air for cleaning the crushers and pulverisers.

The preparation protocol for the ESM samples is as follows:

- Coding: an internal laboratory code is assigned to each sample at the receiving terminal
- Weighing: all samples are weighed
- Drying: the samples are dried at 105 °C for up to 24 h
- Crushing: the entire sample is crushed to obtain nominal 85 % at 5 mm (from February 2006 to July 2007); from August 2007 to present day the entire sample is crushed to obtain nominal 85 % at 2 mm
- Pulverisation: from February 2006 until 17 October 2006, the entire sample was pulverised for 6 min to 11 min to achieve better than 85 % passing 75 µm and a 150 g to 200 g sample was collected from the pulveriser; from 23 October 2006 until July 2007, a 1 kg split of the 5 mm crushed samples was pulverised to achieve better than 85 % passing 75 µm; from August 2007 to present day, a 1 kg split of 2 mm crushed samples is pulverised to achieve better than 85 % passing 75 µm.
- Storing and submitting: the coarse rejects, pulps, and pulp rejects are stored on site for 30 d and then returned to ESM and archived in a dedicated storage facility.

Wet sieve checks are conducted regularly by ALS Chemex (every 15 to 20 samples). Every five samples on average, the equipment is cleaned by passing local sterile rock, and occasionally coarse pure feldspar, through the crushers and the pulveriser.

11.2 SAMPLE ANALYSIS

11.2.1 Sample Analysis for Soil Samples

During the ESM soil sampling programme in 2007 and 2008, a total of 2,536 samples were analysed at ALS Chemex in Vancouver for gold, and 35 other elements using the following analytical methods:

- Au analyses by a fire assay method (method Au-ICP21) followed by an ICP-AES reading with a 1 ppb detection limit
- 35-element suite: analysed by ICP-AES after aqua regia digestion (method ME-ICP41)

11.2.2 Sample Analyses of Rock Chip and Drill Core Samples

ROM Analize was the principal laboratory for all ESM exploration and drilling programmes. At ROM Analize, for gold analysis, approximately 50 g of the pulp was weighed with automatic digital data capture of the results to a computer. For base metals, approximately 0.40 g of pulp was weighed on a separate scale, with similar automatic data captured to the computer. All samples were assayed for Au by 50 g fire assay with AAS finish (0.01 g/t detection). Base metals, Cu (2 ppm detection), Pb (3 ppm detection), Zn (2 ppm detection), and Ag (1 ppm detection) were analysed by AAS methods using an aqua regia digestion. A Varian SpectrAA AAS instrument was used for Au and Cu readings while a separate instrument was used for Ag and other base metals.

During this period, ALS Chemex in Vancouver was the secondary laboratory used for check assays. The samples were assayed for gold by 50 g fire assay with AAS finish (0.001 g/t detection, method code AA24). Base metals were analysed by an AAS method (method code AA45) using an aqua regia digestion.

After ALS Chemex took over the operation of ROM Analize, in 2006, the analytical method changed to the ALS methods of AA26 (50 g fire assay for gold) and AA45 for base metals. The same Varian SpectrAA AAS instrument was used for Au and Cu readings while a separate instrument was used for Ag and other base metals.

After 2006, the OMAC Laboratory was used for check assays. The samples were assayed for gold by 50 g fire assay with AAS finish (0.01 g/t detection) method A4. A 45-element suite including base metals was analysed by an ICP-AES method using an aqua regia digestion.

Both primary and secondary laboratories reported the results digitally to ESM via email, complete with signed paper certificates. General turnaround varied from approximately three days to several weeks.

CCIC MinRes visited the laboratory in November 2020 to review the sample analytical procedures. ALS Chemex completes the following internal QA/QC protocols during the sample preparation:

- Pulp blank: 1 inserted every 50 samples
- Certified reference material (CRM): 2 inserted every 50 samples (5 CRMs in use ranging between 0.76 g/t Au and 5.12 g/t Au)
- Pulp duplicates: 1 every 12 samples
- Pulp re-assay: 4 every 40 samples
- Participation in internal and external ‘round-robin’ practices

All pulps are automatically labelled with the job number, sequence number, and sample number. Duplicate samples are returned to ESM along with the other pulps but are in a different coloured bag (black as opposed to brown).

All systems and procedures were compliant with international operating standards and CCIC MinRes deemed the sample analytical procedures are to industry standards.

11.3 ESM DRILLING PROGRAMMES

ESM drilling programmes at the Colnic, Rovina, and Ciresata deposits have exclusively used diamond-core drilling methods. Recovery, on average, has been greater than 97 % and all core has been split and sampled on mostly 1 m intervals, honouring important lithological boundaries.

In May 2006, AMEC reviewed the ESM QA/QC programme and recommended some adjustments be made in order to monitor various essential elements of the sampling-assaying sequence, in an effort to control or minimise any possible errors (Cinits, 2006b).

These elements included the following:

- Sample collection and splitting (sampling variance, or sampling precision)
- Sample preparation and sub-sampling (sub-sampling variance, or sub-sampling precision; contamination during preparation)
- Analytical accuracy, analytical precision, and analytical contamination
- Reporting (clerical or data transfer) accuracy

Starting in early July 2006, ESM adjusted the QA/QC protocols and incorporated most of AMEC's recommendations. AMEC's review in 2007 of the QA/QC programmes indicated the protocols were to industry standards, which continue to present. The ESM QA/QC programme since January 2007 consists of the following:

- Pulp duplicate, 3 % of all assayed core samples
- Core twin samples, 3 % of all assayed core samples
- Coarse blank samples, 3 % of all assayed core samples
- Pulp blank samples, 3 % of all assayed core samples
- Coarse rejects, 2.6 % of all assayed core samples
- Gold standard reference material, 3.4 % of all assayed core samples
- Copper standard reference material, 2.2 % of all assayed core samples

11.3.1 Summary of the 2006 QA/QC Results by AMEC

Until 22 November 2006, ESM used ROM Analize as the primary laboratory and ALS Chemex as a secondary laboratory. Based on the QA/QC review conducted on the 2006 Rovina Apuseni drilling programme, AMEC concluded the following:

- The sampling precision for Au and Cu during the last portion of the 2006 exploration campaign (July to November 2006) was satisfactory. The sampling precision prior to July 2006 was not assessed.
- Because the entire sample was pulverised during the period of February to October 2006, sub-sampling variance at ROM Analize was not assessed, nor required. For the period of October to 28 November 2006, when the core samples were split at the 85 % passing 5 mm crushing stage, the sub-sampling variance was found to be adequate.
- Analytical precision for Au and Cu at ROM Analize during the last portion of the 2006 campaign (July to November 2006) was satisfactory, although various sample mix-ups appear to have occurred. Analytical variance prior to July 2006 was not assessed. Based on this partial result, however, and because of the evaluation of the check samples, AMEC infers the analytical precision is likely to be within acceptable limits.
- The gold analytical accuracy at ROM Analize was within the acceptable ranges. Although ESM did not include Cu CRMs on the batches, as a result of the evaluation of the check samples, AMEC concludes the Cu analytical accuracy at ROM Analize was also satisfactory.
- In spite of the original insufficiencies of the QA/QC programme implemented at the Rovina project, AMEC was of the opinion the Au and Cu assays of the 2006 drilling exploration campaign were sufficiently precise and accurate for mineral resource and reserve estimation purposes.

11.3.2 Summary of the Late 2006 – Early 2007 QA/QC Results by AMEC

Starting on 28 November 2006, ESM used ALS Chemex (Romania) as its primary laboratory and OMAC (Ireland) as a secondary laboratory.

Based on the QA/QC review conducted on the Colnic and Rovina drilling programme that was completed during late 2006 to May 2007, AMEC concluded that

- The sub-sampling sampling variance for Au and Cu was satisfactory.
- The analytical variance for Au and Cu at ALS Chemex was satisfactory, although some sample mix-ups appear to have occurred.
- The Au and Cu analytical accuracies at ALS Chemex were within the acceptable ranges.
- No significant cross-contamination for Au and Cu was detected during preparation and assaying at ALS Chemex during the late 2006–2007 exploration campaign.
- The Au and Cu assays of the late 2006–2007 drilling exploration campaign at Rovina were sufficiently precise and accurate for mineral resource and reserve estimation purposes.

AMEC recommended the implementation of the following measures during the continuation of the exploration at the Colnic and Rovina deposits:

- ESM investigate the source of pulp sample mix-ups that appear to have occurred.
- ALS Chemex revise the sample preparation procedure to include crushing to a finer particle size (85 % passing 10 mesh), and splitting the samples after this size reduction rather than at a particle size of 5 mm.

11.3.3 Summary of the PEG Review of the Late 2007–2008 QA/QC

During the period of 2007–2008, ESM used ALS Chemex (Romania) as a primary laboratory and OMAC (Ireland) as secondary laboratory. Based on the QA/QC review conducted on the 2007/2008 Rovina drilling programme, PEG concluded that

- The sampling precision for Au and Cu during the late 2007–2008 exploration campaign was satisfactory.
- The gold analytical accuracy at ROM Analize was within the acceptable ranges. Close monitoring of the CRM results needs to be implemented with partial batch resubmission for bracketing the samples surrounding a two-standard deviation failure.
- The Au and Cu analytical accuracies at ALS Chemex were within the acceptable ranges throughout the 2007–2008 exploration programme. The pulp duplicate results indicated a slight degradation in precision near the end of the exploration programme that should be monitored closely.
- No significant cross-contamination for Au and Cu was detected during preparation and assaying at ALS Chemex during the 2007–2008 exploration campaign.
- The Au and Cu assays of the late 2007–2008 drilling exploration campaign at Rovina are sufficiently precise and accurate for mineral resource and reserve estimation purposes.

11.3.4 AGP Review of the 2010–2012 QA/QC Results

During the 2010–2012 drill campaign, ESM primary laboratory remained ALS Chemex (Romania). The secondary laboratory was changed due to ALS Chemex purchasing the facility in Ireland. Check samples are now submitted to the SGS laboratory in Chelopech, Bulgaria. ESM routinely charts all QA/QC samples every third or fourth batch report from the laboratory. If a trend is noticed or samples are deviating from the norm, the batches are resubmitted. No other changes took place regarding the analytical procedure.

11.3.4.1 Coarse and Pulp Blanks

AGP reviewed control charts provided by ESM. Cross-contamination would be considered significant if the blank value exceeded five times the detection limit for the element.

The crushable coarse blank material consists of pyroxene andesite collected on the property and is the same material used by ESM since 1999. Results of the coarse blank show one gold failure and one copper failure exceed five times detection out of 1,461 samples.

The crushable pulp blank material consists of the same material for the coarse blank but is pulverised at the laboratory. Results of the pulp blank show zero gold failure and zero copper failure exceed five times detection out of 1,465 samples. All assays are well below twice the detection limit.

No trend or pattern was noticeable from the data examination.

11.3.4.2 Core Twin

ESM no longer submitted ¼ core twin to the laboratory during the 2010–2012 drill programme as per AMEC's recommendation in their 2006 report.

11.3.4.3 Coarse Rejects

A total of 1,430 coarse rejects were re-inserted in the sample stream. Of these, 71 rejects originated from the 2008 drill programme.

Gold pulp duplicate shows good agreement with the paired sampled. The R2 value is very good at 0.98, and the slope of the regression is 0.984 after nine outliers were removed. The data shows a good distribution on either side of the parity line.

Copper is a little better with an R2 of 0.99 and a slope of regression of 1.001. Data shows good distribution on either side of the parity line after 11 outliers were removed.

11.3.4.4 Pulp Duplicates

A total of 1,464 pulp duplicates have been submitted since the end of the 2008 drill programme. Gold pulp duplicates show good agreement with the paired sampled. The R2 value is very good at 0.95 and the slope of the regression is 1.041. The data also shows a good distribution on either side of the parity line. Copper is virtually the same with an R2 of 0.92 and a slope of regression of 0.99. Data shows good distribution on either side of the parity line.

11.3.4.5 Certified Reference Material (CRM)

For the 2010 to 2012 drill programme ESM used three gold only CRMs purchased from Rocklabs (SG31, SH35, SE29), and seven gold and copper CRMs purchased from CDN Laboratory (CM-11A, CGS-22, CGS-16, CGS-13, CGS-24, CM-3, CM-16). The origin of the material for CRM CM-8, CM-11A, and CGS-13 most closely matched the porphyry type of mineralisation on the Rovina property. For evaluation of the CRMs, AGP examined the gold and copper control charts produced by ESM.

The standard performance of 1,378 assays for gold is considered excellent by AGP for the period reviewed. The CDN CRMs appear to be more stable and consistent than the Rocklabs CRMs. The low-grade CRMs are both provisional and indicate low bias for CGS-16 and a high bias for CM-16 although the failure rate is very low. The higher-grade CRMs through 2010–2012 show an average of 0.59 % low bias with a very low failure rate (see Table 11.1).

Table 11.1: Summary of Gold Best Values and Performance of CRMs

Au STD	No. of Samples	Best Value	Average	Bias (%)	Fail 3 Sigma	Fail 2 Sigma (2 Seq)
SE29	39	0.597	0.589	-1.36%	0	0
CDN-CM-8*	294	0.910	0.901	-1.00%	0	0
CDN-CGS-24	296	0.487	0.496	1.81%	0	0
SG31	38	0.996	0.989	-0.71%	0	0
SH35	18	1.323	1.269	-4.26%	1	0
CDN-CGS-16 (indicated mean)	364	0.140	0.157	-10.83%	1	0
CDN-CGS-13*	78	1.010	0.998	-1.20%	0	0
CDN-CM-11A*	86	1.014	1.017	+0.29%	0	0
CDN-CM-16 (provisional)	90	0.294	0.323	+8.98%	0	0
CDN-CGS-22	75	0.640	0.651	-1.69%	0	0

For copper, a total of 1,283 sample results were reviewed. ESM used seven gold and copper CRMs purchased from the CDN Resources Laboratory. The data for the 2010 to 2012 drill programme indicated a failure rate greater than three times standard deviation of 4.4 % (see Table 11.2). The CDN-CGS-16 CRM shows an abnormally high failure rate despite ESM resubmitting a number of re-assayed batches. Eliminating the CGS-16 CRM from the list, the average failure rate is reduced from 4.4% to 1.9%. AGP notes the high bias (averaging 2.3 %) is shown for all CRMs analysed. When compared to the 2007–2008 data, the bias at that time averaged 0.70 %. AGP is of the opinion ESM needs to investigate the source of possible bias prior to submitting more samples to the ALS Chemex facility and contract an audit of the facility with a specialist in laboratory procedures.

Table 11.2 Summary of Copper CRMs (2010–2012)

Cu STD	No. of Samples	Best Value	Average	Bias (%)	Fail 3 Sigma	Fail 2 Sigma (2 Seq)
CDN-CM-16	90	1840	1893	2.80%	0	0
CDN-CM-8*	294	3640	3792	4.01%	8	5
CDN-CGS-24	296	4860	5000	2.80%	3	1
CDN-CM-11A*	86	3320	3454	3.88%	9	19
CDN-CGS-22	75	7250	7316	0.90%	4	0
CDN-CGS-16	364	1120	1134	1.23%	32	10
CDN-CGS-13*	78	3290	3303	0.39%	1	1

In March 2012, ESM requested ALS Romania to investigate a high bias and failure rate for two routinely submitted copper CRM's during the January to March 2012 period. ALS Romania concluded from their internal QC investigation that improper calibration fluids during this period resulted in a high-bias for copper. This problem was corrected and documented in a complete report (Quality Investigation into ESM Gold QA Issues, May 2015, ALS Romania SRL report to ESM). ESM resubmitted ten batches for re-assay that were affected by the bias for a total of 2,376 samples. These re-assays were utilised in the mineral resource estimate. Results from the re-assay corrected the problem as shown in Figure 11.1; however, a positive bias is noticeable.

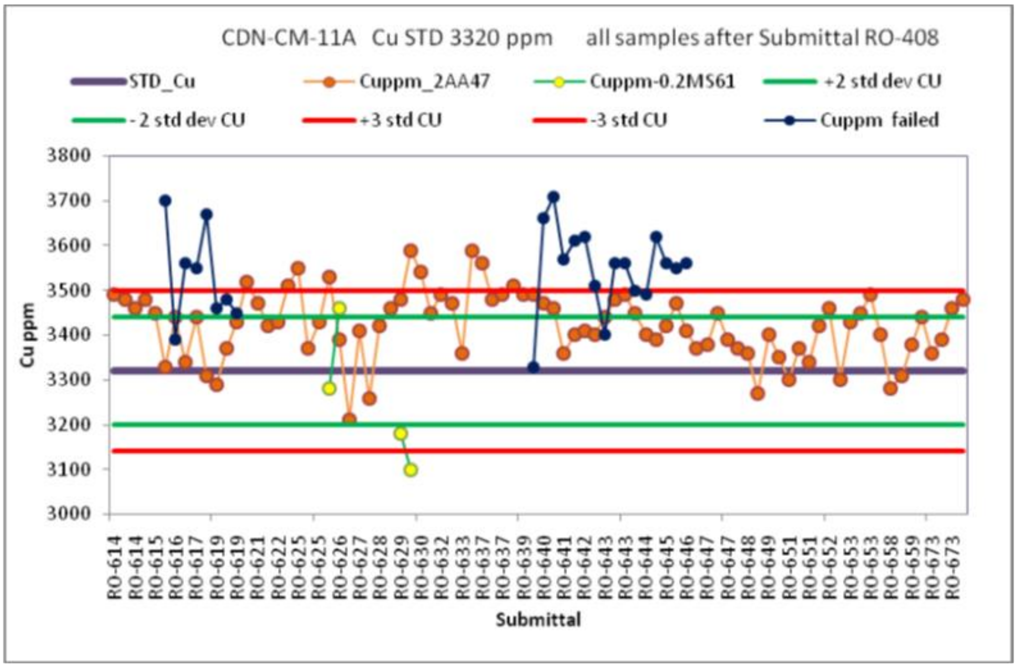


Figure 11.1: CM-11A CRM Batch Resubmission

11.3.4.6 Check Samples

During the 2006–2008 campaign, ESM utilised the OMAC laboratory in Ireland for check assays. For the 2010–2012 drill programme, a total of 1,354 sample pulps from the ALS

Chemex Gura Rosiei facility were submitted to the SGS Laboratory in Chelopech, Bulgaria for check assays. Results indicate after excluding 15 outliers for gold, the regression shows an R2 value of 0.981 with a good distribution around the parity line. Slope of the regression is approaching 1 at 0.996.

For copper, results indicate after excluding 13 outliers, the regression shows an R2 value of 0.986 with a good distribution around the parity line. Slope of the regression is lower at 0.917 and the frequency distribution of the regression residual show a high bias for the sample analysed at the Gura Rosiei facility when compared to the sample analysed at the Chelopech facility.

At the time of writing, ESM is currently investigating this discrepancy since the CRMs submitted with the batch indicated a low bias of 4 % at the Chelopech facility. AGP noted a high bias for copper assays appears to substantiate the high bias seen in ESM CRM, although this could be coincidental and made worse by the low bias seen at the check laboratory.

Based on the QA/QC review conducted on the 2010–2012 Rovina Valley drilling programme, AGP concluded that

- The sampling precision as indicated by the pulp and coarse duplicate for gold and copper during the 2010-2012 exploration campaign was satisfactory.
- The gold analytical accuracy at ALS Chemex was within the acceptable ranges. The copper analytical accuracy at ALS Chemex shows a high bias with an abnormally high rate of failure of the CRMs. This was followed by ESM resubmittal of ten batches containing a total of 2,376 samples, which corrected the failures. This needs to be investigated further.
- Check assays from the secondary laboratory indicated the gold analytical accuracies at ALS Chemex were within the acceptable ranges throughout the 2010–2012 exploration programme. The CRMs submitted with the check assay samples indicated a low bias for Cu from the secondary laboratory. This may have been a factor in the results for copper, which indicated a high bias for samples originating from ESM primary laboratory. At the time of writing this report, this issue was being investigated.
- No significant cross-contamination for Au and Cu was detected during preparation and assaying at ALS Chemex during the 2010–2012 exploration campaign.
- The Au and Cu assays of the late 2010–2012 drilling exploration campaign at the RVP are sufficiently precise and accurate for mineral resource estimation purposes. A copper bias of +2.3 % during the 2010–2012 programme has been detected. This bias needs to be addressed by ESM prior to the next resource update.

11.3.5 CCIC MinRes Review of the QA/QC results

In September 2020, CCIC MinRes conducted a review of the QA/QC results for all drilling since 2006. The outcomes of the review are summarised below:

- Although ESM employs industry leading best practice protocols and procedures for their QA/QC programme, the precision and accuracy of QA/QC results (particularly for gold) were often close to the threshold limits and in some cases beyond the limits.

- ALS Chemex internal repeat assays for gold grades less than 1.5 g/t fall within a 20 % difference. This percentage difference improves gradually to within 10 % for gold grades above 1.5 g/t. This percentage difference, however, is unbiased in that there is an even spread of positive and negative differences.
- Course reject repeats for both gold and copper are within a 20 % difference. This percentage difference, however, is unbiased.
- Pulp reject repeats for gold grades less than 1.5 g/t fall within a 40 % difference, with a very slight positive difference, meaning that the original gold assays are generally higher than the check assays. Copper pulp repeats are within a 20 % difference (unbiased).
- Check assays between ALS and SGS for gold values above 0.5 g/t fall within a 20 % difference. Check assays for copper also fall within a 20 % difference but have a positive difference.
- For course blanks, 0.2 % (8 out of 3716) of the results were outside the tolerance. All of these were during the 2007/2008 period.
- Although the majority of the results from the CRMs were within the two-standard deviation tolerance, there is generally a positive bias, meaning that the results are generally higher than the CRM value. Results of some batches were outside the acceptable limits, as in the case between December 2011 and March 2012, and these batches were re-analysed in May 2015.
- It is recommended that ESM implement database management software to manage QA/QC results and sign offs, prior to acceptance of the assay results. This system will flag batches that are beyond the threshold limits and allow for immediate remedial action. Also, where there is evidence of positive or negative drifts, the laboratory can be notified to calibrate their equipment more regularly.

Despite minor insufficiencies of the QA/QC programme implemented by ESM, CCIC MinRes is of the opinion that the Au and Cu assays from drillhole sampling of the RVP are sufficiently precise and accurate for mineral resource and reserve estimation purposes.

11.4 SECURITY

All the sample collection and handling are carried out by ESM staff at the core shed in Crişcior. ESM uses a double-checking method during sampling which eliminates the risk of sample mix-up.

The core is stored under covered sheds that are secured with padlocks. The perimeter is surrounded by a fence and monitored 24/7 via video camara, and access to the facility is controlled by a security guard.

11.5 DATABASE CONTENT AND INTEGRITY

ESM is currently entering all its geological and geotechnical data (including relevant historical data) into a combination of Excel, MSAccess® (Access), MapInfo, and Micromine databases. Sampling data recorded in the field is manually captured into Excel files by a technician. Following this, the files are transferred to the database manager in Crişcior, who incorporates them into the project Access database. Sampling data is then transferred independently to the Micromine database. Assay data is not manually entered on site because it is digitally sent from the laboratory and imported into the Access and Micromine databases. ESM visually checks each assay on the signed paper certificate against that in the digital databases.

Geological data from the geological logs is entered from the hardcopy logs into the Micromine database and then transferred to the Access database in Crişcior. Downhole surveys with any Reflex instruments are digitally recorded by the instrument on the surface and transferred by the drill contractor to ESM and then imported into Access and Micromine. Drillhole collar survey data is not manually entered because it is digitally sent by the surveyor to ESM and imported into the Access and Micromine databases.

ESM generates up-to-date drill strip-logs and drill sections. For the drillhole results up to the end of the 2008 drill programme, manual section interpretations (on paper) were reconciled to level plans and longitudinal sections to ensure the domains (3D wireframes) were properly constructed and the interpretations were sound. Sections and levels were reviewed on a regular basis to ensure all holes have crossed the target as planned, and sufficient data density exists to make an appropriate interpretation. Three-dimensional wireframe models of all lithologies and alteration types were constructed from sections and plans. For the 2011 and 2012 drill campaign, the 3D wireframe models were updated directly in the Micromine software.

PEG audited the databases that form the basis of the updated resource estimates and found them to be well-organised and transparent. The databases are duplicated in the ESM office in Crişcior for security. ESM dedicates one or two technicians for most of the data entry, including collar data, survey data, drill logs, assay data, etc. The database was subsequently audited by AGP as part of the data validation and AGP arrived at the same conclusion as PEG.

CCIC MinRes reviewed the database as part of the site visit in November 2020 and concluded that the exploration data is well managed and secured. For further database enhancement and efficiency, CCIC MinRes recommends that ESM implement database management software to manage QA/QC results and sign offs, prior to acceptance of the assay results. This will become more important when ESM ramps up drilling meterage in anticipation of mining operations.

12 DATA VERIFICATION

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

The ESM geological staff have numerous verification procedures to assure quality and reliability of the geological database. These include the following:

- Double-checking of the drillhole collar coordinates, downhole survey trace and laboratory assay certificates against the electronic records in the database
- Printing of strip-logs for the completed drillholes, then comparing that against the original hardcopy logs
- Checking significant assay results from the strip-logs against those intervals in the drill core to verify to results
- Printing of drillhole cross-sections to test plausibility of assay results of drillholes within a cross-section, and of drillholes between cross-sections.

12.1 AGP ASSESSMENT 2008 AND 2011

12.1.1 Site Visit

Mr Pierre Desautels, AGP QP for the 2019 mineral resource estimate, visited the Rovina Valley property, accompanied by Mr Randall K. Ruff, ESM Executive Vice-President, Exploration, between 26 and 30 August 2008 and revisited the property between 26 and 29 July 2011. The site visits entailed brief reviews of the following:

- Overview of the geology and exploration history of the Colnic and Rovina porphyries (presented by ESM geologists Mr Randall K. Ruff and Dr Barbara Stefanini)
- Exploration programme at Ciresata (drillhole orientation, depth, number of holes, etc.)
- Infill drill programme for resource category conversion for Rovina and Colnic
- Visits to operating drill rig and drillhole collars at the Colnic, Rovina (2008), and Ciresata Porphyries (2011) accompanied by Sorin Halga
- Drill rig procedures including core handling on site
- Surveying (topography, collar, and downhole deviations)
- Sample collection protocols at the core logging facility
- Sample transportation and sample chain of custody and security
- Core recovery
- QA/QC programme (insertion of standards, blanks, duplicates, etc.)
- Monitoring of the QA/QC programme
- Review of diamond drill core, core logging sheets, and core logging procedures (the review included commentary on typical lithologies, alteration and mineralisation styles, and contact relationships at the various lithological boundaries; the 2011 site visit focused on the newly discovered geological features of the Ciresata deposit)
- Density sample collection
- Geological and geotechnical database structure and all procedures associated with populating the final assay database with information returned from the laboratory
- Change of ownership of ESM primary laboratory

The three-character samples collected during the 2008 site visit were supplemented by three additional samples collected during the 2011 site visit. Character samples consisted of quarter-core duplicate of selected ESM sampling intervals. Mr Desautels retained full custody of the sample from Rovina Valley to the AGP office in Barrie, ON, where the samples were shipped to Activation Laboratories Ltd located at 1428 Sandhill Drive, Ancaster, ON. The main intent of analysing these samples was to confirm the gold and copper presence on the deposit by an independent laboratory not previously used by ESM. These samples were analysed for gold by 50 g fire assay with AA finish and total copper using ICP.

From the assay results shown in Table 12.1, AGP concluded the general range of values returned by the samples corresponded well with those reported by ESM. From the samples collected, gold and copper grade appeared to be very consistent exhibiting a good repeatability between the original 1/2 core sample and the 1/4 core sample.

Table 12.1: Assay Results from Character Sample

Sample Nb	Au (g/t)	Cu (%)	ESM Nb	Au (g/t)	Cu (%)	Au Diff. (g/t)	Cu Diff. (%)
00901 (2008)	2.000	0.950	RRD12257	2.27	0.972	0.27	0.022
00902 (2008)	1.850	0.424	RCD1514	1.75	0.472	-0.1	0.048
00903 (2008)	1.220	0.225	RGD4408	1.00	0.190	-0.22	-0.035
00951 (2011)	0.982	0.135	RGD-10004	0.92	0.134	-0.062	-0.001
00952 (2011)	2.000	0.271	RGD-10183	2.15	0.285	0.15	0.014
00953 (2011)	5.33	0.465	RGD-6690	5.23	0.501	-0.1	0.036

The on-site core handling was found to be very efficient. The core is collected at the drill rigs daily and brought to the ESM office in Crişcior where it is immediately laid out on a large table, measured, and prepared for photographing. The core is then moved inside the core facility for geotechnical logging. Once the geotechnical data acquisition is complete, the core is cut with a diamond saw and sampled prior to logging. The saw blade in the cutting operation is continuously cooled by fresh water (not recirculated water). It was noted the geotechnical logs are much improved since the 2008 site visit. Although the Mr Desautels is not qualified to assess their suitability, the logs are like what is seen at other operations.

Following the cutting operation, the core is sampled. The computer-generated sample intervals are typically 1 m but can reach 2 m in the upper part of Ciresata where the deposit is poorly mineralised. ESM typically samples across lithological boundaries. This is not considered to be an issue in a porphyries system. Similarly, to the 2008 site visit, the logs are completed on paper and then transcribed to computer.

The insertion of the purchased standard, pulp blank, coarse blank, and pulp duplicate in the sampling chain was observed during the core logging facility visit.

Geologists responsible for logging can roughly estimate (high/low) the grade of the core visually by the chalcopyrite content, the stockwork intensity, and to a lesser extent the alteration. Sharp contacts are often seen at the lithological boundaries (depending on the unit) as long as the contact has not been obscured by alteration. For example, during the core review at Rovina, the contact between the POB and POC was transitional over a distance of 7 m while the contact between the Glam Breccia and POC was sharp. The alteration boundaries are more difficult to pinpoint due to the overprinting of multiple alteration phases. ESM geologists commented on the K2/K3 boundary that can be determined within ± 3 m to 5 m accuracy while the argillic/phyllic contact can be determined to ±1 m accuracy. Phyllic and K2 boundary is usually sharp.

Figure 12.1 displays some of the photographs taken during the 2011 site visit.

Core Cutting in Progress

Drill Rig at Ciresata (Hole RGD-41)

Hole RGD-26 GPS Coordinate

QA/QC Samples Ready for Insertion

Inter-Mineral Porphyry and Sediment Contact in Hole RGD-17 at 454 m

Pulp Storage Area

Figure 12.1: Photographs taken during the 2011 Site Visit

12.1.2 Database Validation

Following the site visit and prior to the mineral resource estimation, AGP carried out an internal validation of the drillholes in the Rovina Valley database used in the September 2008 resource estimate. Data validation had been done by AMEC in 2006 and 2007, prior to the completion of the 2007 drill campaign, and did not include any of the 2008 drill data. The hole selection for the validation was heavily weighted on the 2007/2008 drilling with spot checks of the earlier holes verified by AMEC. At that time, a total of 46 holes were partially or completely validated amounting to 10,938 individual samples out of a total of 61,329 (17.8 %) that were either checked against paper copies of the original signed certificate or against the electronic version of the certificate provided by the issuing laboratory.

In 2012, the database was validated by first restoring the archived version of the 2008 resource database and comparing the collar coordinate, survey, lithology, and assays. No changes were noted on the holes validated in 2008 therefore the drillhole selection for AGP's validation was restricted to the 2010/2012 drilling.

12.1.3 Collar Coordinate Validation

Collar coordinates were validated with the aid of a handheld Garmin GPS Map, Model 60CSx. A series of collars was randomly selected, and the GPS position was recorded. The X and Y GPS recorded position was compared to that in the Gems database. It was noted that the differences between the two measurements were within acceptable tolerance, considering the accuracy of a handheld GPS against a total station.

12.1.4 Downhole Survey Validation

The downhole survey data was validated by searching for large discrepancies between dip and azimuth reading against the previous reading. A total of 11,128 readings were evaluated. Vertical holes were separated from the incline holes to be evaluated separately. The absolute differences between the readings were calculated along with the statistical summary at the 10th, 25th, 50th, 75th, 90th, and 99th percentile. Measurements deviating excessively (> 99th percentile) from the previous reading were manually reviewed. Out of the 20 azimuths manually reviewed, eight were considered dubious. For the 16 dip measurements manually reviewed, five were considered dubious. All suspected records were forwarded to Dr Barbara Stefanini for validation against the original downhole instrument value. None were corrected as the database entry coincided with the instrument reading.

12.1.5 Assay Validation

Assay validation proceeded as follows:

- During the 2008 review:
    - Validation of GEMS database entry against the PDF copies of the original signed laboratory certificated for a limited number of holes
    - Validation of the electronic version of the certificates against the Gems database entry

- Electronic comparison of the database maintained by ESM Information Technology (IT) Department against the Micromine database maintained by Dr Barbara Stefanini

- During the 2011 review:
  - Comparing the 2012 assay database against the archive copy of the database used in the 2008 resource estimate
  - Validating the electronic version of the certificate against the GEMS database entry for any assays following the 2008 data cut-off date

The validation against the original signed PDF copies of the certificate consisted of three holes RCD-1, RCD-10, and RRD-2 with 1083 assays re-typed in an XLS spreadsheet. No errors were found.

The validation against the electronic version of the certificates consisted of comparing the values on the certificate against the GEMS database entry.

In 2008, 39 certificates were requested from the ALS Chemex (ESM principal laboratory) in Gura Rosiei, Romania via Mr Randall K. Ruff of ESM. ALS Chemex issued the requested certificates as a series of text files in space delimited format (SIF) directly to the author's email address. A total of 13,030 assay results covering 43 drillholes were compiled from the certificates onto an Excel spreadsheet and matched against the sample number in the GEMS database. A total of 3,175 QA/QC assays did not find a matching sample number in the GEMS database with the remaining 9,855 sample numbers successfully matched.

The 2012 database assay entries for the portion of the data that was used in the 2008 resource estimate were found to be same; therefore, the 2012 validation could focus solely on the data that was added after the 2008 data cut-off date.

An additional 21 certificates were requested from the ALS Chemex in Gura Rosiei, Romania via Mr Randall K. Ruff of ESM. ALS Chemex issued the requested certificates in comma delimited format (CSV) directly to the author's email address. The certificates were preferentially selected to cover the highest gold grade value and also to be representative of the assay distribution between deposits. A total of 5,299 assay results covering 50 drillholes were compiled from the certificates onto an Excel spreadsheet and matched against the sample number in the GEMS database. A total of 572 QA/QC assays did not find a matching sample number in the GEMS database with the remaining 4,727 sample numbers successfully matched.

In 2008, out of the 9,855 sample numbers, only two gold and two copper entries did not match the certificate value, and these were corrected in the 2012 database. Assay validation covers 16 % of the entire database with a total of 27 % coverage for the 2007–2008 data not previously validated by AMEC. In 2012, no error was found (see Table 12.2).

Table 12.2: Assay Verification Statistics

	2008	2012	2008 and 2012
Total number of assays in laboratory cert	13,030	5,299	18,329
Number of laboratory cert with no matching number in	3,175	572	3,747

	2008	2012	2008 and 2012
GEMS (QA/QC samples)			
Total number of laboratory cert with matching sample number	9,855	4,727	14,582
Total number of assays in GEMS database	60,268	53,761	114,029
Percentage checked against total	16%	9%	13%
Number of errors found (gold)	2 (corrected 2012)	0	0
Number of errors found (copper)	2 (corrected 2012)	0	0

12.1.6 Assessment of the Drillholes Completed after the 2012 Data Cut-off Date

When the PEA update study on the Colnic deposit was commissioned by AGP, the 2012 Resource Model had to be reviewed for suitability. Following the 31 May 2012 data cut-off date, a total of nine holes were added on the Ciresata deposit, three holes on Colnic and three holes on Rovina. AGP assessed the possible impact to the resources from the additional drillholes. Not all holes intersected the 2102 resource model; Table 12.3 shows the drillholes completed after the data cut-off.

Table 12.3: Drillholes added after the 2012 Data Cut-Off Date

Deposit	Hole-ID	Comments
Rovina	RRD-85	Hole in mineralised POD
Rovina	RRM-1	Metallurgical hole – in mineralised POB
Rovina	RRD-84	Hole in mineralised IMB but on the immediate contact with unmineralised GLAM
Colnic	RCD-105	Drilled outside the resource model
Colnic	RCD-106	Drilled on the edge of the deposit in poorly mineralised SED and SAT-POR
Colnic	RCM-1	Metallurgical hole drilled in high grade C-POR. Near twin of RCD-3
Ciresata	RDG-65	Not intersecting resource model
Ciresata	RDG-66	Edge of interpolated blocks (only first 300m in resource model) - waste
Ciresata	RDG-67	Abandon
Ciresata	RDG-68	Hole in high grade material HRP and SED extend 369 m pass the resource model lower elevation
Ciresata	RDG-69	Hole in high grade material SED extend 278 m pass the resource model lower elevation
Ciresata	RDG-70	Not intersecting resource model
Ciresata	RDG-71	Not intersecting resource model
Ciresata	RDG-72	Not intersecting resource model
Ciresata	RDG-73	Edge of interpolated blocks (only first 113 m in resource model) – waste
Ciresata	RDG-74	Not intersecting resource model

To assess the possible impact of the new holes on the mineral resources, the drillholes were composited to match the block model benches (or levels). The corresponding block model data was selected using a 5 m radius tube around each drillhole. Bench composite values were then compared with the block model grade.

The grade difference between the drillhole and the block model was compiled for the mean, and the distribution was also compared at 5th, 10th, 25th, 59th, 75th, 90th, and 95th percentiles. The difference between the grade of the drillhole and the grade of the block model was typically less than 0.1 g/t AU and less than 0.05 % Cu. The holes showing the most variations (> ± 0.1 g/t) for gold were Hole RCM-1 at Rovina, RRM-1 at Colnic, and RGD-69 at Ciresata. For copper, the holes showing the most variation (> ± 0.05% Cu) were RRD-84 and RRD-85 (see Table 12.4 and Table 12.5).

Table 12.4: Gold Grade Difference Distribution

Gold Grade Difference	ALL	Rovina		Colnic			Ciresata			
		RCD-106	RCM-1	RRD-84	RRD-85	RRM-1	RGD-66	RGD-68	RGD-69	RGD-73
Mean	0.00	-0.03	-0.06	0.01	0.02	0.02	-0.01	-0.02	-0.12	0.03
5th percentile	-0.01	0.01		0.00	0.00	-0.18	0.00	0.00	-0.02	0.00
10th percentile	-0.03	0.00	-0.17	0.01	0.00	-0.09	-0.01	-0.03	-0.03	0.00
25th percentile	-0.08	0.00	-0.28	-0.06	0.00	-0.03	-0.01	-0.09	-0.06	0.00
Median	-0.09	-0.04	-0.09	-0.11	-0.01	-0.05	0.00	-0.06	-0.14	0.01
75th percentile	-0.04	-0.06	-0.06	0.03	0.02	0.13	0.00	-0.05	-0.18	0.03
90th percentile	-0.08	-0.06	0.36	0.23	0.09	0.22	-0.03	0.08	-0.28	0.12
95th percentile	-0.10	-0.07		0.41	0.12	0.19	-0.03	0.15	-0.03	0.16
Count > ± 0.1 g/t	0	0	3	2	1	3	0	1	3	2

Table 12.5: Copper Grade Difference Distribution

Copper Grade Differences	ALL	Rovina		Colnic			Ciresata			
		RCD-106	RCM-1	RRD-84	RRD-85	RRM-1	RGD-66	RGD-68	RGD-69	RGD-73
Mean	-0.01	-0.01	0.00	-0.03	0.05	0.01	0.00	-0.01	-0.02	0.00
5th percentile	0.00	0.00		0.00	0.00	-0.04	0.00	-0.01	0.00	0.00
10th percentile	0.00	0.00	-0.02	0.00	0.00	-0.03	0.00	-0.03	-0.01	0.00
25th percentile	-0.02	0.00	-0.04	-0.03	0.03	0.07	0.00	-0.02	-0.03	0.00
Median	-0.01	0.00	0.01	-0.09	0.03	0.02	0.00	-0.01	-0.02	0.00
75th percentile	-0.02	-0.05	0.01	-0.05	0.09	0.02	0.00	-0.02	-0.01	0.00
90th percentile	0.00	-0.03	0.00	0.04	0.13	0.03	0.00	0.00	-0.04	0.00
95th percentile	0.04	-0.02		0.05	0.14	0.04	0.00	0.02	-0.02	0.00
Count > ± 0.05 g/t	0	0	0	1	3	1	0	0	0	0

Composite drillholes graded by bench were also plotted against the block model grade. Graphs indicated the drillhole grade follows the resource model grade relatively well when taking into consideration the inherent smoothing of the grade distribution from kriging composite points. A few example graphs are shown below.

Hole RGD-69 at Ciresata and hole RCM-1 at Colnic both display a very good correlation when compared to the block model grade (see Figure 12.2 and Figure 12.3)

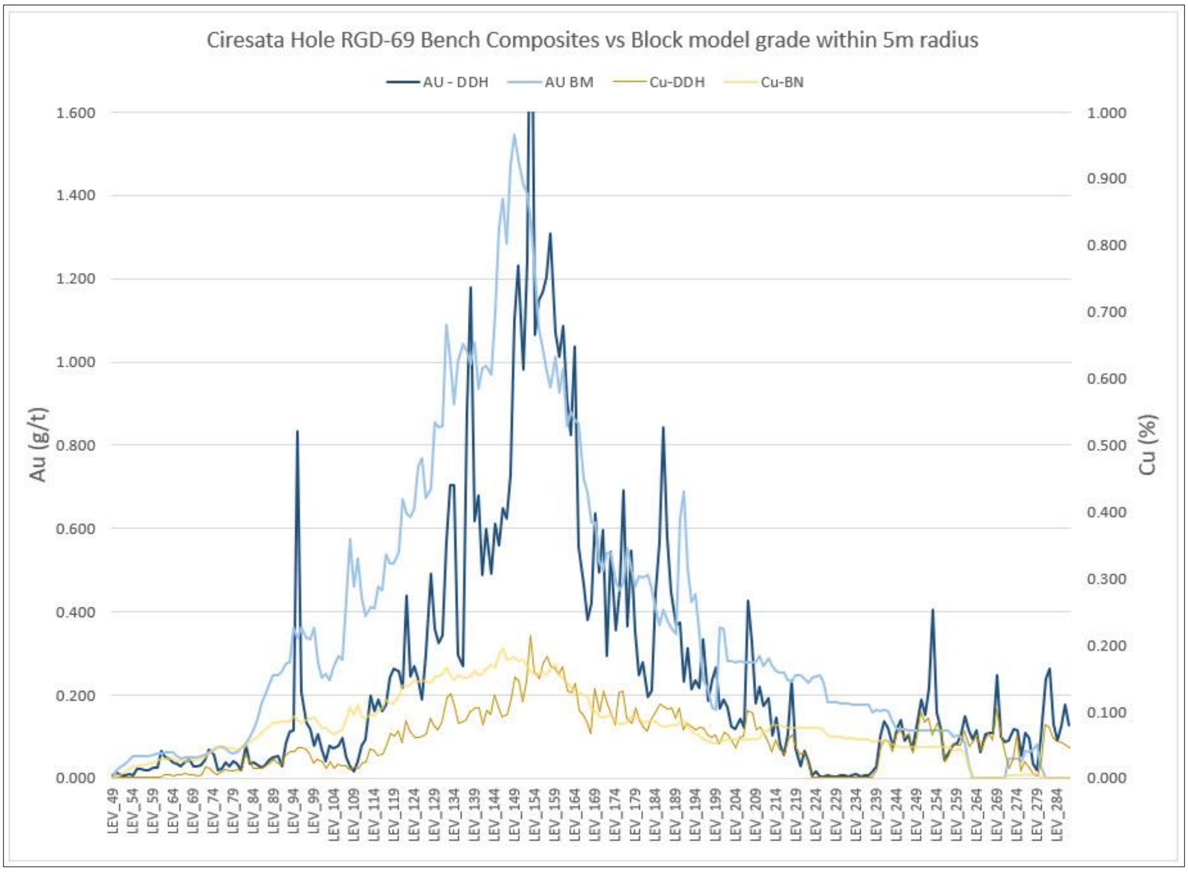


Figure 12.2: RGD-69 versus Resource Model

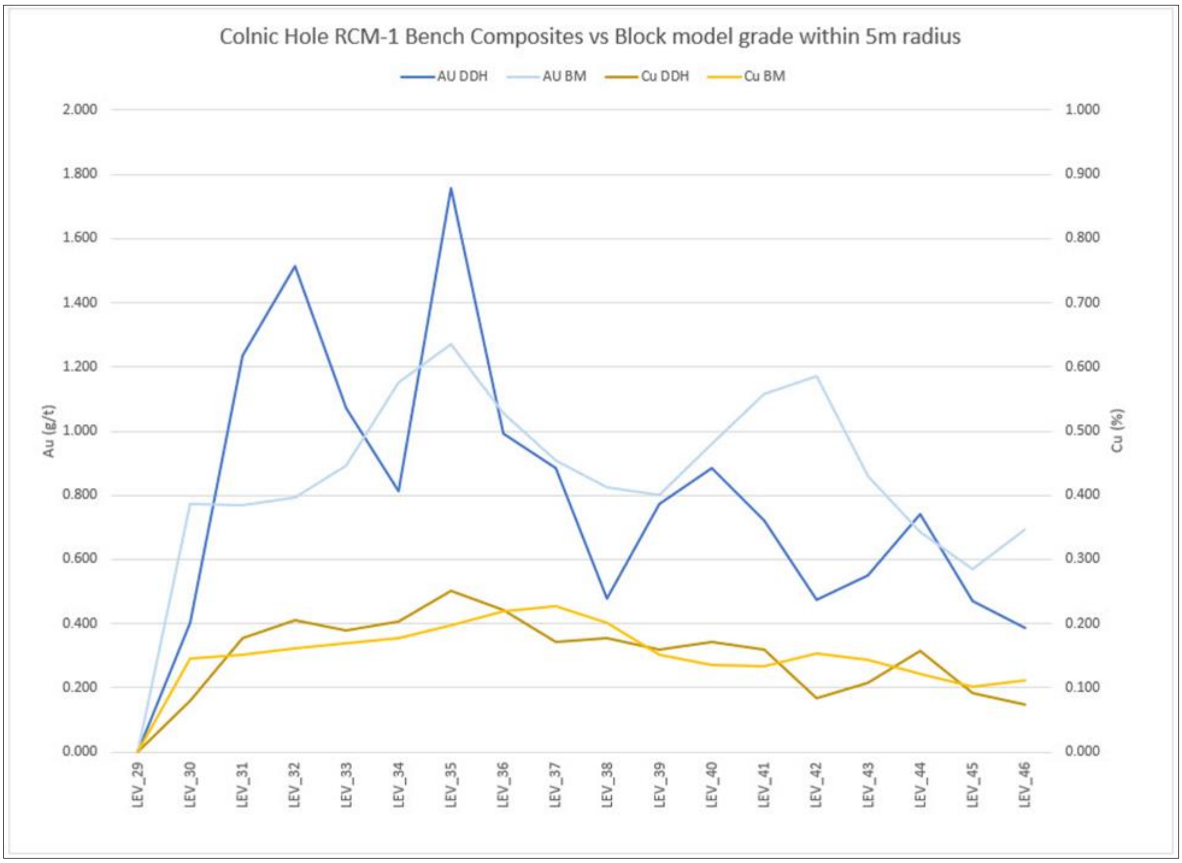


Figure 12.3: RCM-1 versus Resource Model

Hole RRD-84 displays the worst results. The data clearly shows where the drillhole exited the higher grade IMB/POC lithology earlier than anticipated by the 2012 interpretation. As a result, the lower drillhole grade in the GLAM lithology beyond LEV_31 is not reflected in the resource model (see Figure 12.4).

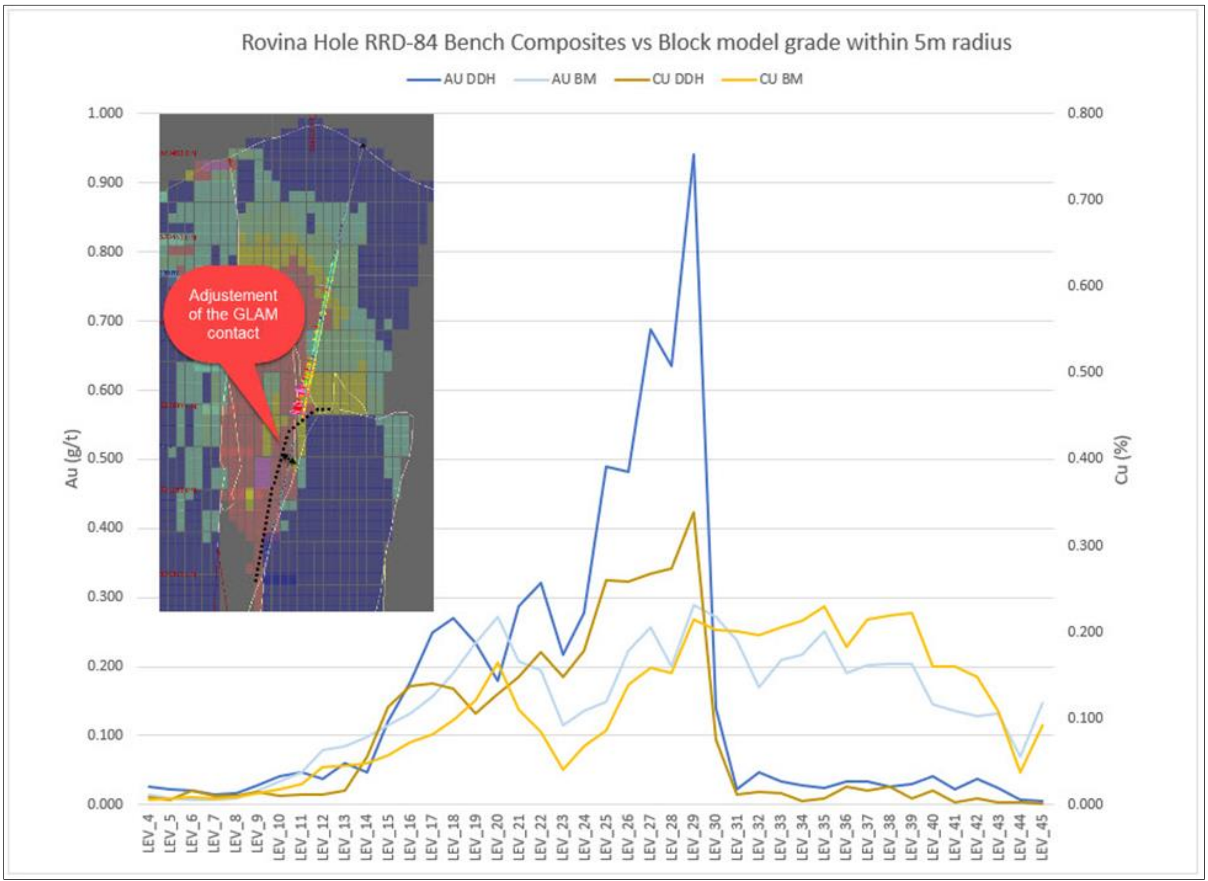


Figure 12.4: RRD-84 versus Resource Model

Table 12.6 shows the expected impact on the 2012 Resource Model due to the additional drillholes completed after the data cut-off date. Except for hole RRD-84, it is expected the new holes added would have no material impact on the 2012 Resource Estimate that would justify a complete update. Adjusting the wireframe for hole RRD-84 would only affect a small volume likely to be insignificant considering the total volume contained in the Rovina deposit.

Table 12.6: Impact of Drillholes added after the 2012 Data Cut-Off Date

Deposit	Hole-ID	Comments
Rovina	RRD-85	No material impact on the resource model expected
Rovina	RRM-1	No material impact on the resource model expected
Rovina	RRD-84	Minor impact expected below 448 m elevation affecting a very limited volume.
Colnic	RCD-106	No material impact on the resource model expected
Colnic	RCM-1	No material impact on the resource model expected
Ciresata	RDG-66	No material impact on the resource model expected (First 300 m)
Ciresata	RDG-68	No material impact on the resource model expected. Hole could possibly extend the mineralisation below the current resource model lower elevation with additional drilling.
Ciresata	RDG-69	No material impact on the resource model expected. Hole is also unlikely to extend the mineralisation below the current resource model lower elevation.
Ciresata	RDG-73	No material impact on the resource model expected

12.1.7 AGP Comments

No material sample bias was identified during the review of the drill data and assays. The data collected by ESM adequately represents the style of mineralisation present on the RVP. The error rate in the drill database for the data that was validated was found to be virtually 0 %. AGP regards the sampling, sample preparation, security, and assay procedures reviewed adequate to support the mineral resource without restriction on classification. Holes completed after the data cut-off date are unlikely to have a significant impact on the resources and AGP, therefore, recommended that ESM not update the 2012 grade or classification model until more holes are added to the resource.

12.2 CCIC MINRES ASSESSMENT 2020

Except for the sixteen geotechnical holes drilled at Colnic (3 holes), Rovina (1 hole) and plant and WMF area (12 holes) in October–December 2020, there was no new geological and sampling data added to the database since the comprehensive data verification by AGP, in 2012 and 2019.

To complement the data verification done by AGP, Mr Sivanesan Subramani of CCIC MinRes conducted a site visit from 9 to 12 November 2020 and completed the following reviews:

- An overview of the RVP geology, controls for mineralisation, exploration history, and data collection protocols was presented by Mr Randall Ruff (online presentation), in the attendance of Mr Dorin Groza, Dr Sorin Halga and Mr Albert Fuer. Mr Fuer accompanied Mr Subramani throughout the visit.
- A review of the core handling, core logging and sampling protocols. This was followed by visual checks and verification of the hardcopy logs against the drill core. A total of eight drillholes from the Rovina and Colnic deposits were selectively chosen by Mr Subramani to represent the important lithological and alteration types. The hole numbers were RRD-45, RRD-26, RRD-65, RRD-22, RCD-72, RCD-78, RCD-52, and RCD-60.
- A review of the electronic logs for the eight drillholes listed above. This included comparison of the electronic logs against the original hardcopy logs, plus visual verification of some significant Au and Cu grades against the drill core.
- A review of the survey of the topographic surface and survey of the drillhole collar position. This was followed by verification of the drillhole collar plaques in the field. A comparison of the collar position measured using a handheld GPS, against that in the database, which is measured by a survey total station, is shown in Figure 12.5. The 4 m difference in the easting is attributed to the difference in precision between a handheld GPS and total station. The drillhole collar elevations were also checked against the surveyed topographic surface in 3D and was found to be within a 0.5 m elevation difference.
- A review of the QA/QC programme and monitoring of the results.
- A visit to the ALS Chemex laboratory in Gura Rosiei to verify the sample preparation and analysis procedures. This included sample receiving, weighting, and registering into the system. Sample crushing, pulverising, and sub-sampling for fire assay; plus, the laboratory's internal QA/QC results (pinned on the notice board). Fire assay and analysis using both gravimetric and atomic absorption, as a check, plus, the

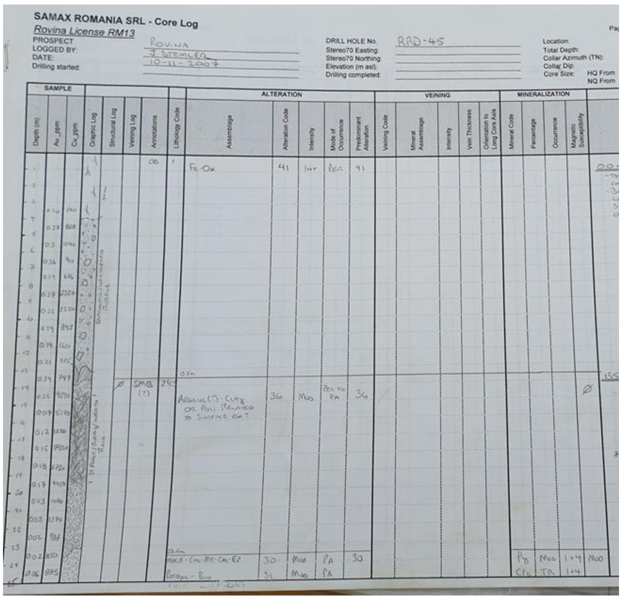
laboratory's internal and external "round-robin" QA/QC results (pinned on the notice board).

Some photographs taken during the site visit are presented in Figure 12.6.

Figure 12.5: Drillhole Collar Verification

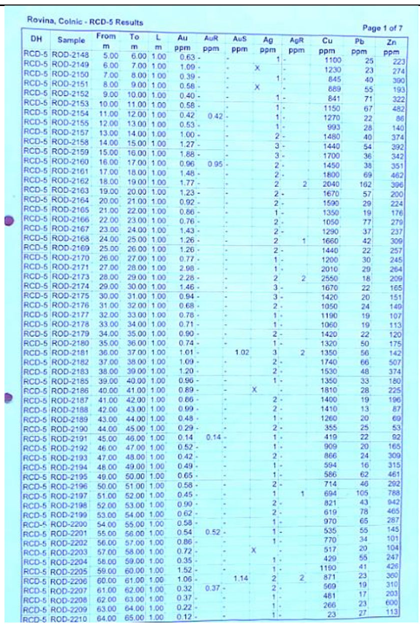

Close to outcrop at Rovina

Storage of pulp rejects



Hardcopy drill-core log


Pulp and coarse rejects storage



Rovina, Colnic - RCD-5 Results

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DH	Sample	From m	To m	L m	Au ppm	AuR ppm	AuS ppm	Ag ppm	AgR ppm	Cu ppm	Pb ppm	Zn ppm
RCD-5	ROD-2148	5.00	6.00	1.00	0.63	-	-	1	-	1100	25	223
RCD-5	ROD-2149	6.00	7.00	1.00	1.09	-	-	X	-	1230	23	274
RCD-5	ROD-2150	7.00	8.00	1.00	0.39	-	-	1	-	845	40	390
RCD-5	ROD-2151	8.00	9.00	1.00	0.58	-	-	X	-	889	55	193
RCD-5	ROD-2152	9.00	10.00	1.00	0.40	-	-	1	-	841	71	322
RCD-5	ROD-2153	10.00	11.00	1.00	0.58	-	-	1	-	1150	67	482
RCD-5	ROD-2154	11.00	12.00	1.00	0.42	0.42	-	1	-	1270	22	86
RCD-5	ROD-2155	12.00	13.00	1.00	0.53	-	-	1	-	993	28	140
RCD-5	ROD-2157	13.00	14.00	1.00	1.00	-	-	2	-	1480	40	374
RCD-5	ROD-2158	14.00	15.00	1.00	1.27	-	-	3	-	1440	54	392
RCD-5	ROD-2159	15.00	16.00	1.00	1.88	-	-	3	-	1700	36	342
RCD-5	ROD-2160	16.00	17.00	1.00	0.96	0.95	-	2	-	1450	38	351
RCD-5	ROD-2161	17.00	18.00	1.00	1.48	-	-	2	-	1800	69	462
RCD-5	ROD-2162	18.00	19.00	1.00	1.77	-	-	2	2	2040	162	396
RCD-5	ROD-2163	19.00	20.00	1.00	1.23	-	-	2	-	1670	57	200
RCD-5	ROD-2164	20.00	21.00	1.00	0.92	-	-	2	-	1590	29	224
RCD-5	ROD-2165	21.00	22.00	1.00	0.86	-	-	1	-	1350	19	176
RCD-5	ROD-2166	22.00	23.00	1.00	0.76	-	-	2	-	1050	77	279
RCD-5	ROD-2167	23.00	24.00	1.00	1.43	-	-	2	-	1290	37	237
RCD-5	ROD-2168	24.00	25.00	1.00	1.26	-	-	2	1	1660	42	309
RCD-5	ROD-2169	25.00	26.00	1.00	1.26	-	-	2	-	1440	22	257
RCD-5	ROD-2170	26.00	27.00	1.00	0.77	-	-	1	-	1200	30	245
RCD-5	ROD-2171	27.00	28.00	1.00	2.98	-	-	1	-	2010	29	264
RCD-5	ROD-2173	28.00	29.00	1.00	2.28	-	-	2	2	2550	18	209
RCD-5	ROD-2174	29.00	30.00	1.00	1.46	-	-	3	-	1670	22	165
RCD-5	ROD-2175	30.00	31.00	1.00	0.94	-	-	3	-	1420	20	151
RCD-5	ROD-2176	31.00	32.00	1.00	0.68	-	-	2	-	1050	24	149
RCD-5	ROD-2177	32.00	33.00	1.00	0.78	-	-	1	-	1190	19	107
RCD-5	ROD-2178	33.00	34.00	1.00	0.71	-	-	1	-	1060	19	113
RCD-5	ROD-2179	34.00	35.00	1.00	0.90	-	-	2	-	1420	22	120
RCD-5	ROD-2180	35.00	36.00	1.00	0.74	-	-	1	-	1320	50	175
RCD-5	ROD-2181	36.00	37.00	1.00	1.01	-	1.02	3	2	1350	56	142
RCD-5	ROD-2182	37.00	38.00	1.00	1.09	-	-	2	-	1740	66	507
RCD-5	ROD-2183	38.00	39.00	1.00	1.20	-	-	2	-	1530	48	374
RCD-5	ROD-2185	39.00	40.00	1.00	0.96	-	-	1	-	1350	33	180
RCD-5	ROD-2186	40.00	41.00	1.00	0.89	-	-	X	-	1810	28	225
RCD-5	ROD-2187	41.00	42.00	1.00	0.86	-	-	2	-	1400	19	196
RCD-5	ROD-2188	42.00	43.00	1.00	0.99	-	-	2	-	1410	13	87
RCD-5	ROD-2189	43.00	44.00	1.00	0.48	-	-	1	-	1260	20	69
RCD-5	ROD-2190	44.00	45.00	1.00	0.29	-	-	2	-	355	25	53
RCD-5	ROD-2191	45.00	46.00	1.00	0.14	0.14	-	1	-	419	22	92
RCD-5	ROD-2192	46.00	47.00	1.00	0.52	-	-	1	-	909	20	165
RCD-5	ROD-2193	47.00	48.00	1.00	0.42	-	-	2	-	866	24	309
RCD-5	ROD-2194	48.00	49.00	1.00	0.49	-	-	1	-	594	16	315
RCD-5	ROD-2195	49.00	50.00	1.00	0.65	-	-	1	-	586	62	461
RCD-5	ROD-2196	50.00	51.00	1.00	0.58	-	-	2	-	714	46	292
RCD-5	ROD-2197	51.00	52.00	1.00	0.45	-	-	1	1	694	105	788
RCD-5	ROD-2198	52.00	53.00	1.00	0.90	-	-	2	-	821	43	942
RCD-5	ROD-2199	53.00	54.00	1.00	0.62	-	-	2	-	619	78	465
RCD-5	ROD-2200	54.00	55.00	1.00	0.58	-	-	1	-	970	65	287
RCD-5	ROD-2201	55.00	56.00	1.00	0.54	0.52	-	1	-	535	55	145
RCD-5	ROD-2202	56.00	57.00	1.00	0.86	-	-	1	-	770	34	101
RCD-5	ROD-2203	57.00	58.00	1.00	0.72	-	-	X	-	517	20	104
RCD-5	ROD-2204	58.00	59.00	1.00	0.35	-	-	1	-	429	55	247
RCD-5	ROD-2205	59.00	60.00	1.00	1.52	-	-	1	-	1190	41	426
RCD-5	ROD-2206	60.00	61.00	1.00	1.06	-	1.14	2	2	871	23	360
RCD-5	ROD-2207	61.00	62.00	1.00	0.32	0.37	-	2	-	569	19	310
RCD-5	ROD-2208	62.00	63.00	1.00	0.37	-	-	1	-	481	17	203
RCD-5	ROD-2209	63.00	64.00	1.00	0.22	-	-	1	-	266	23	600
RCD-5	ROD-2210	64.00	65.00	1.00	0.12	-	-	1	-	23	27	113

Electronic sampling log

Example of potassic and phyllic alteration

Core logging facility

Core logging and storage facility

Figure 12.6: Some Photographs from the CCIC MinRes Site Visit

It is CCIC MinRes's opinion that the data collection and verification procedures implemented by ESM are to industry standards. Although CCIC MinRes makes recommendations for the improved monitoring of QA/QC results, CCIC MinRes is of the opinion that the geological database is of sufficient reliability for use in mineral resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

In February 2019, AGP completed an NI 43-101 PEA on the RVP. In 2020, SENET was requested to complete a DFS on the project as a follow-up to the PEA. SENET reviewed the PEA test work results and proposed a metallurgical test work programme whose results would support the process flowsheet selection. Various laboratories were selected to carry out the DFS test work:

- SGS Canada – Comminution test work (variability and composite samples test work) – Bond Ball Work Index (BBWi), Bond Rod Work Index (BRWi), Abrasion Index (Ai), Semi autogenous grinding Mill Comminution (SMC), Crushability Work Index (CWi), Uniaxial Compressive Strength (UCS)
- Orway Mineral Consultants (OMC) – Interpretation of comminution results and simulation of the comminution circuit
- Pocock – Solid-liquid separation for dry stacking of the final tailings and transport moisture limit for conveyability
- KCB – Geophysical test work
- ARD Global – Geochemical test work

13.1 REVIEW OF THE HISTORICAL TEST WORK

Comminution and bench-scale flotation testing and a flotation pilot plant test programme were performed to investigate the major geometallurgical domains on the RVP. The samples were prepared from core samples to represent each of the domains with respect to copper and gold grade, lithology, and composition. The PEA test work was performed at the following laboratories:

- SGS
- Eriez
- Xstrata

13.1.1 SGS May 2007 Report

This section summarises the report: SGS, 1 May 2007, “An investigation into the recovery of gold and copper from a Au/Cu Porphyry Deposit”, Report 11413-001.

The aim of the test work was to investigate the recovery of copper and gold. Flotation optimisation tests were conducted to determine the best flotation circuit for maximum copper and gold recovery. The optimisation test work included the following:

- Effect of grind, pH and reagents on flotation
- Comparing selective flotation on the ROM against bulk rougher flotation followed by selective flotation
- Effect of regrind and cleaning

The test work was done on samples from Colnic, Rovina and Ciresata.

The test work indicated that the best flotation circuit consisted of a primary grind of 80 % passing 75 µm followed by bulk flotation of sulphides with a mixed xanthate/dithiophosphate collector and methyl isobutyl carbinol (MIBC) frother, then fine grinding the bulk rougher concentrate to 80 % passing 20 µm followed by selective flotation to reject the pyrite producing a saleable copper concentrate containing significant gold. Gold losses consist of that gold which is locked in silicates in the rougher tailings and that gold which is associated with or within pyrite in the cleaner tails. As the majority of gold losses were found to occur in the cleaner tailings, cyanidation of this stream was investigated.

Following establishment of the flotation circuit, locked cycle tests (LCT) were performed to get an indication of the recovery and concentrate grades that can be achieved. Table 13.1 shows a summary of the LCT results.

Table 13.1: Summary of the LCT Results

Test Work			RRD 2-3	RCD 4-5	RCD 3-10
Head Grade	-	Cu (%)	0.42	0.17	0.24
		Au (g/t)	0.71	1.26	1.53
		S^{2-} (%)	2.01	2.88	4.24
LCTs	Recovery	Cu (%)	92.0	90.1	84.9
		Au (%)	65.3	70.2	56.3
	Final Concentrate Grade	Cu (%)	26.8	20.3	21.9
		Au (g/t)	28.4	94.3	97.6
	Leach on 1st Cleaner Tails	Au dissolution	83.0	76.2	62.8
		CN consumption (kg/t)	0.98	1.06	0.71
		Lime consumption (kg/t	1.75	0.76	0.87
Overall Au recovery (flotation and tails leach)			79.5	78.0	70.1

The test work results indicated the following:

- The LCT showed that the ore is amenable to flotation, and saleable concentrate grades can be produced (20.3 % Cu to 26.8 % Cu).
- The LCT tests showed copper recoveries ranging from 84.9 % to 92.0 % and gold recoveries ranging from 56.3 % to 70.2 %.
- Cyanidation on the first cleaner tails showed gold dissolutions ranging from 62.8 % to 83.0 %.
- Overall gold recovery from flotation and leach on the float tails ranged from 70.1 % to 79.5 %.
- A multi-element analysis of the LCT products highlighted that, with the exception of zinc in the concentrate from Colnic, all deleterious elements are within treatment allowances. A high zinc concentration in the Colnic flotation concentrate (1.6 %) should be monitored as test work progresses. In practice, blending with low zinc concentrates from Rovina and Ciresata would dilute this element to acceptable (sub-penalty) levels.

13.1.2 SGS September 2008 Report

This section summarises the report: SGS, 15 September 2008, “A Deportment Study of Gold in composite Met 10 and composite Mett 11”, Report 11992-001.

The aim of the test work was to conduct a mineralogical investigation (gold deportment) on samples from the RVP. Two samples were investigated, namely Met 10 (Ciresata) and Met 11 (Colnic). Since the Rovina Valley DFS did not include the Ciresata deposit, test work on the Ciresata samples was not described in detail in this report.

Mineralogy on Met 11 (Colnic) showed the following:

- Gold grains range in size from 1 µm to 49 µm with an average size of 10 µm. The gold grains are liberated, attached or locked grains. Gold occurs as native gold.
- The gravity recovery method can pre-concentrate 30 % to 40 % of the native gold, which is mainly liberated or exposed and can be recovered by cyanidation. The rest of the gold may be partly recovered after fine grinding. The large amount of gold particles locked in silicates may decrease the extraction of gold.
- The balance of the distribution of gold is considered to be fine (and/or sub-microscopic) gold locked in non-opaque minerals (mainly silicates) and locked in sulphides which are disseminated in non-opaque minerals or in Fe-oxides. Locked gold accounts for over 55 % of the gold mass in the samples. Locking in non-opaque minerals is considered to be the major factor that may affect complete gold recoveries.

13.1.3 SGS Geosol February 2009 Report

This section summarises the report: SGS Geosol, 28 February 2009, “Grindability, Flotation and Leaching Test Work for Gold Ore Samples from Colnic, Rovina and Ciresata”.

The aim of this test work was to establish the comminution (BBWi) and metallurgical characteristics of several composites. All composites were submitted for rougher kinetics flotation testing while batch cleaner testing and cyanidation of flotation tailings were conducted exclusively on the Ciresata composites (MET-17 and MET-18). Table 13.2 shows a summary of the SGS Geosol testwork results.

Table 13.2: Results Summary – SGS Geosol

Test Work		Colnic	Colnic	Rovina	Rovina	Rovina	Ciresata	Ciresata
		Met 12	Met 13	Met 14	Met 15	Met 16	Met 17	Met 18
BBWi	kWh/t	15.3	15.6	14.1	13.6	14.3	16.7	17.3
Head Grade	Cu (%)	0.10	0.12	0.25	0.32	0.21	0.17	0.15
	Au (g/t)	0.39	0.61	0.25	0.36	0.19	0.91	0.56
Rougher Recovery	Cu (%)	91	92	93	93	90	94	88
	Au (%)	76	80	76	84	77	74	67
Cleaner Recovery	Cu (%)	–	–	–	–	–	98	97
	Au (%)	–	–	–	–	–	87	77
Cleaner	Cu (%)	–	–	–	–	–	14.5	15.6

Test Work		Colnic	Colnic	Rovina	Rovina	Rovina	Ciresata	Ciresata
		Met 12	Met 13	Met 14	Met 15	Met 16	Met 17	Met 18
concentrate grade	Au (%)	–	–	–	–	–	50.2	37.9
Leach on Rougher Tails	Carbon in leach (CIL) no regrind (% Au)	–	–	–	–	–	83	81
	CIL with regrind to 80 % passing 45 µm (% Au)	–	–	–	–	–	85	88

The test work results indicated the following:

- BBWi on the samples ranged from 13.6 kWh/t to 17.3 kWh/t, showing that the samples are medium hard to hard in terms of ball milling.
- Excluding Ciresata, rougher copper recovery was above 90 % for all the samples tested, with recoveries ranging from 90 % to 94 %.

13.1.4 SGS June 2010 Report

This section summarises the report: SGS, 21 June 2010, “The Flotation Testing of Ores from the Rovina Valley Project”, 12342-001.

Samples from Rovina, Colnic and Ciresata were received for a flotation test work programme. The flotation programme included a brief series of scoping tests (rougher kinetics and batch cleaner) followed by LCT of each composite.

The programme objectives were to

- Test an open circuit flowsheet, with the hopes of improving LCT stability.
- Explore the effectiveness of lower reagent additions.
- Test the effect of less selective sulphide flotation, thereby recovering more pyrite-locked gold to final concentrate.
- Provide LCT results for metallurgical predictions.

Table 13.3 shows the results summary.

Table 13.3: Results Summary – SGS June 2010

Test Work			Rovina	Colnic	Ciresata
Head Grade		Cu (%)	0.25	0.11	0.16
		Au (g/t)	0.28	0.71	0.95
LCT	Cleaner Recovery	Cu (%)	93.1	91.8	88.9
		Au (%)	67.9	67.7	66.4
	Cleaner Grade	Cu (%)	25.4	17.7	24.2
		Au (g/t)	20.3	81.8	106
	LCT Conditions		Rougher (Ro) P_{80} = 86 µm,	Ro: P_{80} = 67 µm, pH 10.5	Ro: P_{80} = 71 µm, pH 9.5

Test Work		Rovina	Colnic	Ciresata
		pH 9.2		
		Cl: P_{80} = 20 µm, pH 10.2–11.0	Cl: P_{80} = 11 µm, pH 10.8–11.6	Cl: P_{80} = 17 µm, pH 10.6–11.8
		3418A 5.5 g/t, Potassium Amyl xanthate (PAX) 5 g/t	3418A 6.5 g/t, PAX 5 g/t	3418A 5.5 g/t, PAX 5 g/t
		Sodium Isobutyl Xanthate (SIBX) 0 g/t, Lime720 g/t	SIBX 0 g/t, Lime 1,100 g/t	SIBX 0 g/t, Lime 650 g/t
		MIBC 20 g/t	MIBC 20 g/t	MIBC 20 g/t

The test work results indicated the following:

- Cleaner flotation recovery was 93.1 % Cu and 91.8 % Cu for Rovina and Colnic, respectively.
- Cleaner concentrate grade was 25.4 % Cu and 17.7 % Cu for Rovina and Colnic, respectively
- LCT did highlight the sensitivity of these ores to overdosing of collector. In particular, the use of 3418A in LCT appeared to prevent the effective depression of pyrite in the cleaner circuit across a wide range of pulp pH.

13.1.5 Xstrata May 2013 Report

This section summarises the report: Xstrata, 27 May 2013, “Geometallurgical Study of the Rovina Valley Deposit”.

The aim of the test work was to conduct a geometallurgical study on the RVP. The geometallurgical study included mineralogical and metallurgical characterisation of the Rovina Valley deposit.

Five geometallurgical units were selected based on collaboration with project geologists. The units included in the programme were Rovina, Colnic K1, Colnic K2K3, Ciresata Sediment and Ciresata Porphyry.

The mineralogical test work indicated the following:

- Modal analyses show that all the units are dominated by quartz and feldspar.
- Sulphide mineralogy consists primarily of pyrite and chalcopyrite with minor amounts of pyrrhotite in the Colnic (MET-30, MET-31) and Rovina (MET-32) units. Of note is the ratio of pyrite to chalcopyrite, which ranges from 3.5 in MET-27 up to 19.8 in MET-30. Trace levels of sphalerite were detected in MET-30, 31 and 32.
- The chalcopyrite average grain size is consistent between the Ciresata and Colnic units, ranging between 27 µm and 30 µm. The average chalcopyrite grain size in the Rovina composite is coarser, at 37 µm.
- Microprobe analysis indicates that chalcopyrite is the sole copper carrier in four of the five geometallurgical units. Low levels of copper were detected in some chlorite grains (average 0.08 %) within the Ciresata Sediment unit (MET-27).

When combined with the modal abundance, a copper deportment of 98.2 % within chalcopyrite and 1.8 % within chlorite is produced. Copper deportment in the remaining geometallurgical units is 100 % within chalcopyrite.

The metallurgical test work involved LCTs on the geometallurgical units.

Table 13.4 shows a summary of the Xstrata test work results.

Table 13.4: Summary Test Work Results – Xstrata May 2013

Test Work		Met 32 (Rovina)					Met 33 (Rovina/ Colnic Blend)		Met 34 (Colnic K1+K2)	
		LCT 001	LCT 002	LCT 003	LCT 004	LCT 005	LCT 001	LCT 002	LCT 001	LCT 002
Final concentrate recovery	Cu (%)	83.4	85.8	78.0	81.0	86.0	79.3	81.7	78.3	89.7
	Au (%)	61.7	62.1	51.6	56.9	63.5	61.6	67.4	58.9	72.2
Final concentrate grade	Cu (%)	10.4	10.7	26.2	15.6	21.0	16.2	6.5	16.2	11.9
	Au (g/t)	9.7	11.9	22.9	16.5	22.0	52.5	26.8	91.8	60.9

The LCT included modifications to the conditions to try and improve performance.

13.1.5.1 Rovina

Flowsheet modifications achieved the Cu and Au entitlements for the rougher-scavenger circuit but at a higher mass pull than the target. Among the process modifications were the following:

- Addition (LCT001) and elimination (LCT003) of sodium silicate, which was deemed ineffective.
- Addition (LCT002) and elimination (LCT003) of the Alcomer 74L thinning reagent, which was deemed ineffective.
- Switching from ISA milling to mild steel in the regrind (LCT003). Also, switching from stainless and mild steel grinding media to exclusively mild steel grinding media (LCT004). Neither of these had an obvious effect.
- Institution (LCT002) and elimination (LCT004) of split rougher-scavenger cleaning circuits. Operated properly, this circuit has great potential. However, one can argue that the individual concentrates do not offer any advantage over an integrated circuit. Certainly, if one were to add flotation stages to the combined cleaner circuit, one could improve the circuit without resorting to an entire split cleaning circuit.
- Changing to a milder frother (MIBC with a small amount of polyglycol) and moving to more powerful collectors (AF238 and PAX) in LCT003. At the same time, moving a small amount of PAX to the front of the circuit in the grind. No obvious differences were seen.

13.1.5.2 Colnic

The final LCT in this series used a modified version of the previous flowsheet, in which a third scavenger cleaner was added, along with recirculation of the Scavenger (“Cleaner 1A”) concentrate.

13.1.6 Recovery Modelling

Modelling of the LCT tests showed good, predicted recoveries hence the motivation to do a flotation pilot plant. Table 13.5 shows a summary of the predicted recoveries and concentrate.

Table 13.5: Predicted Recoveries and Concentrate Grades

Parameter		Value
Head Grade	Cu (%)	0.32
	Au (g/t)	0.23
Final Concentrate Recovery	Cu (%)	93.81
	Au (%)	74.74
Final Concentrate Grade	Cu (%)	22.65
	Au (g/t)	13.66

13.1.7 Eriez November 2018

This section summarises the report: Eriez, 1 November 2018, “Euro Sun Mining, Rovina Valley Project Copper and Gold Flotation Pilot Plant Report”, Report Number MTR 17-266.

Three composite samples of core from the Colnic and Rovina regions, designated MET 42 (Colnic K1 Domain), MET 43 (Rovina Domain) and MET 44 (Colnic K2K3 Domain), weighing approximately 3 t each were received for the flotation pilot plant. Figure 13.1, Figure 13.2 and Figure 13.3 show a diagrammatic representation of Met 42, Met 43 and Met 44 sampling, respectively.

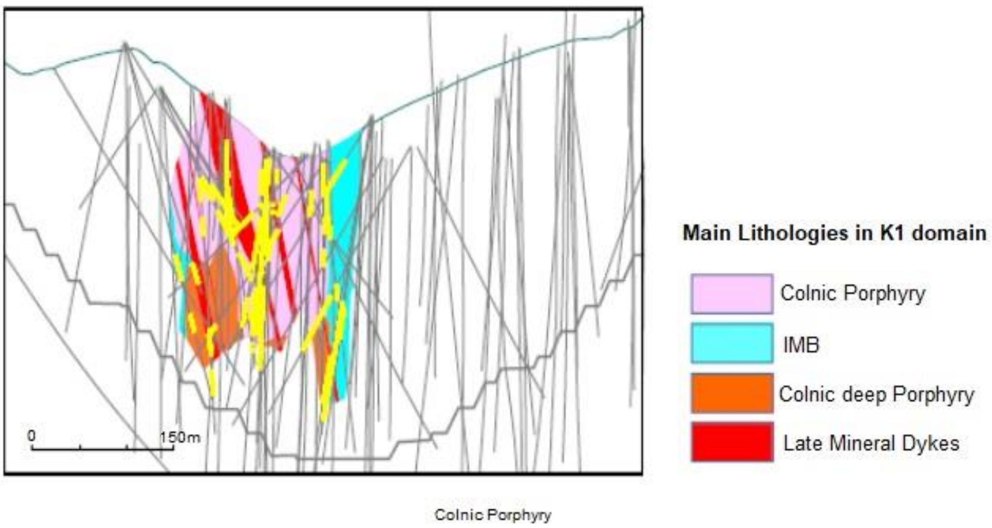


Figure 13.1: K1 Alteration in Colnic: Cross Section at 150° looking NW (Met 42 in yellow)

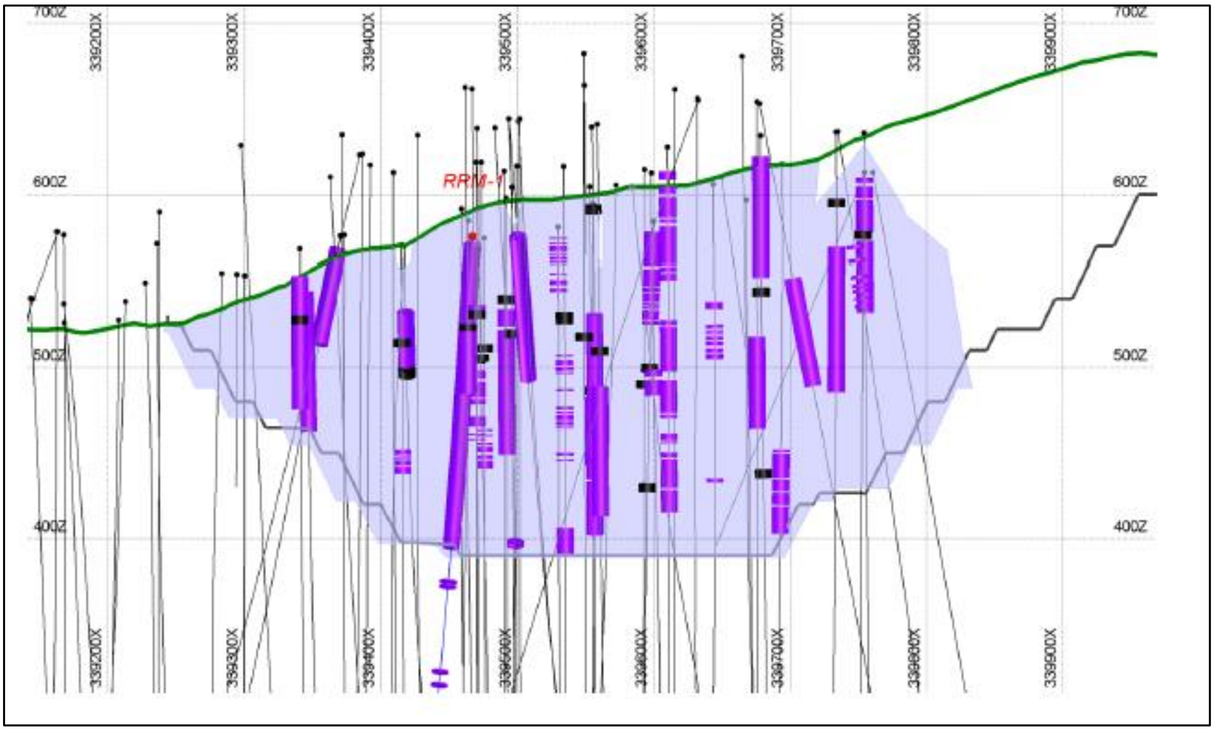


Figure 13.2: Met 43 in Purple – Previous Metallurgical Samples Selected in 2016 in Black (Met 40 and Met 41)

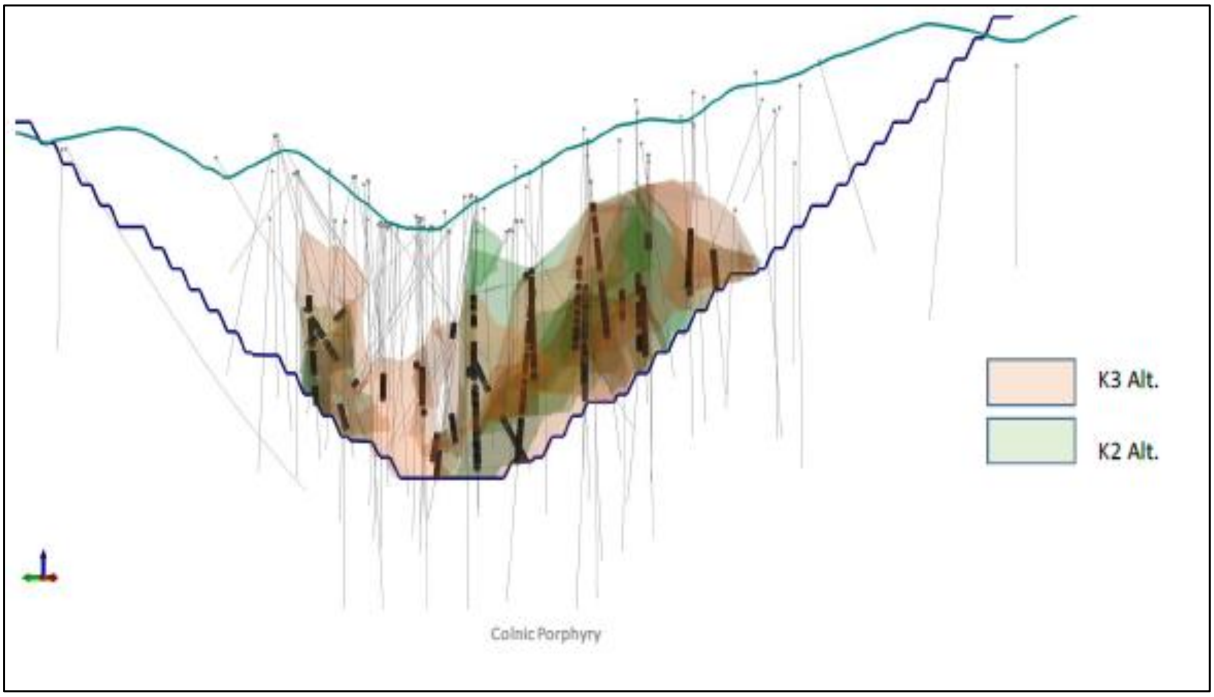


Figure 13.3: Met 44 in Black – K2-K3 Alteration Domain in Colnic: Cross Section at 150° looking NW Pit Outline

The aim of the pilot plant was to investigate the recovery of gold, copper and zinc using the Eriez column flotation technology. Comminution tests were also performed on the composite samples.

Following optimisation of the rougher and scavenger operating parameters, including feed rate, air rate and wash-water rate, prepared ore samples of all types (MET 42, 43, and 44) were treated using the Eriez pilot column cells under steady-state conditions to generate a bulk cleaner circuit feed material. Three stages of cleaning for MET 42 and MET 44, and two stages of cleaning for MET 43 were carried out on the rougher-scavenger bulk concentrates.

When comparing the corresponding bench scale mechanical cell flotation results with the column flotation results, the copper grade and mass recovery improved using column flotation with wash-water addition. In the pilot plant test work, the rougher concentrate was fine milled to 80 % passing 13.5 µm to 13.8 µm as opposed to the 80 % passing 20 µm used in the previous test work. This was done to improve the liberation of gold from pyrite and thereby decrease gold losses. Table 13.6 shows a summary of the pilot plant results.

Table 13.6: Summary of the Pilot Plant Results

Parameter		Met 42 (Colnic)		Met 43 (Rovina)		Met 44 (Colnic K2K3)	
		Test 1	Test 2	Test 1	Test 2	Test 1	Test 2
Head Grade	Cu (%)	0.14	0.15	0.3	0.32	0.13	0.13
	Au (g/t)	0.76	0.78	0.22	0.23	0.55	0.55
Rougher + Scavenger Recovery	Cu (%)	93.17	93.02	95.94	94.40	96.30	96.46
	Au (g/t)	84.68	85.08	78.93	78.78	88.65	88.63
3rd Cleaner Recovery	Cu (%)	83.4	82.4	95.5	94	94.7	93.9
	Au (%)	77.3	78.1	70.6	75.7	86.0	85.1
3rd Cleaner Concentrate Grade	Cu (%)	21.74	22.6	19.10	22.65	21.28	21.13
	Au (g/t)	107.1	111.4	10.7	13	82.9	83
Cleaner Mass Pull		0.5	0.5	1.5	1.3	0.6	0.6
Specific Gravity (SG)		–		–		2.67	
A × b		–		–		24.7	
t 1 a		–		–		0.24	
BRWI	kWh/t	–		18.6		18.3	
BBWi	kWh/t	16.6		16		16.6	
Ai	g	–		–		0.446	

The pilot plant results indicated the following:

- Final copper concentrate grades from the flotation pilot plant ranged from 21.13 % Cu to 22.65 % Cu.
- Final copper recoveries ranged from 82.4 % to 95.5 % and gold recoveries ranged from 70.6 % to 86.0 %.
- For the high Zn content ores, further investigation on reagents schemes for high zinc content ores is recommended.
- Comminution test work showed that the Colnic ore has Bond Ball Work indices of 16.6 kWh/t, which shows that the ore is hard.

- An A × b value of 24.7 indicates that the ore is very hard in terms of Semi Autogenous Grinding (SAG) or Autogenous Grinding (AG) milling. For SAG milling, a recycle pebble crusher will likely be required to handle the pebbles.
- A bond abrasion value of 0.446 g shows that the ore is abrasive.

13.2 TEST WORK GAP ANALYSIS

Comprehensive metallurgical test work has been conducted over the years on the composite samples of the Rovina Valley orebodies to support the studies conducted in 2010, 2012 and the latest PEA conducted in 2019. Historical work included preliminary evaluation of grindability, mineralogical gold deportment, geometallurgical populations and mineralogy, and bench-scale flotation (batch and locked cycle). In 2019, a column flotation pilot campaign was conducted on three composite samples representing the Colnic K1 domain, Rovina domain and Colnic K2K3 domain. Since all historical test work conducted to support the process flowsheet was based on composite samples only, SENET, therefore, proposed a comprehensive metallurgical test work programme to establish the degree of comminution characteristics within the ore domains, solid-liquid separation, and transport moisture limits tests for filter cake conveying and stacking.

It should be noted that the flotation or recovery variability and concentrate solid-liquid separation test work could not be conducted due to a lack of samples; therefore, it was agreed that the column flotation pilot plant results would be used for the flotation plant design. The cleaner tails solid-liquid separation results will be applied for the concentrate equipment design.

13.3 DFS TEST WORK

Comminution, solid-liquid separation tests, geochemical and geotechnical tests were performed in the DFS.

Figure 13.4 shows the comminution test work conducted at SGS Canada.

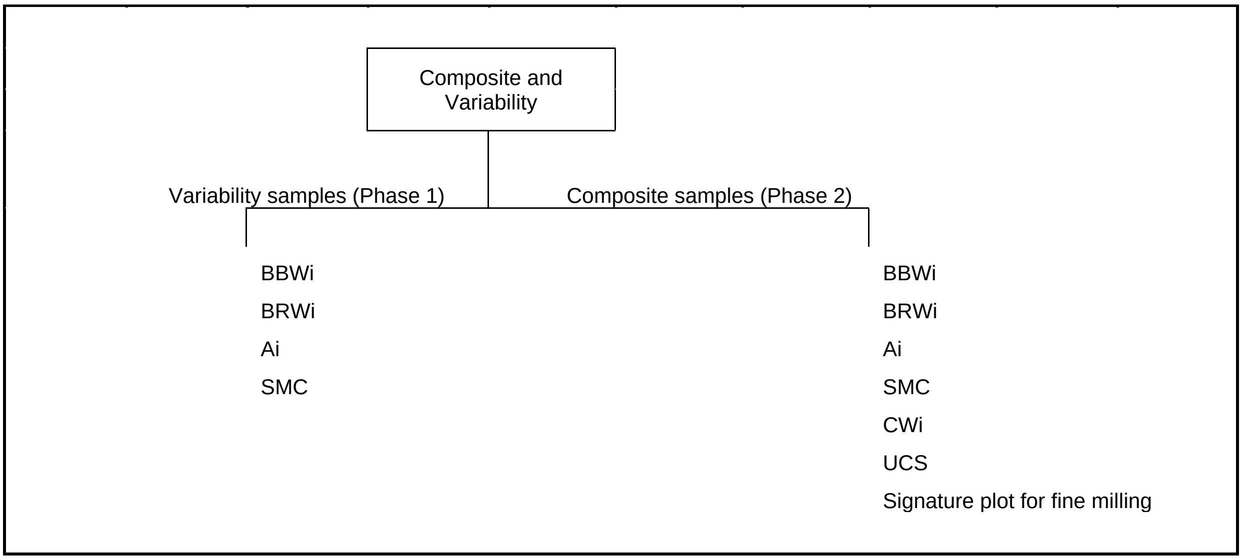


Figure 13.4: Comminution Test Work

Figure 13.5 shows the solid-liquid separation test work that was conducted at Pocock.

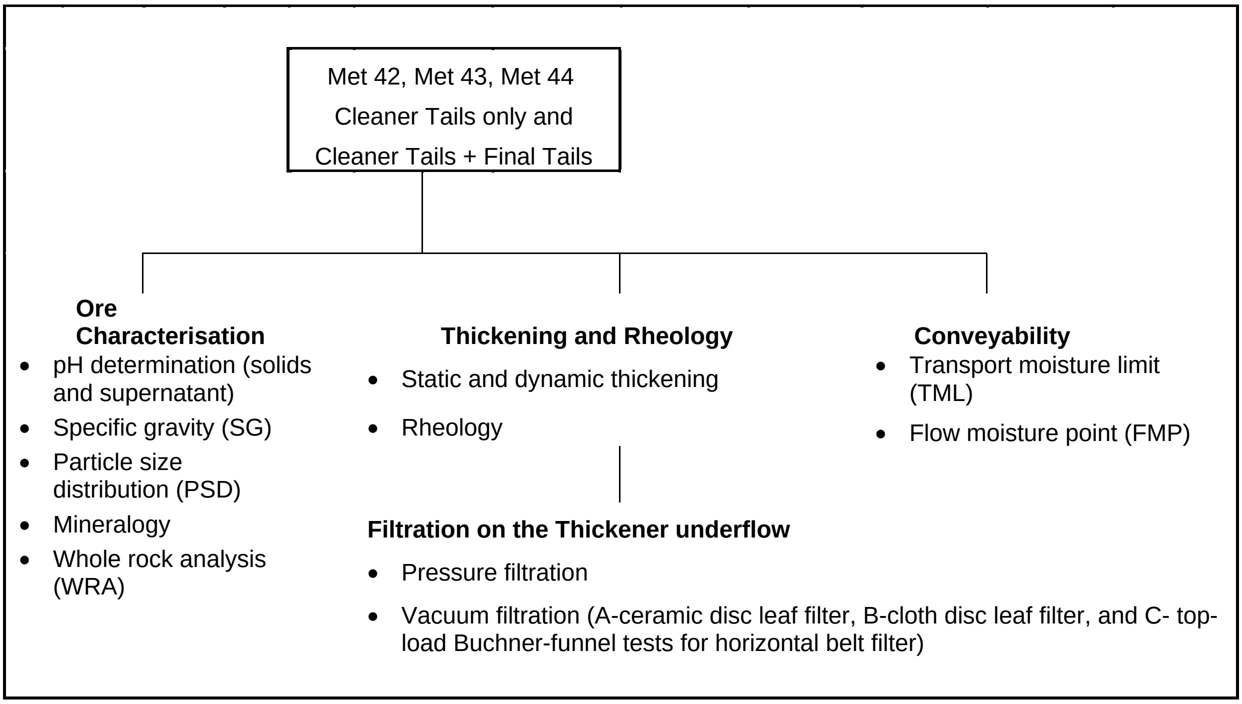


Figure 13.5: Solid-Liquid Separation Test Work

Geochemical and geophysical test work was conducted at KCB and ARD Global

13.3.1 Sample Selection

OMC was requested to perform a comminution sample selection to satisfy a DFS. The approach used for sample selection was to sample and test the orebodies by lithology, taking the oxidation state and weathering into consideration. OMC recommended a comminution test work programme of 24 samples (4 composite + 20 variability). The 20 variability samples were split as 12 from Colnic pit and 8 from Rovina Valley pit. The variability samples were readily available on site and test work on these samples could start first (Phase 1) while the test work on the composite samples, which were being drilled, was done in Phase 2.

13.3.1.1 Phase 1: Variability Sample Selection

The guidelines for sample selection using basic scheduling of lithology variation with mining sequence are shown in Table 13.7 for the Colnic lithology and Table 13.8 for the Rovina Lithology.

Table 13.7: Colnic Lithology

Period	Colnic POR	IMB	LM Dykes	F2 Hill POR	Colnic Deep POR	Dist.
1	68 %	23 %	10 %			101 %
2	43 %	45 %	12 %			100 %
3	51 %	39 %	10 %			100 %

Period	Colnic POR	IMB	LM Dykes	F2 Hill POR	Colnic Deep POR	Dist.
4	35 %	42 %	9 %	15 %		101 %
5	26 %	52 %		22 %		100 %
6	21 %	46 %		33 %		100 %
7	14 %	42 %		44 %		100 %
8	17 %	39 %		43 %		99 %
9	21 %	34 %		45 %		100 %
10	20 %	27 %		40 %	13 %	100 %
11	17 %	14 %		38 %	23 %	92 %
Overall Distribution	**30 %**	**37 %**	**4 %**	**26 %**	**3 %**	**100 %**
Sample Distribution	33 %	42 %	0 %	25 %	0 %	100 %
Recovery Variability Sample Count	**4**	**5**	**0**	**3**	**0**	**12**
	Composite					
	Variability					

Table 13.8: Rovina Lithology

Period	POC	IMB	POB	Glam Brx	OB	Dist
1	50 %	1 %	16 %	10 %	23 %	100 %
2	65 %	6 %	12 %	10 %	6 %	99 %
3	63 %	10 %	15 %	10 %	1 %	99 %
4	63 %	17 %	10 %	10 %	0 %	100 %
5	56 %	24 %	11 %	9 %	0 %	100 %
6	53 %	28 %	10 %	9 %	0 %	100 %
7	65 %	16 %	13 %	6 %	0 %	100 %
8	61 %	0 %	34 %	5 %	0 %	100 %
Overall Distribution	**60 %**	**13 %**	**15 %**	**9 %**	**4 %**	**100 %**
Sample Distribution	63 %	13 %	13 %	13 %	0 %	100 %
Rec. Var. Sample Count	**5**	**1**	**1**	**1**	**0**	**8**
	Composite					
	Variability					

Figure 13.6 and Figure 13.7 show diagrammatic representations of where the comminution variability samples were taken from the Colnic and Rovina Valley deposits, respectively.

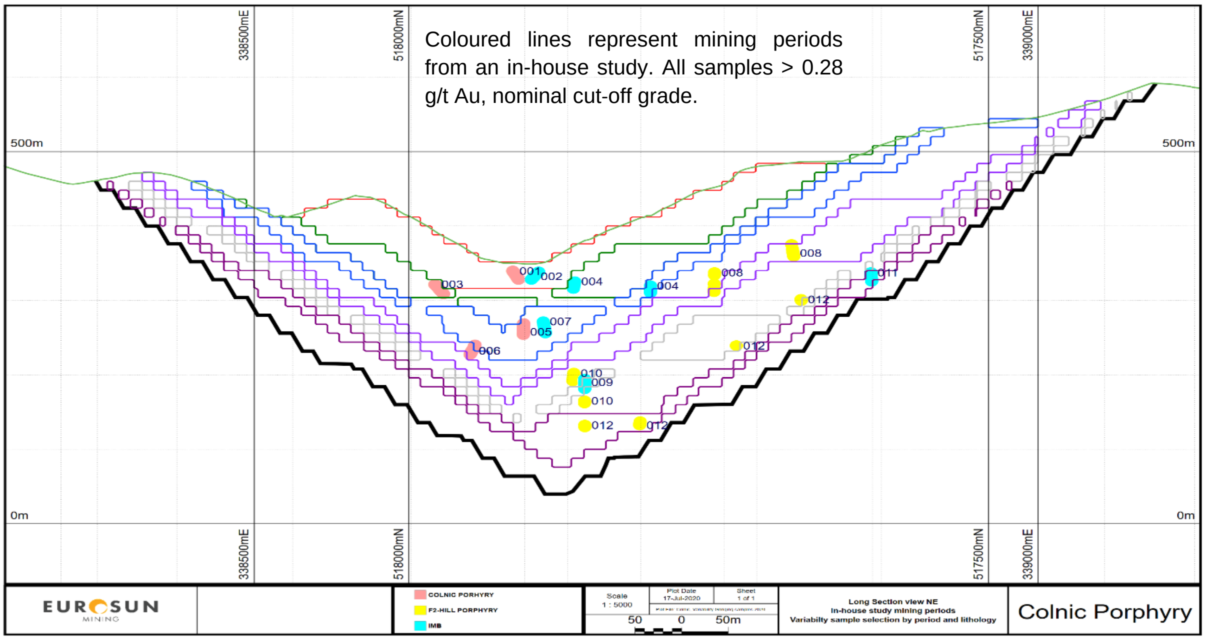


Figure 13.6: Colnic Samples

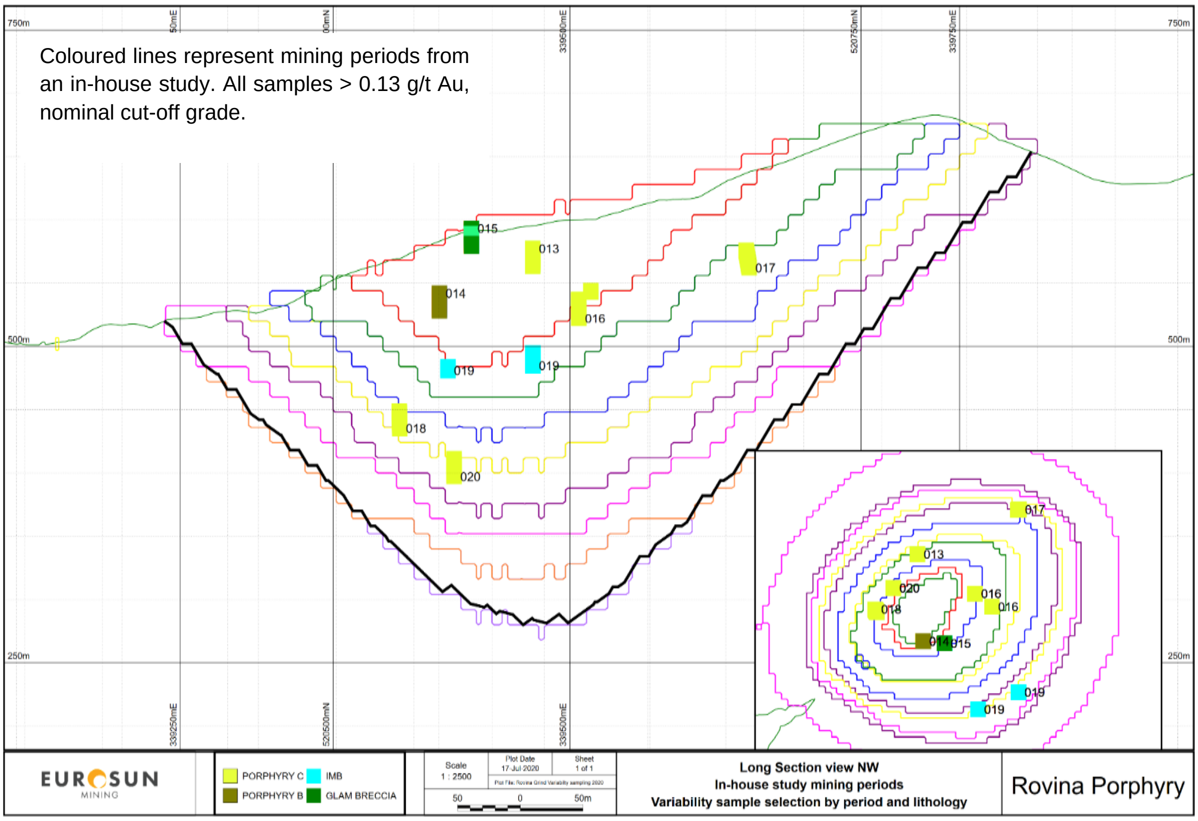


Figure 13.7: Rovina Samples

13.3.1.2 Phase 2: Composite Sample Selection

Sample selection criteria were based on the following:

- Major lithologic types
- Mining sequence (spatial distribution)

Three samples were collected from the Colnic deposit to represent the IMB, Colnic POR and F2 Hill POR lithologies.

One sample was collected from the Rovina deposit to represent the POC lithology.

Table 13.7 and Table 13.8 show the composting for both Colnic and Rovina samples.

There were no available appropriate samples from Rovina to provide the composite sample. Therefore, no comminution tests were performed on the Rovina composite ore.

Figure 13.8 shows a cross section of the Colnic deposit, highlighting where the variability samples were taken from.

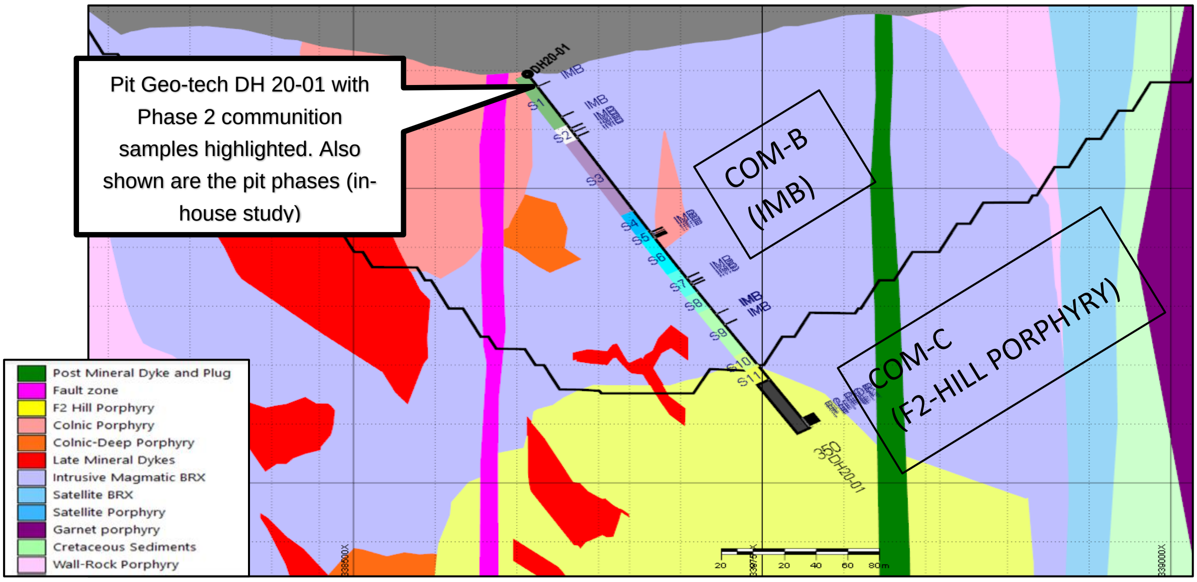


Figure 13.8: Colnic Deposit – Cross Section at 75°

13.3.2 Summary of the DFS Test work

This section shows a summary of the DFS test work results. The detailed test work results, including the test work procedures, can be seen in the following test work reports.

- SGS, 19 October 2020, "The Grindability Characteristics of Twenty Variability Samples from Rovina Project", Report No.18109-01.
- SGS, 19 October 2020, "The Grindability Characteristics of Three Composite Samples from Rovina Project", Report No.18109-02.

- Pocock, October 2020, “Flocculant Screening, Gravity Sedimentation, Paste Rheology, Vacuum Filtration and Pressure Filtration Studies Conducted for Euro-Sun, Rovina Valley Project”.
- Lawrence Consulting Ltd, 15 January 2021 “Rovina Valley Project – ARD Characterization of flotation Tailings by Static Testing”, Report No. LCL 20.059.

13.3.2.1 Comminution Test Work on Variability Samples

Table 13.9 shows a summary of the variability comminution results.

Table 13.9: Summary – Variability Comminution Results

Testwork		Min.	Max.	Median	85th Percentile
SMC	Value (No units)	24.0	55.2	28.0	36.3
	Classification	Very Hard	Medium Hard	Hard	Hard
	Lithology	IMB	GLAMM BRX	–	–
BBWi (106 µm)	Value (kWh/t)	11.8	19.2	15.9	18.1
	Classification	Medium Hard	Hard	Medium Hard	Hard
	Lithology	GLAMM BRX	IMB	–	–
Ai	Value (g)	0.0970	0.5210	0.3165	0.4195
	Classification	Low Abrasive	Abrasive	Medium Abrasive	Medium Abrasive
	Lithology	POC	C-POR	–	–

The comminution test work indicated the following values:

- SMC A× b values ranged from 24.0 to 55.2, showing that the ore ranges from very hard to medium hard for SAG milling. The values also indicate that the ore is a potential candidate for SAG milling.
- BBWi values at 106 µm ranged from 11.8 kWh/t to 19.2 kWh/t, showing that the ore is medium hard to hard.
- Ai values ranged from 0.0970 g to 0.5210 g, showing that the ore is low abrasive to abrasive.

13.3.2.2 Comminution Test Work on Composite Samples

Table 13.10 shows a summary of the comminution results on the composite samples.

Table 13.10: Summary – Composites Comminution Results

Sample Identity			Relative Density	JK Parameters			Work Indices (kWh/t)			Ai	UCS	IsaMill Fine Grinding
Deposit	Lithology	ID		A × b	t_a	SCSE	CWi	BRWi	BBWi	g	MPa	kWh/t
Colnic	POR	COM-A	2.68	24.0	0.23	12.6	15.9	18.0	17.0	0.457	137.0	-
	IMB	COM-B	2.63	25.8	0.25	12.0	14.2	18.8	18.5	0.378	92.6	-
	F2 Hill POR	COM-C	2.67	24.0	0.23	12.6	11.2	18.5	17.2	0.416	146.2	-
Colnic	33 % POR, 42 % IMB,	Composite	-	-	-	-	-	-	-	-	-	61.2

Sample Identity			Relative Density	JK Parameters			Work Indices (kWh/t)			Ai	UCS	IsaMill Fine Grinding
Deposit	Lithology	ID		A × b	t_a	SCSE	CWi	BRWi	BBWi	g	MPa	kWh/t
	25 % F2 Hill											
Rovina	100 % POC	Composite	-	-	-	-	-	-	-	-	-	76.1

For the composite samples, only samples from Colnic deposit were tested. There were no samples from Rovina Valley available for testing.

The comminution test work on the Colnic samples indicated the following:

- SMC A × b values range from 24.0 to 25.8. This is in line with the variability results, which showed A × b values ranging from 25.0 to 34.5. The composite sample test work indicates that the ore is hard in terms of SAG milling and that the ore is a potential candidate for SAG milling.
- BBWi values range from 17.0 kWh/t to 18.5 kWh/t. This is in line with the variability results, which ranged from 13.9 kWh/t to 19.2 kWh/t. The composite test work indicates that the ore is hard in terms of ball milling.
- Ai values range from 0.378 g to 0.457 g. This is in line with the variability results, which showed Ai values ranging from 0.1460 g to 0.5210 g. The composite results indicate that the Colnic ore is medium abrasive.
  It was proposed to conduct fine grinding signature plot test work to determine the power requirements to fine mill the flotation concentrate from P_{80} passing 75 µm to P_{80} passing 13.5 µm. There were no flotation concentrate samples available for the test work, and it was suggested to use feed or ROM material for the fine milling test work. The test work results indicated power requirements of 76.1 kWh/h and 61.2 kWh/h for Rovina and Colnic, respectively. The values are high, and the reasons could be that the ROM contains gangue, which is hard to fine mill. It is recommended to produce flotation concentrates to repeat the signature plot test work.

13.3.2.3 Ore Characterisation for Thickening Test Work

Table 13.11 shows a summary of the thickening test work results.

Table 13.11: Summary – Thickening Test Work Results

Sample Tested	80% passing (µm)	Tested pH	Liquid SG used for Calculations	Solids SG used for Calculations
Met 42 Cleaner Tails	12.77	8.6	1.0	3.98
Met 43 Cleaner Tails	13.78	8.6	1.0	3.79
Met 44 Cleaner Tails	12.60	9.1	1.0	3.51
Average Cleaner Tails	13.05	8.77	1.00	3.76
Met 42 Rougher Scavenger Tails	62.93	7.6	1.0	2.72
Met 43 Rougher Scavenger Tails	53.49	7.5	1.0	2.75
Met 44 Rougher Scavenger Tails	73.24	7.9	1.0	2.79
Average Rougher Scavenger Tails	63.22	7.67	1.00	2.75

Sample Tested	80% passing (µm)	Tested pH	Liquid SG used for Calculations	Solids SG used for Calculations
Met 42 Combined Tails	67.06	7.8	1.0	2.85
Met 43 Combined Tails	53.23	7.4	1.0	2.79
Met 44 Combined Tails	72.05	8.1	1.0	2.80
Average Combined Tails	64.11	7.77	1.00	2.82

The results indicate the following:

- The cleaner tails, rougher scavenger tails and combined tails have an average PSD of 80 % passing 13.05 µm, 63.22 µm and 64.11 µm, respectively.
- The cleaner tails, rougher scavenger tails and combined tails have an average solids SG of 3.76, 2.75 and 2.82, respectively.

13.3.2.4 Dynamic Thickening Test Work (Conventional Thickening)

Table 13.12 shows a summary of the dynamic thickening test work results for conventional thickening.

Table 13.12: Summary – Dynamic Thickening Test Work Results (Conventional Thickening)

Material Tested	Recommended Conventional Thickener Operating Parameter Ranges				
	Flocculant Dose, Type, and Concentration	Rise Rate and Unit Area at Specified Feed Solids Concentration and Underflow Density			
		Feed Solids Concentration (%)	Rise Rate (m^3/m^2h)	Unit Area ($m^2/t{\cdot}h$)	Underflow Density (%)
Met 42 Cleaner Tails	40 g/t of SNF AN 905 SH added at 0.1 g/L	20 25 30	6.08 3.23 1.40	4.224 4.536 5.928	63.5 63.5 63.5
Met 43 Cleaner Tails	45 g/t of SNF AN 905 SH added at 0.1 g/L	20 25 30	5.37 2.12 0.94	4.896 5.088 5.544	62.0 62.0 62.0
Met 44 Cleaner Tails	35 g/t of SNF AN 905 SH added at 0.1 g/L	20 25 30	3.94 1.79 0.56	4.800 6.120 12.576	57.0 57.0 57.0
Met 42 Rougher Scavenger Tails	20 g/t of SNF AN 905 SH added at 0.1 g/L	15 20 25	6.26 3.74 1.51	6.504 6.624 8.208	64.0 64.0 64.0
Met 43 Rougher Scavenger Tails	25 g/t of SNF AN 905 SH added at 0.1 g/L	15 20 25	4.70 1.29 0.67	7.392 8.016 12.312	57.0 57.0 57.0
Met 44 Rougher Scavenger Tails	15 g/t of SNF AN 905 SH added at 0.1 g/L	15 20 25	7.84 3.84 1.52	5.784 6.120 7.992	64.0 64.0 64.0
Met 42 Combined Tails	25 g/t of SNF AN 905 SH added at 0.1 g/L	15 20 25	7.12 2.21 1.02	5.904 6.792 8.976	63.0 63.0 63.0

Material Tested	Recommended Conventional Thickener Operating Parameter Ranges				
	Flocculant Dose, Type, and Concentration·	Rise Rate and Unit Area at Specified Feed Solids Concentration and Underflow Density			
		Feed Solids Concentration (%)	Rise Rate (m^3/m^2h)	Unit Area ($m^2/t{\cdot}h$)	Underflow Density (%)
Met 43 Combined Tails	35 g/t of SNF AN 905 SH added at 0.1 g/L	15 20 25	6.30 2.36 1.15	7.368 8.448 9.864	58.0 58.0 58.0
Met 44 Combined Tails	25 g/t of SNF AN 905 SH added at 0.1 g/L	15 20 25	7.36 3.10 1.32	5.856 5.952 9.336	64.0 64.0 64.0

The results indicate the following for conventional thickening:

- The cleaner and rougher tails can achieve underflow densities ranging from 57.0 % to 64 %.
- The combined tails can achieve underflow densities ranging from 58 % to 64 %.

13.3.2.5 Dynamic Thickening (High Rate and High Density)

Table 13.13 shows a summary of the dynamic thickening results for high-rate and high-density thickening.

Table 13.13: Summary – Dynamic thickening Test Work Results (High-Rate and High-Density Thickening)

Material Tested		Recommended High-Rate, High-Density, and Paste Thickener Operating Parameter Ranges							
		Tested Feed Solids (%)	Flocculant			Predicted Overflow TSS Concentrate Range (mg/L)	Design Basis		Max. Underflow Density (%)
			Type	Dose (g/t)	Concentration (g/L)		Thickener Type	Net Feed Loading Rate (m^3/m^2h)	
Rougher Scavenger Tails	Met 42	20.5	SNF AN 905 SH	25–30	0.1–0.25	150–250	High-Rate	3.02	62.0
							High-Density	2.37	65.0
							Paste	3.42	70.0
	Met 43	17.3	SNF AN 905 SH	30–35	0.1–0.25	150–250	High-Rate	2.57	57.0
							High-Density	1.91	59.0
							Paste	3.26	63.0
	Met 44	19.9	SNF AN 905 SH	20–25	0.1–0.25	150–250	High-Rate	3.04	63.0
							High-Density	2.32	65.0
							Paste	3.64	70.0
Cleaner Tails	Met 42	19.9	SNF AN 905 SH	45–50	0.1–0.25	150–250	High-Rate	3.87	63.5
							High-Density	2.97	66.0
							Paste	4.51	70.0
	Met 43	20.2	SNF AN 905 SH	50–55	0.1–0.25	150–250	High-Rate	3.82	62.0
							High-Density	3.02	65.0
							Paste	4.35	68.5
	Met 44	19.7	SNF AN 905 SH	40–45	0.1–0.25	150–250	High-Rate	2.79	57.0
							High-Density	2.22	60.5
							Paste	3.31	66.0

Material Tested		Recommended High-Rate, High-Density, and Paste Thickener Operating Parameter Ranges							
		Tested Feed Solids (%)	Flocculant			Predicted Overflow TSS Concentrate Range (mg/L)	Design Basis		Max. Underflow Density (%)
			Type	Dose (g/t)	Concentration (g/L)		Thickener Type	Net Feed Loading Rate (m^3/m^2h)	
Combined Tails	Met 42	19.2	SNF AN 905 SH	25–30	0.1–0.25	150–250	High-Rate	2.53	63.0
							High-Density	3.04	65.0
							Paste	3.62	70.0
	Met 43	20.3	SNF AN 905 SH	30–35	0.1–0.25	150–250	High-Rate	2.02	58.0
							High-Density	3.00	60.5
							Paste	3.45	65.0
	Met 44	20.4	SNF AN 905 SH	20–25	0.1–0.25	150–250	High-Rate	2.65	64.0
							High-Density	2.80	66.0
							Paste	3.67	70.0
TSS = Total Suspended Solids									

The results indicate that for

- High-rate thickeners,
  - The cleaner and rougher tails can achieve underflow densities ranging from 57.0 % to 63.5 %.
  - The combined tails can achieve underflow densities ranging from 58 % to 64 %.
- High-density thickeners
  - The cleaner and rougher tails can achieve underflow densities ranging from 59 % to 66 %.
  - The combined tails can achieve underflow densities ranging from 60.5 % to 66 %.

13.3.2.6 Viscosity

Table 13.14 shows a summary of the viscosity results.

Table 13.14: Summary – Viscosity Test Work Results

Material	Measurement Method	Solids Concentration (%)	Yield Value (Pascals or N/m^2)	Apparent Viscosity, (Pa•sec) at the following Shear Rates:								
				5 Sec^{-1}	10 Sec^{-1}	25 Sec^{-1}	50 Sec^{-1}	100 Sec^{-1}	150 Sec^{-1}	300 Sec^{-1}	600 Sec^{-1}	1,000 Sec^{-1}
MET 42 Cleaner Tails Thickened Underflow	Haake VT550 FL100 Vane	78.0	33,026.2	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		76.4	965.0	–	–	–	–	–	–	–	–	–
		74.0	227.3	–	–	–	–	–	–	–	–	–
		72.5	150.9	–	–	–	–	–	–	–	–	–
		68.6	84.9	–	–	–	–	–	–	–	–	–
		66.4	65.4	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	64.8	44.0	–	–	–	–	–	–	–	–	–
		64.2	37.4	0.065	4.31	2.70	1.46	0.91	0.57	0.43	0.27	0.17
		61.8	18.2	0.036	2.31	1.43	0.76	0.47	0.29	0.22	0.13	0.08
		59.7	12.2	0.021	1.57	0.95	0.49	0.30	0.18	0.13	0.08	0.05
MET 43 Cleaner Tails Thickened Underflow	Haake VT550 FL100 Vane	56.6	7.0	0.011	0.98	0.57	0.28	0.16	0.10	0.07	0.04	0.02
		74.7	2,416.3	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		74.0	1,836.4	–	–	–	–	–	–	–	–	–
		72.9	808.8	–	–	–	–	–	–	–	–	–
		71.4	198.0	–	–	–	–	–	–	–	–	–
		65.8	54.1	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	62.7	32.6	0.045	3.58	2.23	1.19	0.74	0.46	0.35	0.22	0.14
		60.1	18.2	0.030	2.22	1.36	0.71	0.44	0.27	0.20	0.12	0.08
		57.2	10.9	0.014	1.42	0.83	0.41	0.24	0.14	0.10	0.06	0.04
		50.5	3.7	0.006	0.46	0.28	0.15	0.09	0.06	0.04	0.03	0.02

Material	Measurement Method	Solids Concentration (%)	Yield Value (Pascals or N/m²)	Apparent Viscosity, (Pa•sec) at the following Shear Rates:								
				5 Sec⁻¹	10 Sec⁻¹	25 Sec⁻¹	50 Sec⁻¹	100 Sec⁻¹	150 Sec⁻¹	300 Sec⁻¹	600 Sec⁻¹	1,000 Sec⁻¹
MET 44 Cleaner Tails Thickened Underflow	Haake VT550 FL100 Vane	76.9	28,261.0	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		75.1	1,851.0	–	–	–	–	–	–	–	–	–
		74.5	978.0	–	–	–	–	–	–	–	–	–
		72.6	321.2	–	–	–	–	–	–	–	–	–
		68.8	132.0	–	–	–	–	–	–	–	–	–
		66.6	94.8	–	–	–	–	–	–	–	–	–
		61.7	57.5	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	60.8	52.1	0.075	5.71	3.58	1.93	1.21	0.76	0.58	0.36	0.23
		57.1	25.3	0.043	3.12	1.90	0.99	0.61	0.37	0.28	0.17	0.10
		52.5	13.2	0.021	1.49	0.94	0.51	0.32	0.20	0.15	0.10	0.06
		49.6	8.3	0.009	1.11	0.64	0.31	0.18	0.10	0.08	0.04	0.03
MET 42 Rougher Scavenger Tails Thickened Underflow	Haake VT550 FL100 Vane	75.8	1,861.8	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		75.5	1,227.8	–	–	–	–	–	–	–	–	–
		74.9	463.6	–	–	–	–	–	–	–	–	–
		72.8	224.5	–	–	–	–	–	–	–	–	–
		70.7	108.3	–	–	–	–	–	–	–	–	–
		67.8	70.5	–	–	–	–	–	–	–	–	–
		65.5	53.0	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	65.4	54.9	0.095	6.47	4.03	2.16	1.35	0.84	0.64	0.40	0.25
		63.1	30.8	0.054	3.80	2.34	1.23	0.76	0.46	0.35	0.22	0.13
		59.8	14.6	0.028	2.05	1.22	0.61	0.36	0.22	0.16	0.09	0.06
		56.7	9.5	0.015	1.28	0.76	0.38	0.22	0.13	0.10	0.06	0.03
MET 43 Rougher Scavenger Tails Thickened	Haake VT550 FL100 Vane	72.3	2,454.9	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		71.4	1,753.5	–	–	–	–	–	–	–	–	–
		70.6	742.2	–	–	–	–	–	–	–	–	–
		69.2	394.2	–	–	–	–	–	–	–	–	–

Material	Measurement Method	Solids Concentration (%)	Yield Value (Pascals or N/m²)	Apparent Viscosity, (Pa•sec) at the following Shear Rates:								
				5 Sec⁻¹	10 Sec⁻¹	25 Sec⁻¹	50 Sec⁻¹	100 Sec⁻¹	150 Sec⁻¹	300 Sec⁻¹	600 Sec⁻¹	1,000 Sec⁻¹
Underflow		66.5	206.4	–	–	–	–	–	–	–	–	–
		64.0	130.7	–	–	–	–	–	–	–	–	–
		58.8	49.6	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	59.1	52.2	0.061	5.73	3.52	1.85	1.14	0.70	0.53	0.32	0.20
		57.4	30.9	0.046	3.75	2.28	1.18	0.72	0.43	0.32	0.20	0.12
		54.4	16.2	0.025	2.34	1.35	0.65	0.38	0.22	0.16	0.09	0.05
		49.5	6.5	0.014	1.10	0.63	0.30	0.17	0.10	0.07	0.04	0.02
MET 44 Rougher Scavenger Tails Thickened Underflow	Haake VT550 FL100 Vane	77.8	2,454.9	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		76.1	647.2	–	–	–	–	–	–	–	–	–
		73.4	174.1	–	–	–	–	–	–	–	–	–
		72.3	127.9	–	–	–	–	–	–	–	–	–
		70.0	90.0	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	66.7	47.0	0.133	6.13	3.93	2.18	1.40	0.89	0.69	0.44	0.28
		63.1	27.3	0.064	3.50	2.20	1.19	0.75	0.47	0.36	0.22	0.14
		59.4	12.3	0.030	1.80	1.09	0.56	0.34	0.21	0.15	0.09	0.06
		55.4	7.6	0.015	1.05	0.63	0.32	0.19	0.11	0.09	0.05	0.03
MET 42 Combined Tails Thickened Underflow	Haake VT550 FL100 Vane	76.0	2,192.5	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		75.8	1,694.3	–	–	–	–	–	–	–	–	–
		75.1	1,084.0	–	–	–	–	–	–	–	–	–
		73.2	487.3	–	–	–	–	–	–	–	–	–
		69.6	112.2	–	–	–	–	–	–	–	–	–
		67.3	67.1	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	64.2	42.8	0.046	4.52	2.79	1.48	0.92	0.57	0.43	0.27	0.16
		62.5	28.7	0.036	3.28	2.01	1.05	0.64	0.39	0.29	0.18	0.11
		60.1	18.8	0.028	2.41	1.46	0.75	0.45	0.27	0.20	0.12	0.07
		58.1	13.4	0.019	1.80	1.06	0.52	0.31	0.18	0.13	0.08	0.05

Material	Measurement Method	Solids Concentration (%)	Yield Value (Pascals or N/m²)	Apparent Viscosity, (Pa•sec) at the following Shear Rates:								
				5 Sec⁻¹	10 Sec⁻¹	25 Sec⁻¹	50 Sec⁻¹	100 Sec⁻¹	150 Sec⁻¹	300 Sec⁻¹	600 Sec⁻¹	1,000 Sec⁻¹
MET 43 Combined Tails Thickened Underflow	Haake VT550 FL100 Vane	72.1	4,096.4	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		71.1	1,498.5	–	–	–	–	–	–	–	–	–
		69.9	667.2	–	–	–	–	–	–	–	–	–
		68.8	304.6	–	–	–	–	–	–	–	–	–
		66.7	170.9	–	–	–	–	–	–	–	–	–
		65.0	108.0	–	–	–	–	–	–	–	–	–
		64.3	94.8	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	61.2	57.4	0.068	6.38	3.92	2.06	1.26	0.77	0.58	0.36	0.22
		59.3	38.0	0.041	4.26	2.59	1.34	0.81	0.49	0.37	0.22	0.14
		56.2	21.1	0.026	2.64	1.57	0.79	0.47	0.28	0.21	0.12	0.07
		53.9	12.7	0.017	1.60	0.95	0.48	0.29	0.17	0.13	0.08	0.04
MET 44 Combined Tails Thickened Underflow	Haake VT550 FL100 Vane	78.0	3,683.4	*Formed Filter Cake*	–	–	–	–	–	–	–	–
		75.8	909.7	–	–	–	–	–	–	–	–	–
		73.4	318.7	–	–	–	–	–	–	–	–	–
		72.5	203.6	–	–	–	–	–	–	–	–	–
		71.2	125.2	–	–	–	–	–	–	–	–	–
		69.7	95.9	–	–	–	–	–	–	–	–	–
		67.4	62.0	–	–	–	–	–	–	–	–	–
	Fann Model 35A Bob and Rotor	66.4	49.7	0.111	6.29	3.94	2.12	1.33	0.83	0.63	0.40	0.25
		63.8	27.0	0.050	3.42	2.11	1.11	0.68	0.42	0.32	0.19	0.12
		60.2	12.6	0.021	1.79	1.05	0.51	0.30	0.18	0.13	0.07	0.04
		57.0	7.2	0.013	0.95	0.57	0.29	0.18	0.11	0.08	0.05	0.03

The decreasing apparent viscosity, with increasing shear rate or "shear thinning" behaviour of the pulps examined, is characteristic of the pseudoplastic class of non-Newtonian fluid. It demonstrates the need to achieve and maintain a specific velocity gradient or shear rate in order to initiate and maintain flow.

13.3.2.7 Pressure and Vacuum filtration

Table 13.15 shows a summary of the pressure and vacuum filtration test work results.

Table 13.15: Summary – Pressure and Vacuum Filtration Results

Test Work			Feed Solids Concentration (% w/w)	Dry Cake Moisture (% w/w)		Cycle Times (min)	Volumetric Production Rate ($kg/m^3.h$)		Area Production ($kg/m^2.h$)	
			Range	Average	Range		Range	Average	Range	Average
Pressure Filtration	Cleaner Tails	Blow Only	57.2 to 60.2	15.70	14.5 to 18.1	12.5	5,794.2 to 8,183.3	7,199.4	162.8 to 230.0	202.3
		Squeeze and Blow	57.2 to 60.2	14.90	9.9 to 15.8	12.5	6,470.6 to 9,569.8	7,896.2	157.2 to 252.3	205.0
	Rougher Scavenger Tails	Blow Only	49.7 to 50.8	16.60	14.5 to 19.0	12.4	5,208 to 6,063.5	5,659.1	146.3 to 175.9	161.80
		Squeeze and Blow	49.7 to 50.8	15.70	13.7 to 19.0	11.8	1,910.1 to 7,231.1	6,599.7	156.6 to 183.5	168.90
Vacuum Filtration	Cleaner Tails	Top Load HVB	56.3 to 63.6	22.10	20.8 to 24.6	4.4	-	-	181.6 to 500.9	331.2
		Ceramic Disc Filtration	56.3 to 59.2	21.03	19.6 to 23.5	-	-	-	71.6 to 184.2	134.5
	Rougher Scavenger Tails	Top Load HVB	50.1 to 61.1	22.45	21.1 to 24.3	4.7	-	-	154.2 to 435.6	267.0
		Ceramic Disc Filtration	50.1 to 50.7	21.40	20.0 to 22.7	-	-	-	91.0 to 195.1	131.9

Vacuum filtration generally gave a higher cake moisture compared to pressure filtration. Pressure filtration was, therefore, selected as the preferred filtration technology.

13.3.2.8 Transport Moisture Limit

Table 13.16 shows a summary of the Transport Moisture Limit results.

Table 13.16: Summary – Transport Moisture Limit Results.

Sample	Flow Moisture Point (% Moisture)	Transportable Moisture Limit (% Moisture)	Tamping Pressure Used (kPa)
Met 42 Cleaner Tails	14.5	13.1	40.9
Met 43 Cleaner Tails	15.5	10.4	41.4
Met 44 Cleaner Tails	18.9	17.0	38.2
Met 42 Rougher Scavenger Tails	16.9	15.2	31.9
Met 43 Rougher Scavenger Tails	18.3	16.5	30.3
Met 44 Rougher Scavenger Tails	16.8	15.1	32.0
Met 42 Combined Tails	16.4	14.8	32.4
Met 43 Combined Tails	17.5	15.0	32.6
Met 44 Combined Tails	16.4	14.8	30.1

The results indicate transport moisture limit values ranging from 10.4 % to 17.0 %. These values are within the range of the moisture content achieved via pressure filtration.

13.3.2.9 Acid Base Accounting (ABA) and Net Acid Generation on Float Tails

Tests were performed to determine the acid generation potential of the flotation tailings:

- The rougher-scavenger tailings are non-acid generating, and their disposal will not produce acidity or significant ML.
- The cleaner tailings, however, are strongly acid generating, will apparently oxidise readily, and will represent a high risk for the generation of acidic drainage containing significant metal loadings.
- If the rougher-scavenger and cleaner tailings are not kept separate in the processing plant, then the combined tailings will be acid generating and will represent a risk to the quality of water draining from the waste facility, not only on their own but also by increasing the risk of an earlier onset of ARD from the co-deposited waste rock.

A mineralogical analysis X-ray diffraction (XRD) was performed on the float tails samples. Table 13.17 shows a summary of the mineralogy results.

Table 13.17: Summary of the Mineralogy Results

Mineral	Met 42 Rougher Scavenger Tails	Met 43 Rougher Scavenger Tails	Met 44 Rougher Scavenger Tails	Met 42 Cleaner Tails	Met 43 Cleaner Tails	Met 44 Cleaner Tails
Actinolite	1.8	0.2	2.1	0.6	0.4	0.8
Plagioclase (albite-calcian)	32.6	26.7	36.5	9.2	11.3	16.8
Biotite	2.8	3.4	3.4	–	–	1.9
Calcite	2.5	3.3	2.4	–	–	–
Chalcopyrite	–	–	–	3.6	2.8	4.0
Clinochlore	7.0	6.7	5.1	2.5	3.4	4.9
Gypsum	–	–	–	1.8	1.9	3.3
Illite/Muscovite 2M1	4.5	7.5	4.1	–	–	–
Iron-alpha	–	–	–	0.3	0.7	0.4
Maghemite	0.7	0.6		–	–	–
Magnetite	1.4	1.9	1.3	0.4	0.5	3.6
Molybdenite	–	–	–	0.2	–	–
Orthoclase	14.0	13.9	13.0	2.8	4.6	6.3
Pyrite	–	0.4	–	67.0	57.5	37.3
Pyrrhotite	–	–	–	1.6	1.8	7.2
Quartz	32.9	35.5	32.1	9.9	15.3	13.5
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**

With respect to ARD, the minerals of interest are the sulphides providing the potential for acid generation and those minerals capable of neutralising acidity generated by sulphide oxidation. The sulphide content of the rougher-scavenger tailings would be expected to be very low as evidenced in Table 13.17. The cleaner tails, however, would be expected to be high in sulphides and the data in Table 13.17 shows that pyrite is the dominant sulphide present, with lesser but significant quantities of chalcopyrite and pyrrhotite.

14 MINERAL RESOURCE ESTIMATES

The information for this section was sourced from AGP's PEA NI 43-101 2019 Report and edited where necessary.

14.1 BACKGROUND

In 2007, AMEC completed the maiden mineral resource estimate for the Colnic and Rovina deposits. This was followed by further resource definition drilling and in 2009, PEG updated the mineral resource estimate for the Colnic and Rovina deposits and completed a maiden mineral resource estimate for the Ciresata deposit. In preparation for a prefeasibility study in 2011, ESM embarked on an aggressive infill drilling programme, and in 2012 commissioned AGP to complete an updated mineral resource estimate on the three deposits. The 2012 mineral resource estimate included drillhole information as at 31 May 2012. After 31 May 2012, the ESM-Barrick exploration collaboration drilled an additional six holes in the Rovina and Colnic deposits; a summary of this drilling is given in Table 12.3. From 2012 until 2018, there was a hiatus in exploration activities as required by law, while ESM submitted the application for a conversion of their Rovina exploration licence to an exploitation licence, which was granted in November 2018.

In February 2019, AGP completed a PEA study on the Rovina, Colnic and Ciresata deposits. During this study, AGP assessed the possible impact of additional drilling completed by the ESM-Barrick exploration collaboration (after the 31 May 2012 data cut-off date) on the mineral resource estimates and concluded that "*Holes completed after the data cut-off date are unlikely to have a significant impact on the resources and AGP, therefore, recommended that ESM not update the 2012 grade or classification model until more holes are added to the resource*" (Section 12.1.7). The February 2019 mineral resource estimate was, therefore, updated to reflect current metal prices and operation parameters at that time, with no changes to the geological and mineral resource block models.

14.2 INTRODUCTION

In March 2020, ESM commissioned SENET to complete a DFS on the open-pit Rovina and Colnic deposits. As part of this study, CCIC MinRes completed a detailed technical audit of the resource models, including an assessment on the possible impact of the ESM-Barrick exploration collaboration drilling on the mineral resource estimates. A summary of the findings is in Table 14.1. The three holes drilled at Rovina are as follows: RRM-1 was drilled for additional metallurgical test work, RRD-85 was a Twin of RRD-55, and RRD-84 was a new intersection inside of the mineral resource constraining shell; however, there is no risk of overestimating the mineral resources. A cross-section illustrating the grade comparison between the drillhole assays and model estimates for Au in Figure 14.1 shows that the model estimates are very similar or less than the sample grades. The three holes drilled at Colnic are as follows: RCM-1 was drilled for additional metallurgical test work, RCD-105 was drilled outside of the resource area, and RCD-106 intersected mineralisation at approximately 150 m below the mineral resource constraining shell, shown in Figure 14.2. CCIC MinRes also recommended that ESM not update the 2012 geological and mineral resource block models until more holes are added to the resource database.

The March 2021 mineral resource estimate for the Rovina and Colnic deposits is, therefore, updated to reflect current metal prices and updated operating parameters derived during the

DFS. Mr Sivanesan Subramani, BSc Hons (Geology), Pri.Sci.Nat (400184/06), is the QP for this mineral resource estimate. The mineral resources are constrained to a Lerchs-Grossmann pit shell using appropriate cut-off grades. The geological model and mineral resource block models for Colnic and Rovina remain unchanged in this current estimate. The mineral resource estimate for Ciresata remains unchanged from February 2019.

Table 14.1: Summary of Drillholes Excluded from the Resource Model

Deposit	Hole-ID	Finding
Rovina	RRD-84	Hole inside resource constraining shell, no risk of overestimation
Rovina	RRD-85	Twin of RRD-55
Rovina	RRM-1	Metallurgical test work hole
Colnic	RCD-105	Drilled outside the resource model
Colnic	RCD-106	Drilled outside of the resource constraining shell
Colnic	RCM-1	Metallurgical test work hole

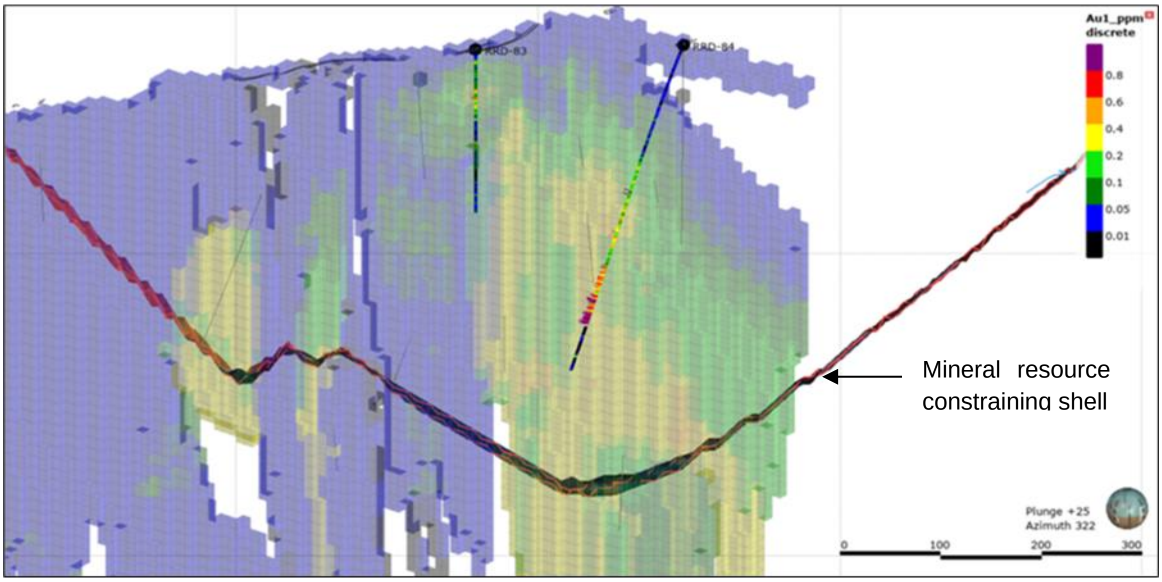


Figure 14.1: Cross-Section of Drillhole RRD-84 at the Rovina Deposit against Model Estimates, for Au

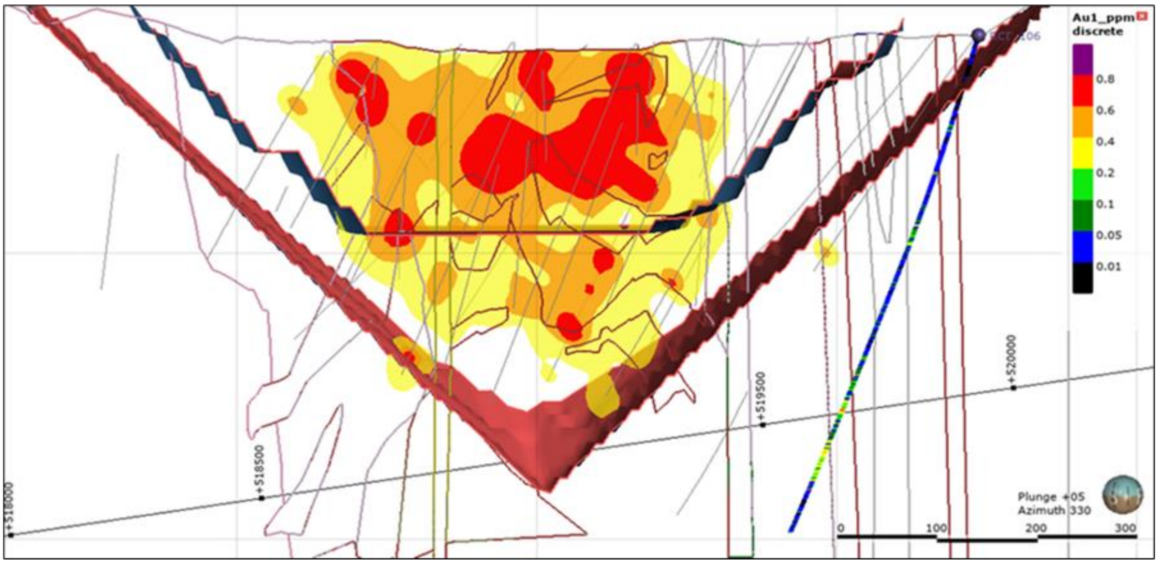


Figure 14.2: Cross-Section of Drillhole RCD-106 at the Colnic Deposit outside of Resource Shell

Gemcom software GEMS 6.3.1™ was used for the resource estimate, in conjunction with Sage 2001 for the variography. The metals of interest at RVP are Au and Cu. Minor amounts of Pb, Zn and Mo are also present. Zn, while not reported as a mineral resource, was estimated in the model due to a smelter penalty associated with a high Zn tenor in the concentrate.

ESM provided the digital data files as a series of XLS spreadsheets, in stages as drilling progressed. The Ciresata data was provided on 12 December 2011, the Colnic data was provided on 15 February 2012, and the Rovina data was provided on 31 May 2012. The digital data files consisted of drillhole collar, survey, lithology, alteration, and assay tables, along with 3D triangulations representing the latest interpretation for the lithological and alteration units. A topographical surface and the bottom of the overburden surface were also provided.

The effective data cut-off date for the resource estimation was 31 May 2012. At that time, the provided database consisted of 82 drillholes on the Rovina deposit, 106 holes on the Colnic porphyry, and 63 holes on the Ciresata porphyry. On the Rovina porphyry, three holes (RRD-12, -19 and -47) were abandoned due to ground conditions and re-drilled with a new number. An additional 16 drillholes were added after the data cut-off date in 2012. Three holes at Rovina, three holes at Colnic, and ten holes at Ciresata.

Historical holes drilled by Minexfor were not used in the estimate; mineral resource estimation is only based on drilling done by ESM, from 2006 to 2012. Table 14.2 summarises the number of holes from each deposit used in the resource estimate. A list of all the holes in the database can be found in Appendix A of the AGP PEA NI 43-101 2019 Report.

Table 14.2: Total Number of Drillholes Used

Deposit	No. of Holes	No. of Unused Holes in Resource	Total Core Length (m)	No. of Assays	Core Length per Assay (m)	Last Holes in Database
Rovina	82	3 (re-drilled)	39,133	37,888	1.03	RRD-83
Colnic	106	0	33,994	32,789	1.04	RCD-104
Ciresata	63	0	47,129	43,079	1.09	RGD-63
Total	**251**	**3**	**120,256**	**113,756**	**1.06**	
Holes Added After the Data Cut-off Date						
Rovina	3	3	1,133	1,034	1.09	RRD-85

14.3 GEOLOGICAL INTERPRETATION

The three deposits in the RVP were divided into several lithological units, superimposed by various alteration phases. A series of 3D wireframe models were provided by ESM, each consisting of a detailed lithology model and a separate alteration model, based on the dominant alteration type in the drill core. The alteration is typically complex, with more than one overprinting alteration type present. Where several alteration phases coexist, the dominant alteration type was assigned a logging code based on mineralogy and intensity. These were then grouped into units from which the alteration model was created.

In 2008, ESM geologists first modelled the deposit on a series of cross-sections, followed by a series of longitudinal sections and plans. The lithological and alteration boundaries were adjusted to coincide in all three orientations. The paper sections and plans were then digitised and tied together and wireframed in 3D on the computer. In 2012, these models were updated directly in the Micromine™ software. The complete list of the lithology and alteration wireframe components, with short names is shown in Table 14.3.

Table 14.3: Lithology and Alteration Wireframe Names

RVP Lithology	Name	RVP Alteration	Name
Glamm Breccia	GLAM	Oxidation (Fe-Ox)	OXIDE
Porphyry E (Potassic-Silicic Porphyry)	POE	Propylitic Alteration within Glamm Breccia	GLAM-POT
Porphyry D (Baroc Valley Porphyry)	POD	Phyllic Alteration (Ser-Py-Qtz)	PHYLLIC
Porphyry C (Rovina Porphyry)	POC	MACE Alteration (Mag-Qtz-Amph-Chl-Ep-Carb)	MACE
Porphyry B (Late Mineral Porphyry)	POB	Potassic Alteration (Bio-Kspar)	POT
Porphyry A (North West Baroc Porphyry)	POA	Silicification (Silica)	SILICIC
Intrusive Magmatic Breccia	IMB	Potassic-Silicic Alteration (Bio-Kspar-Sil+/-Po)	KSIL
Flysch Sediment	SED		
Post-Mineral Dyke (Late Intrusion)	LD		
Basement Sediment	SED	Oxidation (Fe-Ox)	OXIDE
Cornitel (Wallrock) Porphyry	WR_POR	K3 Alteration (Bio-Kspar, Qtz, Mag, diss.	K3

RVP Lithology	Name	RVP Alteration	Name
		sulph.)	
Intrusive Magmatic Breccia	IMB	K2 Alteration (Chl-Ep, Carb, Mag, Py, Cpx)	K2
Colnic Porphyry and Breccia Package	C_POR	K Alteration (K2, K3, phyllic, and argillic mix)	K
Deep Coherent Colnic Porphyry Stock	CD_POR	Transitional Phyllic Alteration	TRPH
F2-Hill Porphyry and Breccia Package	F2_POR	Phyllic Alteration (Ser-Py-Qtz)	PHYLLIC
Late Mineral Dyke	LM_POR	Potassic-Argillic Alteration - Satellite breccia	K3_ARG
Garnet Bearing Porphyry	G_POR	Argillic Alteration - in Chubby fault	ARG
Satellite Porphyry and Breccia Package	S_POR	Propylitic Alteration - in LD dyke	PROP_D
	S_BX		
Post Mineral Porphyry Dykes and Plug	LD		
Chubby's Fault	FLT		
Ciresata Lithology	Name	Ciresata Alteration	Name
Early mineral porphyry	EM_P	Potassic Alteration (Bio-Kspar)	POT
Basement Sediment	SED	Phyllic Alteration (Ser-Py-Qtz)	PHYLLIC
Inter Mineral Porphyry	IM-P		
West Porphyry	WP		
Host Rock Porphyry	HR-P		
Late-Mineral Porphyry Dykes	LM_P		

The corresponding lithology and alteration solids are shown in Figure 14.3 to Figure 14.8. The completed solids were validated for interpretational consistency and were found to honour the drillhole data and interpreted geology. The solids were used to code the drillhole data prior to final domain definition. Identical colour profiles were assigned to the solids and drillhole data, and the two datasets were visually inspected on sections and plans to ensure the proper assignment of domains to drillholes.

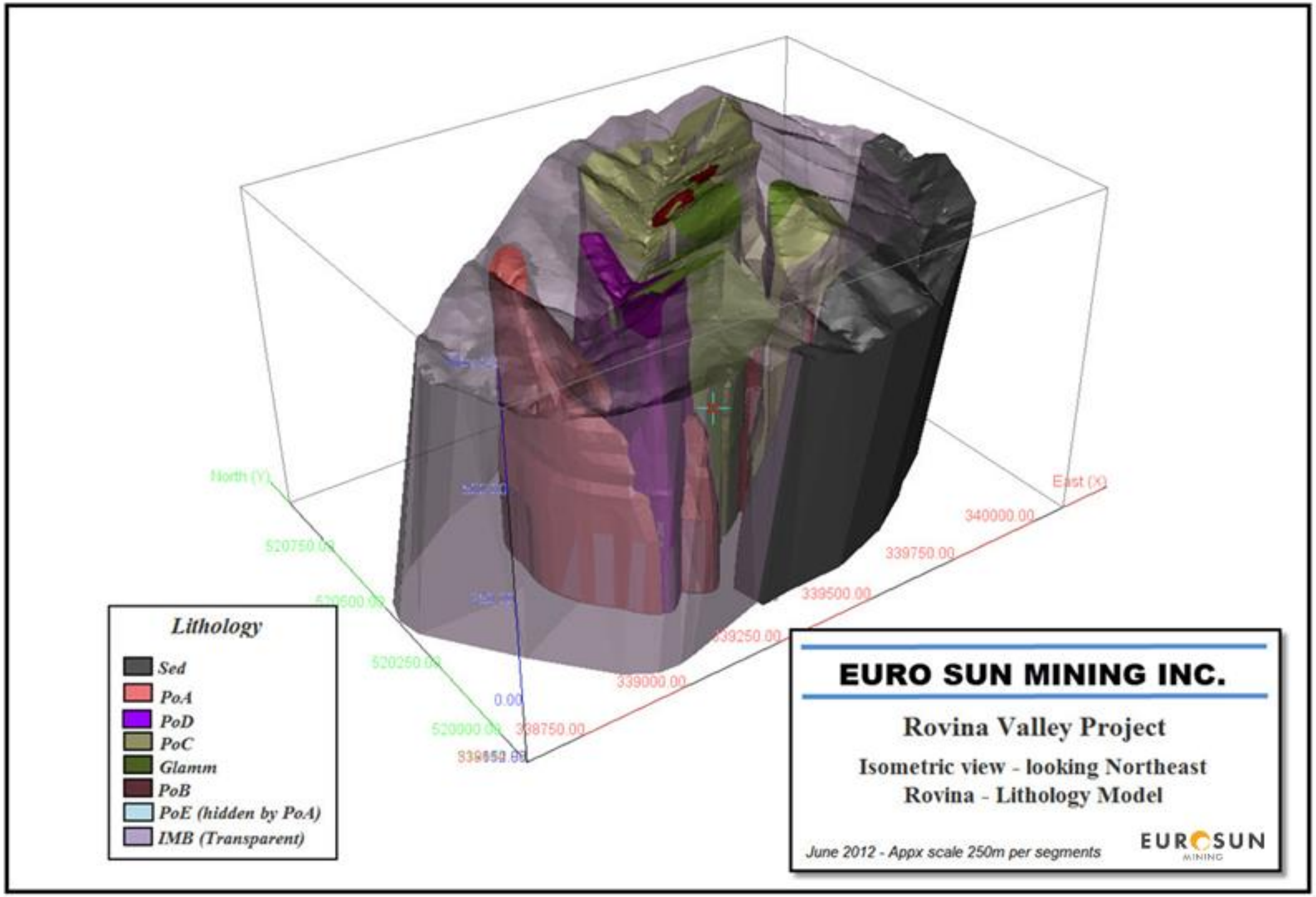


Figure 14.3: Rovina Deposit Lithology Domains

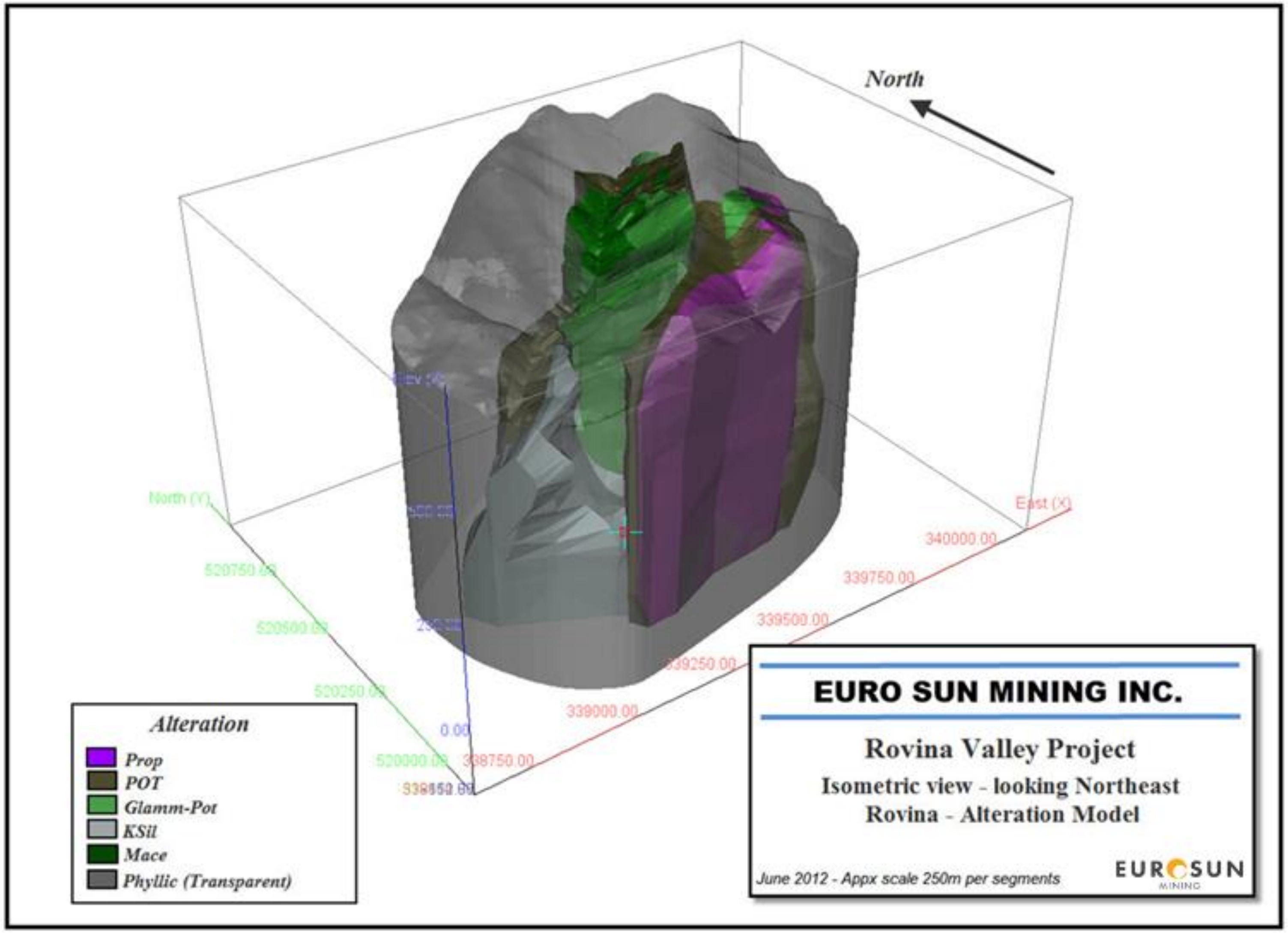


Figure 14.4: Rovina Deposit Alteration Domains

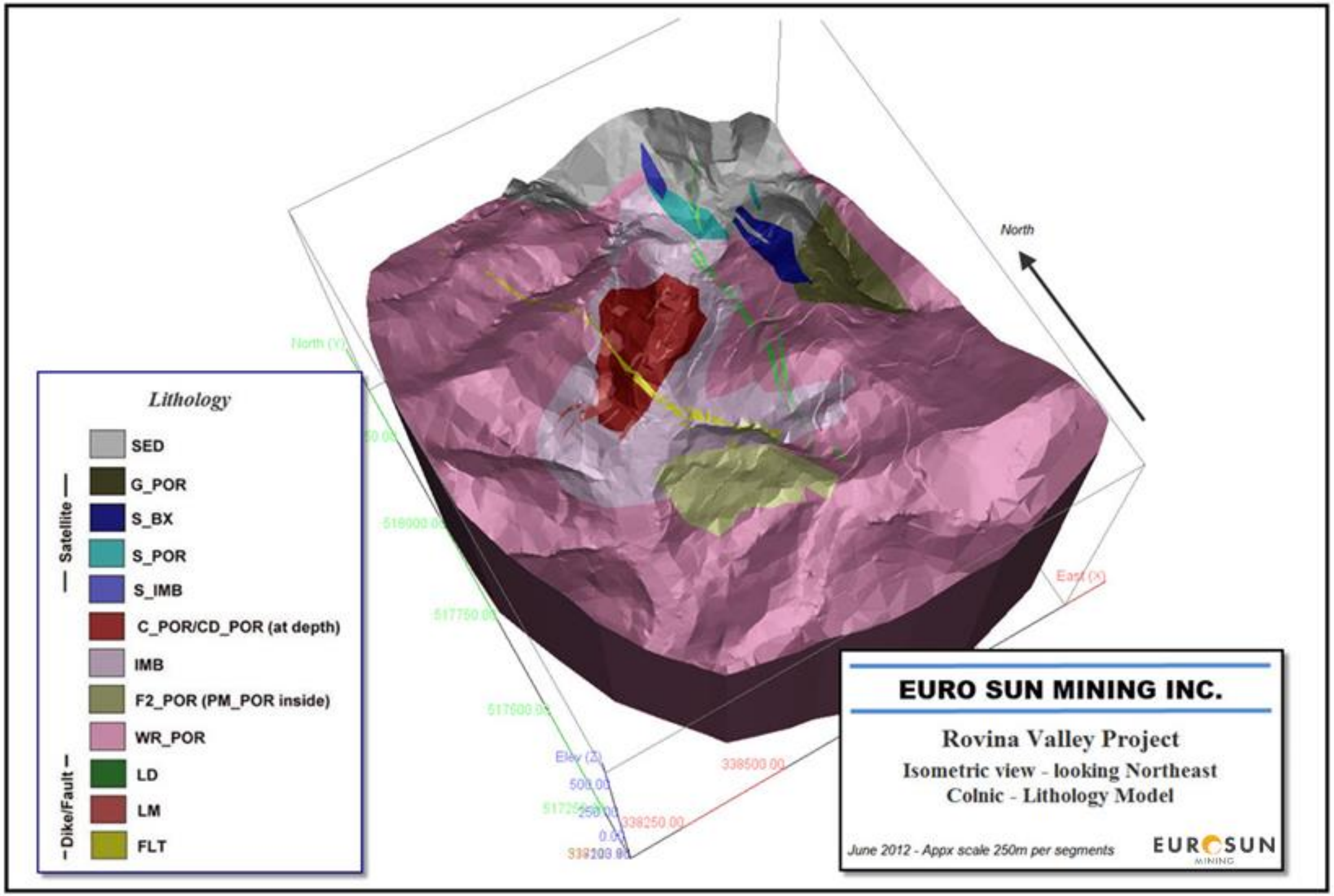


Figure 14.5: Colnic Deposit Lithology Domains

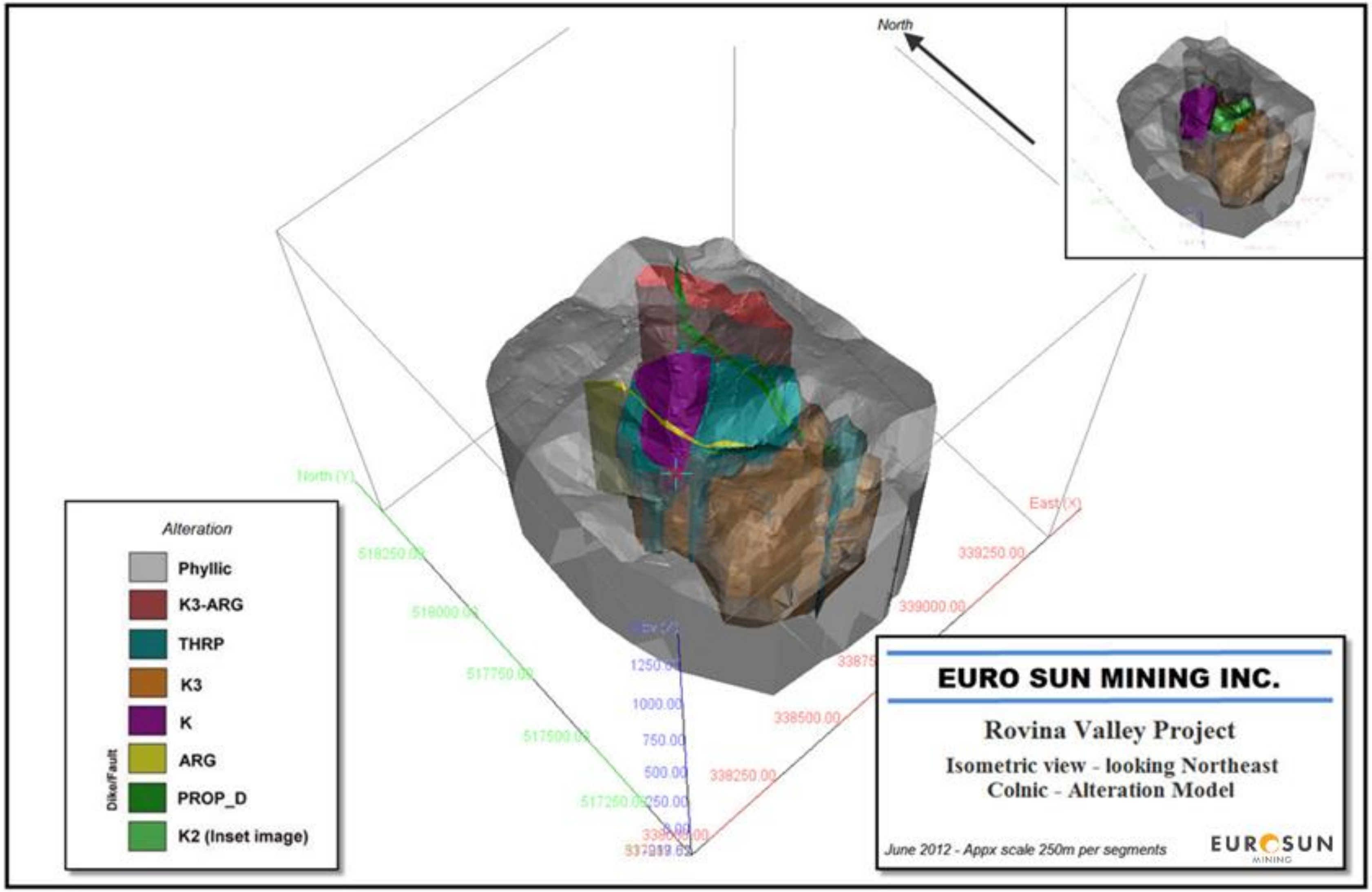


Figure 14.6: Colnic Deposit Alteration Domains

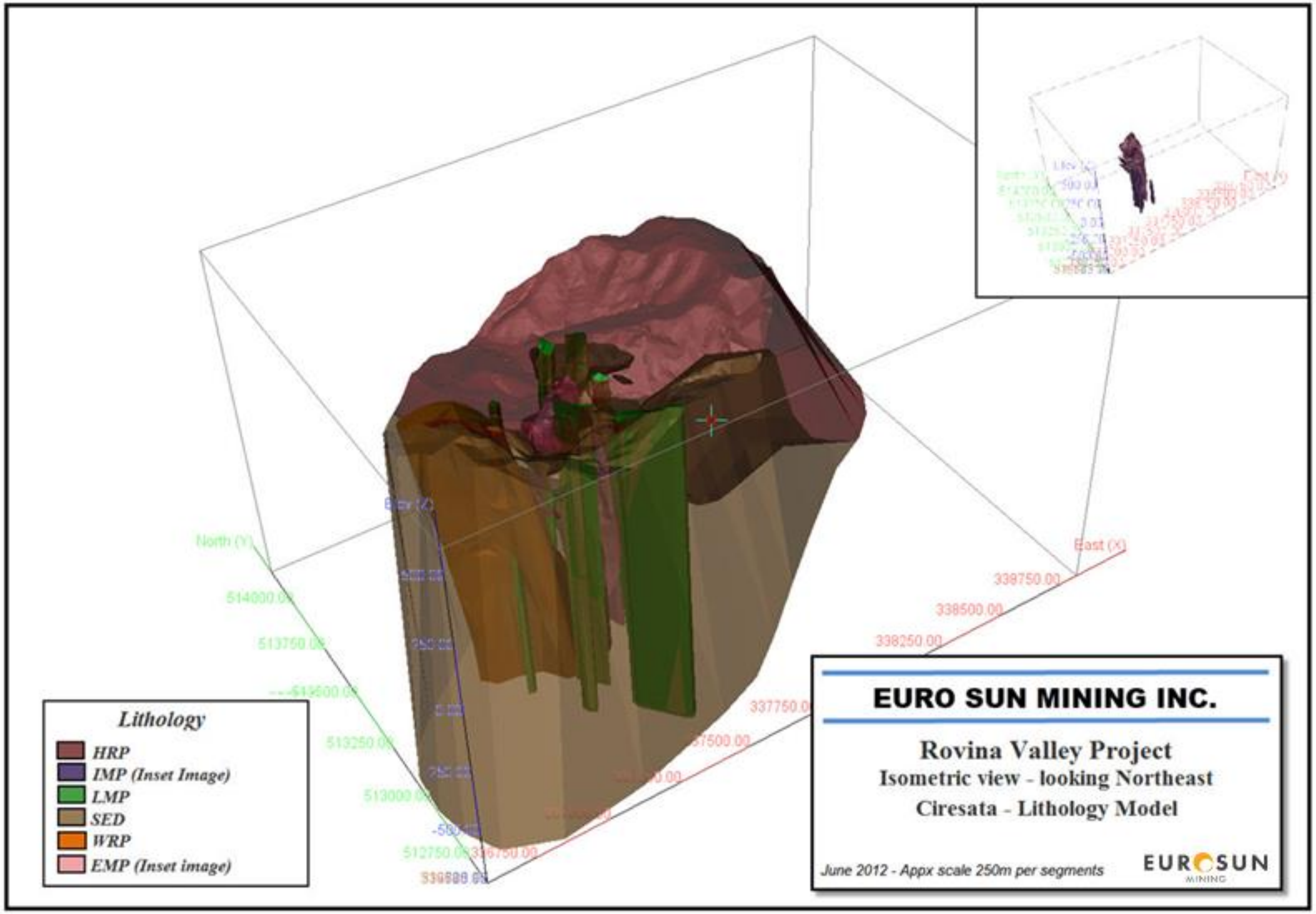


Figure 14.7: Ciresata Deposit Lithology Domains

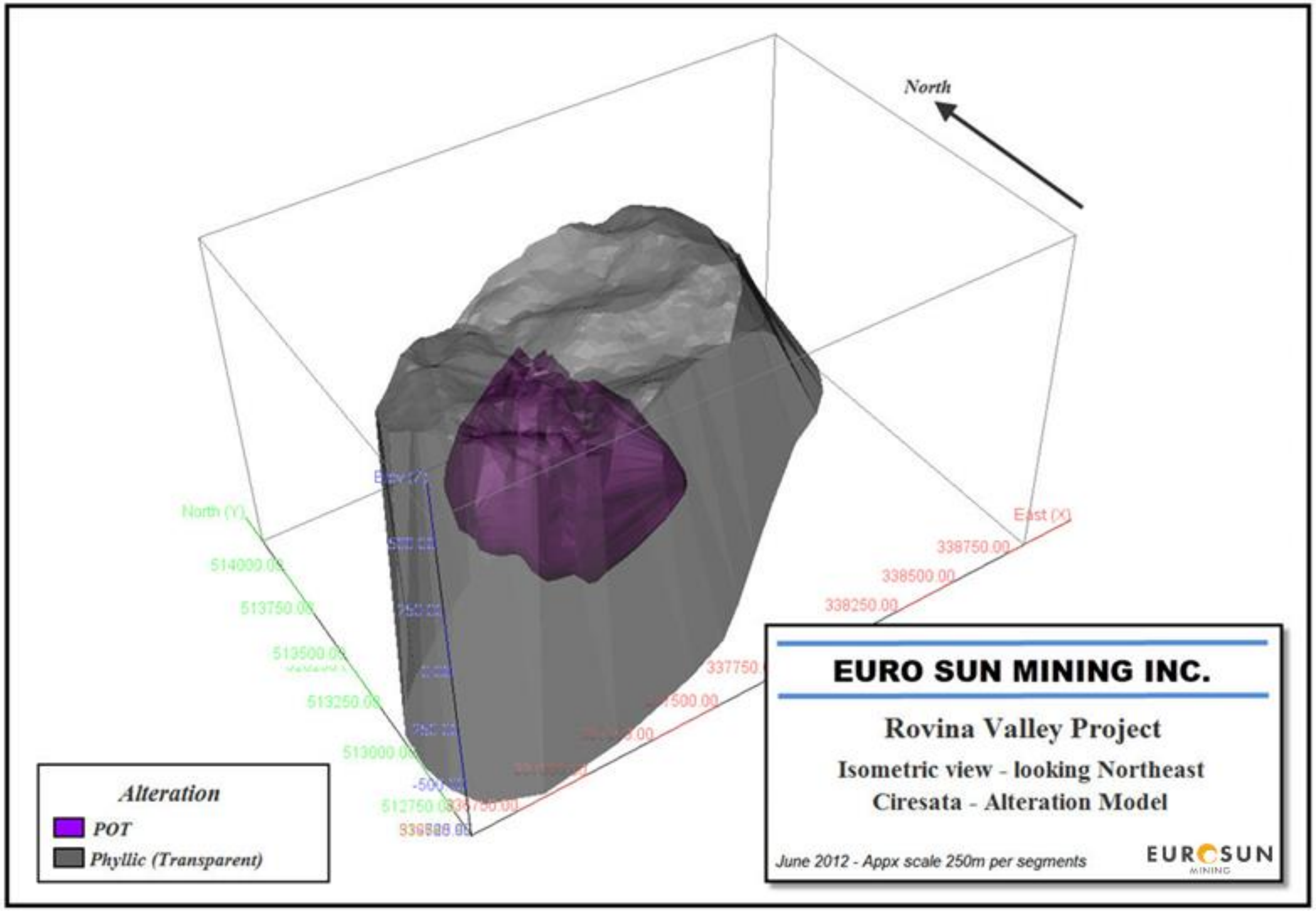


Figure 14.8: Ciresata Deposit Alteration Domains

14.4 EXPLORATORY DATA ANALYSIS

Exploratory data analysis is the application of various statistical tools to characterise the statistical behaviour or grade distributions of the data set. In this case, the objective was to understand the population distribution of the grade elements in the various units using tools such as histograms, descriptive statistics, probability plots, and boundary analysis.

Statistical analysis of the data was performed on each of the lithology and alteration units and the combination of lithology and alteration units to assist in the selection of the statistical domains for use in grade interpolation.

14.4.1 Domain Definition

For all three deposits, the delineation of the statistical domains was based primarily on field information gathered by the ESM project geologists, along with an examination of the drill core. It also relied on the following statistical tools for confirmation of the field observation:

- Mean value statistics by lithology, alteration, and a combination of the lithology and alteration units
- Box plot statistics by lithology alteration and a combination of the lithology and alteration units
- Contact profiles to assist in understanding the grade distribution at the boundaries between the different units
- Analysis of variance (ANOVA), which aided separation of the low-grade lithological/alteration combination units from the high-grade units

14.4.1.1 Rovina Deposit

The box plots for Au and Cu by lithology show POB, POC, and POD as separate entities, each with higher values for Cu. While the POE unit could fall in that group as well, it was not as clearly defined.

The box plots by alterations showed the MACE, POT, and SILICIC alterations as a separate domain for both Au and Cu. Note that the SILICIC alteration is only located below the MACEB alteration unit.

The box plot with the combined alteration and lithology codes showed some dominant features, but proper ranking of them was difficult. ANOVA was used to assist in the separation of the combination domains. While this statistical tool optimally functions with normal distributed data, it was used in this study as a guide for classification, cross-referencing with the box plot and mean data. During the ANOVA analysis, a table was produced showing the critical differences in the variances within the group pairs along with the calculation of their significance (P value). A second table was then produced that calculated subsets of the grouping variables, which were significantly different from each other.

From the information obtained during the site visit, the box plot, mean values, and ANOVA, the following conclusions were reached:

- The lithological units appear to have the most weight in determining the grade of the deposit; however, the MACE and POT alteration cannot be discounted as the best grade are in the MACE.
- The KSIL alteration is deep-seated, affecting only the POE lithological unit.
- It was not possible to group the MACE alteration into a single domain, encompassing all lithologies. Each lithological unit in MACE, therefore, formed its own separate domain. This was because of the relatively large grade variations between lithology and MACE combined units.
- GLAM is a brecciated unit with high-grade values in the clasts; this unit was treated as a separate domain.

The Rovina deposit domains for Au and Cu outlined by this work are as follows:

- OXIDE cap
- GLAM breccia
- POA in all alterations
- IMB in all alterations – except MACE and POT
- IMB in MACE
- IMB in POT
- POB in MACE
- POB in POT
- POC in all alterations – except MACE and POT
- POC in MACE
- POC in POT
- POD in all alterations – except MACE and POT
- POD in MACE
- POD in POT
- POE (in all alterations, but primarily KSIL alteration)

14.4.1.2 Colnic Deposit

The box plots for Au by lithology show C_Por, Cd_Por, IMB, F2_Por, and PM_Por clearly stand out as separate domains, with higher Au values. The Cu values are similar to gold, except for the satellite area, which shows elevated Cu values. The CD_Por unit is viewed as an extension of the C_Por unit at depth, although the grades are generally lower in this unit. The Chubby's Fault, LM, and LD dyke units show elevated values and, therefore, form their own domains.

Box plots sorted by alteration show that the elevated values for Au and Cu are in the K alteration, followed by K2_MACE, K3, and TRPH. The distribution for Au and Cu is similar, apart from the K3_ARG alteration located in the Satellite porphyry area, which shows elevated Cu values.

ANOVA was also used at the Colnic deposit to assist in the separation of the statistical domains.

From the information obtained during the site visit, the box plot, the mean values, and the ANOVA, the following conclusions were reached:

- The lithological units appear to have the most weight in determining the grade of the deposit; however, the K, K2_MACE, and K3_TRPH alterations cannot be discounted.
- The F2_POR and IMB are adjacent to each other, with contact profiles suggesting a soft boundary. The mean values are also very similar; however, from the description "...magmatic foliations are also common, especially adjacent to the margins of the F2-Hill porphyry" it appears the contact is fairly distinguishable in the core and was, therefore, modelled as two separate domains.
- The LD_POR is a low-grade propylitic alteration plug inside the F2_POR, with radiating dykes extending away from the F2_POR. The boundary analysis shows a sharp boundary with the F2_POR.
- Like the Rovina deposit, the K3_ARG and PHILLIC alteration are adjacent to each other and show a soft boundary for Cu but not for Au. Grades are relatively low, regardless of the lithological unit.
- The satellite porphyry area is located mostly in the K3_ARG and PHYLLIC alterations. By using lithology as the main control for the domains, the Au and Cu variance in the K3_ARG and PHYLLIC alterations will be automatically honoured.
- The K3 and TRPH alterations are adjacent to each other. A boundary analysis suggests soft to semi-soft boundaries for both Au and Cu. However, the Cu grade difference between these two alterations suggests they should be interpolated separately. This contrasts with Au, where the grade variance is minor, indicating they can be combined.

The Colnic domains determined by this work are as follows:

- OXIDE cap
- C_Por in K alteration
- C_Por in K2_MACE alteration
- C_Por in K3+TRPH alteration combination for gold
- C_Por in K3 alteration for copper
- C_Por in TRPH alteration for copper
- F2_Por in all alterations except K2_MACE and K3+TRPH combination
- F2_Por in K2_MACE alteration
- F2_Por in K3 and TRPH alteration combination for gold
- F2_Por in K3 alteration for copper
- F2_Por in TRPH alteration for copper
- IMB in all alterations except K, K2_MACE, and K3+TRPH combination
- IMB in K alteration
- IMB in K2_MACE alteration
- IMB in K3 and TRPH alteration combination for gold
- IMB_Por in K3 alteration for copper
- IMB_Por in TRPH alteration for copper
- WR_Por in all alterations except K, K2_MACE, and K3+TRPH combination
- WR_Por in K2_MACE alteration
- WR_Por in K3 and TRPH alteration combination for gold

- WR_Por in K3 alteration for copper
- WR_Por in TRPH alteration for copper
- SED in all alterations

Satellite porphyry area:

- G_Por in all alterations
- S_BX in all alterations
- S_POR in all alterations

Pre- and post-mineralised dykes and faults are separate domains:

- FLT – Chubby's Fault
- LD – mostly barren dyke
- LM – medium to high grade late mineralised dyke swarm

14.4.1.3 Ciresata Deposit

At Ciresata the lithological and alteration domains are simpler, with an EM_P central core surrounded by HR_P and SED, cut by late mineralised (LM) dykes and an IMP plug at depth. Alteration is a high-grade potassic (K) central core, surrounded by a phyllic halo.

The Ciresata domains were defined by combining the lithology and alterations. The late mineralised dyke and IMP lithologies in K and PH alterations were grouped together to provide sufficient data points. Cu and Au were statistically correlated; therefore, the domains for these two elements were the same. The final domains were defined as follows:

- EM_P in K alteration
- EM_P in PH alteration
- EM_P in K and PH for the Northeast dyke-like domain
- HR_P in K alteration
- HR_P in PH alteration
- IMP in K and PH alterations
- SED in K alteration
- SED in PH alteration
- WP in K alteration
- WP in PH alteration
- Late-mineralised dyke
- LM_P in K and PH alterations

14.5 ASSAYS

Typical of porphyry deposits, the units have a low coefficient of variation values, confirming the low variability of the grades in the main mineralised units.

Table 14.4 shows the mean assay values for the different statistical domains for the three deposits. The Au and Cu distribution in the RVP can be described as follows:

- Rovina is considered a copper/gold porphyry deposit, with an Au:Cu ratio of 1.3:1

- Colnic is considered a gold-rich porphyry deposit with minor copper, with an Au:Cu ratio of 5.3:1
- Ciresata is also considered a gold-rich porphyry with minor copper, with an Au:Cu ratio of 4.7:1

Table 14.4: Raw Assay Mean Grade Statistics

Rovina – Mean Value by Domains				Colnic – Mean Value by Domains				Ciresata – Mean Value by Domains			
Domains	Nb	Au (g/t)	Cu (%)	Domains	Nb	Au (g/t)	Cu (%)	Domains	Nb	Au (g/t)	Cu (%)
GLAM	6,628	0.154	0.082	C_POR+K	3,726	0.826	0.144	WP+PH	730	0.061	0.010
IMB+ALL	3,872	0.071	0.043	C_POR+K2_MACE	195	0.491	0.122	HR-P+PH	6,157	0.104	0.026
IMB+MACE	2,077	0.210	0.149	C_POR+K3/TRPH	1647	0.405	0.087	SED+PH	3,349	0.139	0.035
IMB+POT	1,473	0.153	0.135	F2_POR+all	1,248	0.097	0.008	LM-P+PH	717	0.163	0.032
POA	1,085	0.064	0.060	F2_POR+K2_MACE	1,940	0.442	0.110	IMP+PH	227	0.233	0.040
POB+MACE	2,798	0.559	0.247	F2_POR+K3/TRPH	3,171	0.285	0.076	LM-P+K	951	0.272	0.045
POB+POT	795	0.267	0.227	FLT	773	0.499	0.065	HR-P+K	4,074	0.355	0.090
POC+ALL	2,608	0.102	0.096	G_POR	403	0.019	0.005	WP+K	900	0.358	0.071
POC+MACE	3,935	0.353	0.320	IMB+all	3,368	0.102	0.010	IMP+K	966	0.538	0.094
POC+POT	8,870	0.202	0.216	IMB+K	1,160	0.579	0.112	SED+K	21,031	0.541	0.117
POD+ALL	1,689	0.114	0.132	IMB+K2_MACE	680	0.592	0.096	EM-P+PH	87	0.616	0.112
POD+MACE	277	0.310	0.214	IMB+K3/TRPH	3,857	0.339	0.044	EM-P+K	3882	0.945	0.161
POD+POT	791	0.196	0.159	LD	1,323	0.143	0.026	-	-	-	-
POE+ALL	809	0.088	0.096	LM	1,526	0.411	0.077	-	-	-	-
SED	188	0.034	0.014	OXIDE	55	0.113	0.022	-	-	-	-
-	-	-	-	S_BX	400	0.102	0.045	-	-	-	-
-	-	-	-	S_POR	354	0.053	0.041	-	-	-	-
-	-	-	-	SED	451	0.047	0.049	-	-	-	-
-	-	-	-	WR_POR+all	5114	0.045	0.008	-	-	-	-
-	-	-	-	WR_POR+K2_MACE	122	0.438	0.051	-	-	-	-
-	-	-	-	WR_POR+K3/TRPH	1276	0.209	0.043	-	-	-	-
Nb number of samples											

14.6 CAPPING

High-grade sample outliers can contribute excessively to the total metal content of a deposit. Probability plots and a decile analysis were used to identify grade outliers for Au and Cu.

In general, the capping strategy is applied if

- The last decile contains more than 40 % of the metal.
- The last decile contains more than 2.3 times the metal quantity in the penultimate decile.
- The last centile contains more than 10 % of the metal.
- The last centile contains more than 1.75 times the metal quantity contained in the penultimate centile.

Each statistical domain was evaluated separately for Au and Cu, and a combination of grade capping and search restrictions were used to restrict the influence of the outliers. The results of the analysis indicate that, with a few exceptions, no grade capping was warranted for Cu at Colnic and Ciresata. In all three deposits, Au required some form of restriction. Due to the higher Cu content at the Rovina deposit, outlier restrictions were applied.

For Au, the outlier population was controlled by using a high cap value on raw assays combined with a search restriction applied to the “mild” outliers. With a search restriction, composites above a given threshold are used at face value, but their range of influence is limited to a localised area. The range and grade threshold were based on a series of indicator variograms. For the Rovina deposit, 0.8 % of the composites were at or above the threshold used in the search restriction. For the Colnic and Ciresata deposits, the percentages were 0.9 % and 1.3 %, respectively. For Cu, a simple cap value with no search restriction was used.

All assays were capped prior to compositing; capping values are tabulated in Table 14.5, including the number of affected samples.

Table 14.5: Outlier Capping

Prospect	Domain Name > Code	Au Cap Level (g/t)	Au Nb Affected by Capping	Au SR (Grade g/t – Range m)	Cu Cap Level (%)	Cu Nb Affected by Capping
Rovina	GLAM -->6100	2.5	19	Not used	0.75	12
Rovina	IMB+ALL -->6700	0.7	10	0.45 – 75	0.4	11
Rovina	IMB+MACE -->6730	1.5	14	1.3 – 50	0.5	15
Rovina	IMB+POT -->6731	1.5	3	1.3 – 50	0.5	2
Rovina	POA -->6600	0.7	2	0.45 – 75	0.4	0
Rovina	POB+MACE -->6530	3.0	14	1.3 – 50	1.2	4
Rovina	POB+POT -->6531	1.7	4	1.3 – 50	0.75	4
Rovina	POC+ALL -->6400	0.7	10	0.45 – 75	0.5	5
Rovina	POC+MACE -->6430	1.5	12	1.3 – 50	1.0	10
Rovina	POC+POT -->6431	1.5	19	1.3 – 50	1.0	24
Rovina	POD+ALL -->6300	0.7	10	0.45 – 75	0.6	3

Prospect	Domain Name > Code	Au Cap Level (g/t)	Au Nb Affected by Capping	Au SR (Grade g/t – Range m)	Cu Cap Level (%)	Cu Nb Affected by Capping
Rovina	POD+MACE -->6330	1.0	2	1.0 – 50	0.5	3
Rovina	POD+POT -->6331	1.2	3	1.2 – 50	0.6	7
Rovina	POE+ALL -->6200	1.5	3	0.45 – 75	0.5	3
Rovina	SED -->6800	No Cap	0	Not used	0.1	1
Colnic	C_POR+K -->5012	4.0	12	2.0 – 25	0.50	4
Colnic	C_POR+K2_MACE -->5011	1.5	3	1.5 – 40	0.15	41
Colnic	C_POR+K3/TRPH -->5010	2.0	12	1.5 – 40	0.30	3
Colnic	F2_POR+all -->4030	3.5	1	1.5 – 40	0.10	1
Colnic	F2_POR+K2_MACE -->4011	5.0	2	1.5 – 40	0.25	22
Colnic	F2_POR+K3/TRPH -->4010	5.0	3	1.5 – 40	0.35	5
Colnic	FLT -->2000	2.0	11	Not used	0.30	4
Colnic	G_POR -->1200	0.3	3	Not used	0.03	4
Colnic	IMB+all -->8030	5.0	1	1.5 – 40	0.1	17
Colnic	IMB+K -->8012	4.0	2	2.0 – 25	0.35	5
Colnic	IMB+K2_MACE -->8011	2.0	18	1.5 – 40	0.25	4
Colnic	IMB+K3/TRPH -->8010	5.0	7	1.5 – 40	0.25	10
Colnic	LD -->1500	4.0	2	Not used	0.25	0
Colnic	LM -->7000	5.0	1	Not used	0.25	14
Colnic	OXIDE -->1600	1.8	0	Not used	0.18	1
Colnic	S_BX -->1000	0.6	3	Not used	0.20	15
Colnic	S_POR -->1100	1.5	1	Not used	0.25	0
Colnic	SED -->1300	0.3	3	Not used	0.25	3
Colnic	WR_POR+all -->1430	4.0	2	1.5 – 40	0.11	22
Colnic	WR_POR+K2_MACE -->1411	2.0	0	1.5 – 40	0.11	7
Colnic	WR_POR+K3/TRPH -->1410	2.0	6	1.5 – 40	0.11	76
Ciresata	EM-P +K -->1012	5.0	9	2.4 – 50	0.5	8
Ciresata	EM-P +PH -->1030	2.0	1	0.5 – 50	0.5	
Ciresata	HR-P +K -->3012	1.1	66	2.4 – 50	0.25	18
Ciresata	HR-P +PH -->3030	1.4	11	0.5 – 50	0.25	
Ciresata	LM-P +K -->2012	8.0	1	1.0 – 50	0.25	9
Ciresata	LM-P +PH -->2030	3.5	2	1.0 – 50	0.25	
Ciresata	SED + K -->4012	3.5	45	2.4 – 50	0.35	100
Ciresata	SED +PH -->4030	2.0	5	0.5 – 50	0.35	
Ciresata	IMP + K -->5012	2.3	9	1.0 – 50	0.3	4
Ciresata	IMP +PH -->5030	1.0	3	1.0 - 50	0.3	
Ciresata	WP + K -->6012	1.2	15	2.4 – 50	0.25	1
Ciresata	WP +PH -->6030	0.5	4	0.5 – 50	0.25	

14.7 COMPOSITES

Sampling intervals average 1.0 m. The upper third quartile of the sampling length shows a value close to 1.0 m.

Composite lengths consistent with the bulk mining scenario suggested in the 2008 PEA study were chosen. At the Rovina and Colnic deposits, a 4 m composite length was selected, providing for three data points within a 10 m × 10 m × 12 m cell size in the block model. At Ciresata, a 5 m composite length was selected, providing one data point in the 10 m × 10 m × 5 m cell size. It is expected that the composite lengths will result in model estimates that will be more representative of the grade variability expected from the bulk-mining methods envisaged for the RVP. Assays were length-weighted averaged, and any grade capping was applied to the raw assay data prior to compositing.

The composite intervals were created in a downward fashion from the collar of the holes to the bottom, resetting the composite intervals at the domain boundaries. Composite remnants (composites under half the nominal composite length) were added to the previous interval.

Six holes intersected historical underground exploration drifts at the Rovina deposit, and the sampling gaps created by these intersections were ignored during the compositing process; all other gaps in sampling were assigned a “trace” grade.

Summary statistics for composite grades are shown in Table 14.6.

Table 14.6: Composite Grade Statistics

Rovina – Mean Value by Domain				Colnic – Mean Value by Domain				Ciresata – Mean Value by Domain			
Domain Name/Code	No.	Au Cap (g/t)	Cu Cap (%)	Domain Name/Code	No.	Au Cap (g/t)	Cu Cap (%)	Domain Name/Code	No.	Au Cap (g/t)	Cu Cap (%)
GLAM -->6100	1,706	0.145	0.079	C_POR+K --> 5012	932	0.819	0.144	WP+PH --> 6030	151	0.058	0.010
IMB+ALL -->6700	1,026	0.067	0.041	C_POR+K2_MACE --> 5011	50	0.485	0.112	HR-P+PH --> 3030	1862	0.085	0.021
IMB+MACE -->6730	520	0.208	0.148	C_POR+K3/TRPH --> 5010	411	0.400	0.087	SED+PH --> 4030	780	0.125	0.032
IMB+POT -->6731	378	0.147	0.132	F2_POR+all --> 4030	321	0.094	0.008	LM-P+PH --> 2030	145	0.162	0.032
OXI -->41	193	0.101	0.059	F2_POR+K2_MACE --> 4011	484	0.424	0.110	IMP+PH --> 5030	46	0.229	0.040
POA -->6600	271	0.063	0.060	F2_POR+K3/TRPH --> 4010	801	0.277	0.075	LM-P+K --> 2012	191	0.264	0.044
POB+MACE -->6530	695	0.557	0.248	FLT --> 2000	195	0.486	0.064	HR-P+K --> 3012	832	0.348	0.089
POB+POT -->6531	199	0.263	0.226	G_POR --> 1200	123	0.018	0.005	WP+K --> 6012	180	0.355	0.071
POC+ALL -->6400	647	0.100	0.096	IMB+all --> 8030	849	0.098	0.010	IMP+K --> 5012	195	0.538	0.094
POC+MACE -->6430	984	0.350	0.319	IMB+K --> 8012	292	0.574	0.112	SED+K --> 4012	4235	0.533	0.116
POC+POT -->6431	2,196	0.197	0.216	IMB+K2_MACE --> 8011	170	0.565	0.095	EM-P+PH --> 1030	18	0.592	0.106
POD+ALL -->6300	419	0.113	0.132	IMB+K3/TRPH --> 8010	969	0.331	0.043	EM-P+K --> 1012	775	0.933	0.161
POD+MACE -->6330	69	0.307	0.213	LD --> 1500	336	0.138	0.026	–	–	–	–
POD+POT -->6331	200	0.197	0.159	LM --> 7000	388	0.399	0.077	–	–	–	–
POE+ALL -->6200	203	0.084	0.094	OXIDE --> 1600	26	0.048	0.010	–	–	–	–
SED -->6800	47	0.034	0.014	S_BX --> 1000	103	0.067	0.044	–	–	–	–
–	–	–	–	S_POR --> 1100	90	0.051	0.040	–	–	–	–
–	–	–	–	SED --> 1300	116	0.049	0.050	–	–	–	–
–	–	–	–	WR_POR+all --> 1430	1318	0.044	0.007	–	–	–	–
–	–	–	–	WR_POR+K2_MACE --> 1411	35	0.418	0.046	–	–	–	–
–	–	–	–	WR_POR+K3/TRPH --> 1410	318	0.200	0.041	–	–	–	–

14.8 BULK DENSITY

The ESM database contains a total of 1,125 specific gravity (SG) measurements. The sampling collection averaged one measurement per 95 m, 92 m, and 137 m of core at the Rovina, Colnic, and Ciresata deposits, respectively.

The samples were sent to the ALS Laboratory at Gura Rosiei where samples were dried and coated in a thin layer of lacquer or shellac before being weighed in air (**W***air*) and in water (**W***water*). The SGs were calculated using the standard formula:

$$SG = W_{air}/(W_{air}-W_{water})$$

The rock types found at Colnic and Rovina are generally non-porous. Results from the statistical study indicated that the SG is relatively consistent. It was noted that the alteration phases have more influence on the overall density on the deposit than the lithology. Table 14.7 shows the overall SG statistics for RVP.

Table 14.7: Specific Gravity by Deposit

Deposit	Count	Minimum	Maximum	Average	Standard Deviation
Ciresata	345	2.22	2.87	2.69	0.08
Colnic	368	2.37	2.88	2.65	0.08
Rovina	412	2.20	3.18	2.64	0.10

Blocks in the mineral resource model were assigned a density value based on the combined lithological and alteration codes in the model (see Table 14.8). An exception was made for domains for which there were an insufficient number of measurements; in these cases, an average density was applied, as shown in Table 14.7.

Considering the spatial spread and geological representativeness of the SG database, CCIC MinRes is of the opinion that the measurements are representative of the in-situ bulk density of the deposit, and sufficient for mineral resource estimation purposes.

Table 14.8: Bulk Density Values per Domain

Rovina Density			Colnic Density			Ciresata Density		
Domain Name —> Code	**SG**	**No. of Readings**	**Domain Name —> Code**	**SG**	**No. of Readings**	**Domain Name —> Code**	**SG**	**No. of Readings**
Oxidation	2.64 (Avg)	0	C_POR+K - 5012	2.67	58	EM-P+K - 1012	2.72	50
GLAM - 6100	2.55	69	C_POR+K3/TRPH - 5010	2.68	27	EM-P+PH - 1030	2.61 (PH avg)	1
POE-ALL - 6200	2.68	14	F2_POR+all - 4030	2.66	5	HR-P+K - 3012	2.71	37
POD-ALL - 6300	2.69	21	F2_POR+K2_MACE - 4011	2.69	27	HR-P+PH - 3030	2.61	45
POD+MACE - 6330	2.64 (Avg)	5	F2_POR+K3/TRPH - 4010	2.68	39	IMP+K - 5012	2.72 (K avg)	10
POD+POT - 6331	2.64 (Avg)	2	FLT - 2000	2.63	20	IMP+PH - 5030	2.61 (PH avg)	2
POC+ALL - 6400	2.61	12	G_POR - 1200	2.54	7	LM-P+K - 2012	2.72 (K avg)	9
POC+MACE - 6430	2.64	61	IMB+all - 8030	2.6	21	LM-P+PH - 2030	2.61 (PH avg)	6
POC+POT - 6431	2.65	95	IMB+K - 8012	2.69	9	SED+K - 4012	2.73	146
POB_MACE - 6530	2.67	52	IMB+K2_MACE - 8011	2.67	17	SED+PH - 4030	2.58	26
POB+POT - 6531	2.67	14	IMB+K3/TRPH - 8010	2.64	45	WP+K - 6012	2.72 (K avg)	8
POA - 6600	2.75	8	LD - 1500	2.67	28	WP+PH - 6030	2.61 (PH avg)	5
IMB+ALL - 6700	2.61	17	LM - 7000	2.65	20	–	–	–
IMB+MACE - 6730	2.68	28	S_BX - 1000	2.53	9	–	–	–
IMB+POT - 6731	2.75	12	S_POR - 1100	2.61	4	–	–	–
Sed - 6800	2.67	2	SED - 1300	2.69	3	–	–	–
–	–	–	WR_POR+all - 1430	2.58	9	–	–	–
–	–	–	WR_POR+K2_MACE - 1411	2.68	1	–	–	–
–	–	–	WR_POR+K3/TRPH - 1410	2.61	19	–	–	–
Average SG/Total No.	**2.64**	**412**	**Average SG/Total No.**	**2.65**	**368**	**Average SG/Total No.**	**2.69**	**345**

14.9 SPATIAL ANALYSIS

Geostatisticians use a variety of tools to analyse the pattern of spatial continuity or strength of the spatial similarity of a variable, for a separation distance and direction. The correlogram measures the correlation between data values as a function of their separation distance and direction. If samples that are close together are compared, it is common to observe that their values are quite similar, and the correlation coefficient for closely spaced samples is near 1.0. As the separation between samples increases, there is less similarity in the values, and the correlogram tends to decrease toward 0.0. The distance at which the correlogram reaches zero is called the “range of correlation” or simply the range. The range of the correlogram corresponds roughly to the more qualitative notion of the “range of influence” of a sample; it is the distance over which sample values show some persistence of correlation. The shape of the correlogram describes the pattern of spatial continuity. A very rapid decrease near the origin is indicative of short scale variability. A more gradual decrease moving away from the origin suggests longer scale continuity.

Using the Sage 2001 software, directional sample correlograms were calculated for gold and copper using a variable lag distance. After fitting the variance parameters, the algorithm then fits an ellipsoid to all ranges from the directional models for each structure. The final models of anisotropy are given by the lengths and orientations of the axes of the ellipsoids.

Variography was attempted for lithological and alteration combinations. Adjoining domains with similar statistical characteristics were combined to provide sufficient data points. Generally, the best variograms were obtained in the core of each deposit. The phyllic halo was difficult to model since the data points were distributed around the core, which led to poor results.

All anisotropy models generated by SAGE 2001 were visually inspected in Gems to compare the output with the expected geological controls on the mineralisation. A variogram was considered inconclusive if the anisotropy range and angle did not appear to coincide with any known or expected trend of the mineralisation.

Table 14.9, Table 14.10, and Table 14.11 summarise the results of the variography for the domains that were able to produce a conclusive variogram model. The traditional exponential range R in the tables is defined as Gam(3R) = 0.95 * Sill as defined by the first edition of GSLIB (Deutsch and Journel). The order and direction of the rotations around the three axes are as follows:

- The first rotation is around the Z axis; the direction is given by the right-hand rule.
- The second rotation is around the rotated Y axis; the direction is given by the right-hand rule.
- The third rotation is around the rotated Z axis; the direction is given by the right-hand rule.

14.9.1 Rovina Deposit

In general, the Rovina deposit indicated a preferred continuity in the 300° to 330° azimuth with a near vertical dip. Variography was conclusive for POB in MACE and POT alterations, IMB+POC+POD in Mace, and IMB+POC+POD in POT domain. Variography was inconclusive for the GLAM breccia and IMB+POA+POC+POD+POE in the phyllic halo. The SED and the OXIDE layer did not have adequate data points.

Table 14.9: Variogram Model Parameters – Rovina Deposit

Variogram/ (Domain Codes)	Component	Value	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
ICD_M_AU (6331, 6431, 6731)	Nugget C0	0.193							
	Exponential C1	0.807	ZXZ	-63	10	32	29.6	88.3	157.7
ICD_P_AU (6330, 6430, 6730)	Nugget C0	0.240							
	Exponential C1	0.760	ZXZ	-51	3	-8	16.9	56.5	116.8
POB_AU (6530, 6531)	Nugget C0	0.083							
	Exponential C1	0.917	ZXZ	-17	1	-67	13	14.7	70.1
ICD_M_CU (6331, 6431, 6731)	Nugget C0	0.145							
	Exponential C1	0.855	ZXZ	-63	10	-10	28.4	42.6	108.8
ICD_P_CU (6330, 6430, 6730)	Nugget C0	0.198							
	Exponential C1	0.802	ZXZ	-4	0	33	57.9	20.1	139.4
POB_CU (6530, 6531)	Nugget C0	0.257							
	Exponential C1	0.743	ZXZ	0	87	4	9.8	67.5	9.4

14.9.2 Colnic Deposit

The variogram models for the Colnic deposit were not as consistent as those for Rovina, due in part to the complex domaining that resulted in a limited number of data points for each domain, and also to the circular configuration for some of the less drilled, external domains, such as the SED unit. Variography was conclusive for the combination C_POR + IMB in K alteration, C_POR + F2_POR + IMB + WR_POR in K2_MACE and K3/TRPH alterations, and F2_POR + IMB + WR_POR in Phyllic. Variography was inconclusive for the FLT, G_POR, LD, LM, SED and all satellite porphyry (S_BX and S_POR). In general, the variograms are all steeply dipping, with an elongated component pointing down dip. In the K alteration, the variogram returned an elongated ellipsoid pointing in a northeast direction. In the K2_MACE and K3/TRPH, the long axis pointed toward the 350° azimuth.

Table 14.10: Variogram Model Parameters – Colnic Deposit

Variogram/ (Domain Codes)	Component	Value	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
S1-Au (5012, 8012)	Nugget C0	0.334	-	-	-	-	-	-	-
	Spherical C1	0.666	ZXZ	48	5	-7	55.3	45.5	90.3
S2-Au (1410, 1411, 4010, 4011, 5010, 5011, 8010, 8011)	Nugget C0	0.302	-	-	-	-	-	-	-
	Exponential C1	0.439	ZXZ	-53	-20	71	25.6	50.9	38.8
	Exponential C2	0.259	ZXZ	48	71	47	28.7	59.5	248.3
S3-Au (1430, 4030, 8030)	Nugget C0	0.472	-	-	-	-	-	-	-
	Exponential C1	0.528	ZXZ	-40	78	-11	11.1	112.8	29
S1-Cu (5012, 8012)	Nugget C0	0.334	-	-	-	-	-	-	-
	Spherical C1	0.666	ZXZ	57	58	-3	14.3	35.5	26.3
S2-Cu	Nugget C0	0.302	-	-	-	-	-	-	-

Variogram/ (Domain Codes)	Component	Value	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
(1410, 1411, 1420, 1430, 4010, 4011, 4030, 5010, 5011, 8010, 8011, 8020)	Spherical C1	0.439	ZXZ	-62	-19	-56	231.7	76.3	245.6
	Spherical C2	0.259	ZXZ	34	-85	12	194.3	345.1	1,104.9
S3-Cu (8030)	Nugget C0	0.472	-	-	-	-	-	-	-
	Exponential C1	0.528	ZXZ	109	23	-46	32.4	17.2	240.1

14.9.3 Ciresata Deposit

Variography for the Ciresata deposit was conclusive for EM-P + HR-P + SED in POT alteration, and for EM-P + HR-P + SED in Phyllic alteration. In general, the variogram model in the POT alteration pointed in the 130° to 310° azimuth. The anisotropy does not display the strong elongated vertical component seen in the Rovina and Colnic deposits.

Table 14.11: Variogram Model Parameters – Ciresata Deposit

Variogram/ (Domain Codes)	Component	Value	Rotation	Angle1	Angle2	Angle3	Range1	Range2	Range3
Au- in K (1012, 3012, 4012, 6012)	Nugget C0	0.038	-	-	-	-	-	-	-
	Exponential C1	0.774	ZXZ	31	-42	15	53.5	82.6	88.3
	Exponential C2	0.188	ZXZ	10	-89	9	71.8	93.8	42.5
Au- in PH (1030, 3030, 4030, 6030)	Nugget C0	0.189	-	-	-	-	-	-	-
	Exponential C1	0.811	ZXZ	21	46	-49	96	66.4	112
Cu- in K (1012, 3012, 4012, 6012)	Nugget C0	0.100	-	-	-	-	-	-	-
	Exponential C1	0.348	ZXZ	-3	-5	-50	80	103.7	345.7
	Exponential C2	0.552	ZXZ	-15	10	-24	377.6	159.9	320.7
Cu- in PH (1030, 3030, 4030, 6030)	Nugget C0	0.240	-	-	-	-	-	-	-
	Exponential C1	0.760	ZXZ	-26	52	-17	518.7	200.8	439.2

14.10 RESOURCE BLOCK MODEL

A single block model was constructed in Gemcom's GEMS version 6.31™ software for each of the three deposits. For Rovina and Colnic, a 10 m × 10 m × 12 m block size was selected, based on open-pit mining selectivity considerations. For Ciresata, a 10 m × 10 m × 5 m block size was selected, based on mining selectivity in a bulk underground mining scenario.

The block model matrix for all three deposits was defined on the Romanian Dealul Piscului 1970/Stereo 70 coordinate reference system with no rotation.

Table 14.12 lists the model's upper southeast corner and model definition per deposit.

Table 14.12: Block Model Definition

	Rovina	Colnic	Ciresata
Minimum Easting (m)	338,450	338,010	336,550
Minimum Northing (m)	519,650	517,110	512,600
Maximum Elevation (m)	700	700	700
Rotation Angle (degree)	0	0	0
Block size (X, Y, Z)	10 × 10 × 12	10 × 10 × 12	10 × 10 × 5
Blocks in the X Direction (Nb)	180	127	230
Blocks in the Y Direction (Nb)	180	132	160
Blocks in the Z Direction (Nb)	80	80	287

For all three deposits, the lithology and alteration models were coded using the lithology and alteration wireframes, using a 50:50 rule. For blocks with a lithology code greater than zero, the alteration code was added to obtain the combination code. For example, in Colnic the C_POR (code 5000) in K3 alteration (code 12) resulted in combination code 5012. The combination codes were exported to Microsoft Access (MSAccess) and manipulated using a dictionary replacement approach to generate the final domain codes that were used to control the grade interpolation. The domain model was re-imported into GEMS and verified for accuracy against the wireframes. These domains were used to control the Au and Cu interpolations. For Colnic, a special domain model was constructed to handle the interpolation of the Cu and Zn grades where the combined K3/TRPH for gold was split into its individual component (K3 and TRPH) alterations. The additional domain accounted for the larger difference in grade between the K3 and TRPH alterations for Cu and Zn, which did not exist for Au, as shown in Table 14.13.

Table 14.13: Mean Grade in TRPH and K3 Alterations

	Au (g/t)	Cu (%)	Zn (ppm)
TRPH	0.290	0.032	514
K3	0.328	0.081	198
Percentage Change	-12 %	-60 %	+160 %

For the Colnic and Ciresata deposits, a partial (percentage) model was used in order to handle dilution associated with the blocks overlapping the contact of the narrow LM dyke swarm, the LD dyke, the Chubby's Fault at Colnic, and the dyke-like EMP and LMP lithologies at Ciresata.

For Rovina, most of the Au and Cu mineralisation occurring within the glamm breccia is restricted to clasts of strongly veined porphyritic wallrock that have been incorporated into this unit close to the margins of the breccia pipe. Due to the random occurrences of the clasts, an indicator model was used to generate the domain codes for Au and Cu. The model was built using indicator cut-off of 0.4 g/t Au and 0.25 % Cu. A value between zero and 1 was interpolated in the indicator model representing the percentage of the block higher than the indicator value. An indicator value of 0.5 was used to separate the glamm domain model into high-grade and low-grade components. The final domain codes are representative of the high/low grades and the general position of the wireframe, which was used to optimise the search ellipsoid orientation. For example, in Table 14.14 the GLAM_HN is in the north portion of the glamm high-grade domain.

Table 14.14: Block Model Codes

Rovina – Main Model		Colnic – Main Model		Ciresata – Main Model	
Domain	**Code in Model**	**Domain**	**Code in Model**	**Domain**	**Code in Model**
OXIDE	41	S_BX	1000	EMP+K	1012
POE	6200	S_POR	1100	EMP+PH	1030
POD+ALL	6300	G_POR	1200	HRP+K	3012
POD+MACE	6330	SED	1300	HRP+PH	3030
POD+POT	6331	WR_POR+K3/TRPH (gold only)	1410	SED+K	4012
POC+ALL	6400	WR_POR+TRPH (copper and zinc)	1420	SED+PH	4030
POC+MACE	6430	WR_POR+K3 (copper and zinc)	1410	WP+K	6012
POC+POT	6431	WR_POR+K2_MACE	1411	WP+PH	6030
POD+MACE	6530	WR_POR+all	1430	-	-
POD+POT	6531	F2_POR+K3/TRPH (gold only)	4010	-	-
POA	6600	F2_POR+TRPH (copper and zinc)	4020	-	-
IMB+ALL	6700	F2_POR+K3 (copper and zinc)	4010	-	-
IMB+MACE	6730	F2_POR+K2_MACE	4011	-	-
IMB+POT	6731	F2_POR+all	4030	-	-
SED	6800	C_POR+K3/TRPH (gold only)	5010	-	-
		C_POR+TRPH (copper and zinc)	5020	-	-
		C_POR+K3 (copper and zinc)	5010	-	-
		C_POR+K2_MACE	5011	-	-
		C_POR+K	5012	-	-
		IMB+K3/TRPH (gold only)	8010	-	-
		IMB+TRPH (copper and zinc)	8020	-	-
		IMB+K3 (copper and zinc)	8010	-	-
		IMB+K2_MACE	8011	-	-
		IMB+K	8012	-	-
		IMB+all	8030	-	-
Rovina – Glamm Model		**Colnic – Dyke Model**		**Ciresata – Dyke Model**	
Domain	**Code in Model**	**Domain**	**Code in Model**	**Domain**	**Code in Model**
GLAM_HN	6122	FLT	2000	EMP (Subdomain 1)	1001
GLAM_LN	6112	LM-P	7000	LMP (Subdomain 1)	2001
GLAM_HE	6123	LD	1500	LMP (Subdomain 2)	2002
GLAM_LE	6113	OXIDE	1600	LMP (Subdomain 3)	2003
GLAM_HW	6121			IMP (Subdomain 1)	5001
GLAM_LW	6111			IMP (Subdomain 2)	5002
				IMP (Subdomain 3)	5003

14.11 GRADE INTERPOLATION

Grade interpolation was carried out using a multi-pass approach, with an increasing search dimension interpolating only the blocks that were not interpolated in the earlier pass. Ordinary kriging was used for all domains where conclusive variograms could be modelled. For domains where the variography was inconclusive, an inverse distance cubed, anisotropic-weighted methodology was employed, and the interpolated value was written to the ordinary krige element in the model to differentiate it from the full inverse distance check model. The pass number and the distance to the closest samples were written back to the model to assist in the resource classification.

For the Colnic and Ciresata deposits, the final model grade was calculated by weighting the dyke grade with the grade in the ordinary krige model using the volume of the dyke as a weighting factor.

A full Inverse Distance Squared model (ID2) with a true distance weighting, and a Nearest Neighbour (NN) model were also interpolated for model validation purposes.

14.12 BOUNDARY TREATMENT

Boundary relationships can be used to determine the inclusion or exclusion of sample data points used in the interpolation of one grade domain, and to assist in confirming geological interpretations. A gradational contact (or soft boundary) generally allows the interpolation parameters to include a limited number of samples from the adjoining domain, while a sharp contact (or hard boundary) will restrict the sample points used in the interpolation to its own domain.

As part of the domain definition, the boundary relationship between the individual lithology+alteration unit combinations were examined. Au and Cu were treated separately for this study.

Results from the analysis guided the inclusion (or exclusion) of the composites in the adjacent units during the interpolation. A semi-soft boundary included composites from the adjacent domains for any blocks overlapping the boundary in a special Pass 0. A soft boundary included samples from the adjacent domains for the most restrictive Pass 1. For the subsequent Passes 2 and 3, the composites used were restricted to their domains to prevent smearing of “distant” high-grade intercepts in one domain with the values in another. The sample inclusion matrices are included in Appendix B of the AGP PEA NI 43-101 2019 Report.

14.13 SEARCH PARAMETERS

The search ellipsoids’ orientations and dips were based on the variography results and adjusted to coincide with the geological units. The search ranges were based on the drillhole spacing for the first pass, to a maximum range based on approximately 95 % to 98 % of the sill value. Generally, the ratio between the major, semi-minor, and minor axes were kept similar to those of the three-variogram axis. Where variography was inconclusive, the search ellipsoid dimension and orientation were based on the size and shape of the 3D wireframe. Ellipsoid dimensions for various domains are listed in Table 14.15. The Gemcom ZXZ rotation angle follows the right-hand rule convention.

Table 14.15: Search Ellipsoid Dimensions by Domain

Deposit	Domain	Element	Rotation Z, X, Z (degrees)	Pass 1 Range X, Y, Z (m)	Pass 2 Range X, Y, Z (m)	Pass 3 Range X, Y, Z (m)
Rovina	IMB+POC+POD in all alterations	Au - Cu	-57, 6.5, 12	20, 48, 113	37, 86, 203	66, 156, 365
	IMB+POC+POD in POT	Au - Cu	-51, 3, -8	26, 51, 105	46, 92, 189	83, 165, 340
	IMB+POC+POD in MACE	Au - Cu	-63, 10, 32	43, 75, 200	64, 113, 300	96, 169, 450
	POB in all alterations	Au - Cu	-17, 1, -67	20, 23, 105	29, 34, 158	44, 51, 236
	POE in all alterations	Au - Cu	-50, 0, 0	60, 36, 48	90, 54, 72	135, 81, 108
	POA in all alterations	Au - Cu	-80, 3, 15	60, 30, 100	90, 45, 150	153, 77, 255
	GLAM Search Ellipsoid East	Au - Cu	28, 0, 0	39, 31, 161	55, 43, 225	77, 60, 316
	GLAM Search Ellipsoid North	Au - Cu	58, 0, 0	39, 31, 161	55, 43, 225	77, 60, 316
	GLAM Search Ellipsoid West	Au - Cu	-28, 0, 0	39, 31, 161	55, 43, 225	77, 60, 316
	SED in all alteration	Au - Cu	25, -80, 0	60, 80, 24	96, 128, 38	154, 205, 61
Colnic	C_POR+K and IMB+K	Au - Cu	38, -83, -5	30, 60, 24	54, 108, 43	97, 194, 78
	All lithos. in K2_Mace and K3/TRPH	Au - Cu	-40, -55, -22	68, 53, 38	135, 105, 75	270, 210, 150
	Phillic alteration in all lithologies	Au - Cu	-83, 78, -11	20, 100, 50	34, 170, 85	58, 289, 145
	Chubby's Fault	Au - Cu	-65, 87, 0	50, 75, 10	75, 113, 15	150, 225, 30
	Satellite area GxP, SED	Au - Cu	-31, -63, 0	45, 60, 27	77, 102, 46	130, 173, 78
	Satellite area SBx	Au - Cu	-83, -85, 0	45, 60, 27	77, 102, 46	130, 173, 78
	Satellite area SAT_POR	Au - Cu	-72, -85, 0	45, 60, 27	77, 102, 46	130, 173, 78
	LD dyke	Au - Cu	88, 86, 0	63, 113, 38	100, 180, 60	160, 288, 96
	LM	Au - Cu	32, 79, 0	90, 60, 40	162, 108, 72	275, 184, 122
Colnic	K in all lithologies	Au - Cu	40, -42, 12	36, 51, 57	58, 82, 91	161, 228, 255
	PH in all lithologies	Au - Cu	21, 46, -49	67, 46, 78	108, 74, 125	183, 126, 213
	LMP in K + PH Sub-domain 1	Au - Cu	-52, -87, 0	60, 60, 15	180, 180, 27	194, 194, 49
	LMP in K + PH Sub-domain 2	Au - Cu	-50, -78, 0	60, 60, 15	180, 180, 27	194, 194, 49
	LMP in K + PH Sub-domain 3	Au - Cu	-88, 83, 0	60, 60, 15	180, 180, 27	194, 194, 49

14.14 MODEL VALIDATION

The RVP resource models were validated by three methods:

- Visual comparison of colour-coded block model grades with composite grades on section plots.
- Comparison of the global mean block grades for ordinary kriging, inverse distance, NN models, composite grades, and raw assay grades.
- Comparison using grade profiles or swath plots in the X, Y, and Z directions, inspecting the results for local bias in the estimate.

14.14.1 Visual Comparisons

Visual comparisons of the block model grade with composite grade showed a reasonable correlation between the values. No significant discrepancies were apparent from the sections and plans reviewed.

14.14.2 Global Comparisons

A list of the main mineralisation domains for each of the three deposits are contained in Table 14.16. Grade statistics for the raw assay, composite, ordinary kriging, nearest neighbour, and inverse distance models are shown in Table 14.17; Figure 14.9 and Figure 14.10 show the differences.. Grade statistics for the composite mean grades compared very well to raw assay grades. The model mean grade compared against the composites showed a relatively steep reduction in values for Au and Cu. This reduction is due to the wireframe extending at or beyond the limit of the drilling, which introduced a series of un-interpolated blocks that are reported in the global comparison at "trace" grade, as expected towards the periphery of a porphyry.

Table 14.16: Main Mineralisation Domains

Rovina	Colnic	Ciresata
IMB+MACE	CXP_K	EMP-K
IMB+POT	CXP_K3T	HRP-K
POB+MACE	CXP_MACE	SED-K
POB+POT	F2P_K3T	WP_K
POC+MACE	F2P_MACE	
POC+POT	IMB_K	
POD+MACE	IMB_K3T	
POD+POT	IMB_MACE	
	WRP_K3T	
	WRP_MACE	

Table 14.17: Global Statistical Comparison

	Rovina		Colnic		Ciresata	
	Au (g/t)	Cu (%)	Au (g/t)	Cu (%)	Au (g/t)	Cu (%)
Within all lithologies at 0.00 Au g/t Cut-Off – Cat 0-3						
Assay	0.200	0.159	0.307	0.058	0.440	0.093
Composite	0.197	0.156	0.304	0.058	0.404	0.085
NN Model	0.115	0.101	0.200	0.044	0.272	0.060
ID model	0.117	0.103	0.199	0.044	0.269	0.060
OK Model	0.116	0.101	0.201	0.044	0.269	0.060
Major ore-bearing lithologies at 0.00 Au g/t Cut-Off - Cat 0-3						
Assay	0.275	0.225	0.458	0.089	0.563	0.118
Composite	0.275	0.225	0.458	0.088	0.554	0.117
NN Model	0.218	0.194	0.361	0.080	0.463	0.099
ID model	0.222	0.197	0.360	0.081	0.460	0.099
OK Model	0.219	0.195	0.361	0.080	0.461	0.099

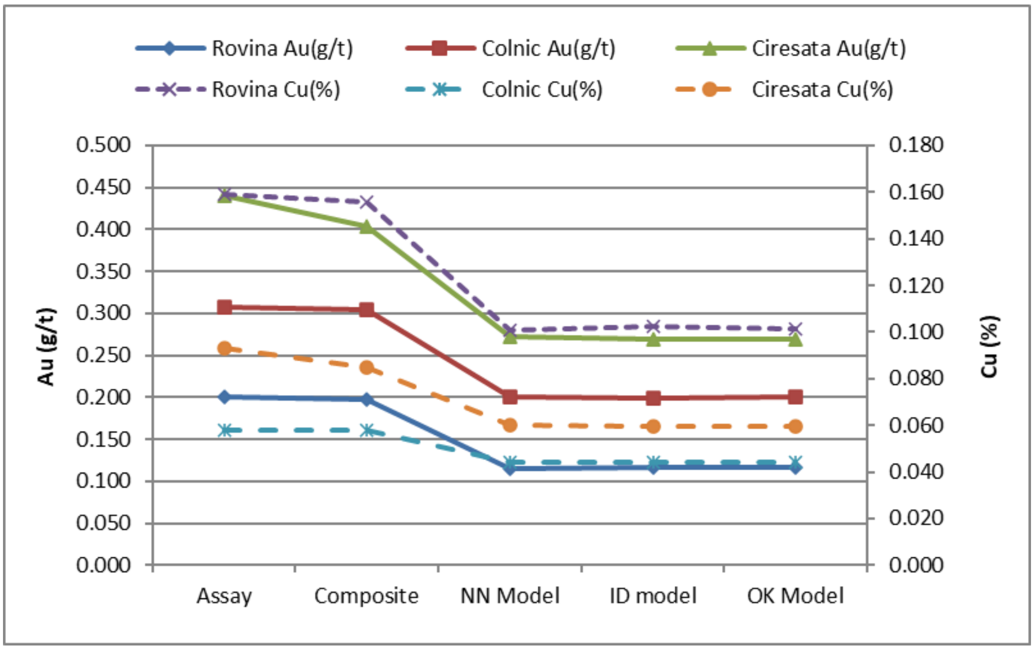


Figure 14.9: Global Statistical Comparison – All Lithologies at 0.00 Cut-Off

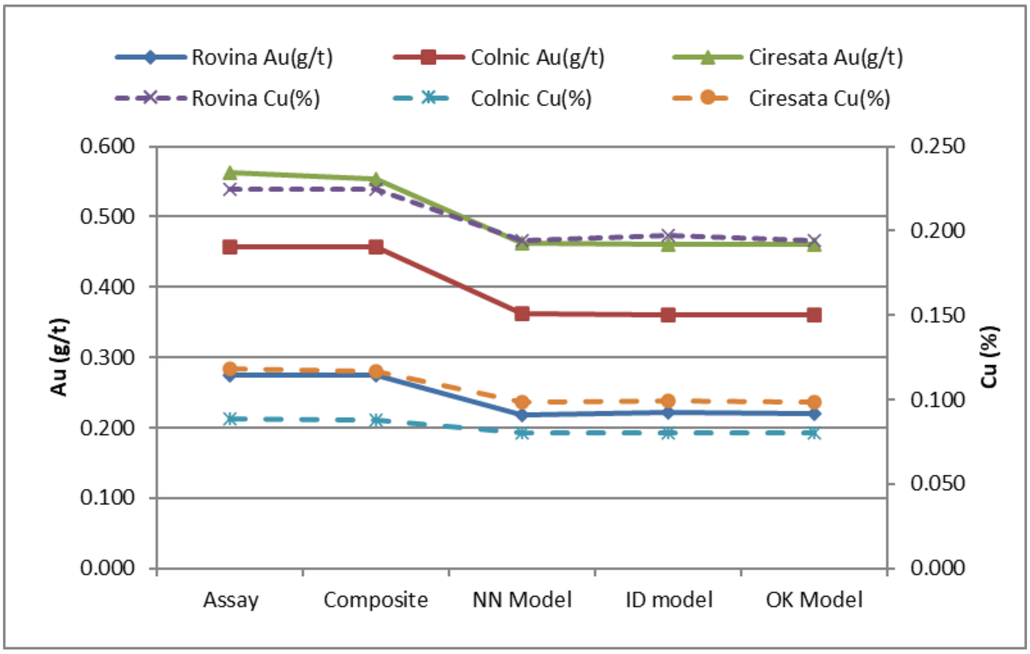


Figure 14.10: Global Statistical Comparison – Main Mineralisation Lithologies at 0.00 Cut-Off

14.14.3 Local Comparisons – Swath Plots

Comparison of the grade profiles (swath plots) of the raw assay, composites, and estimated grades allows for a visual verification of an over or underestimation of the model at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots. The output consists of three swath plots generated at 50 m intervals in each of the X, Y and vertical directions for Au and Cu.

Both the inverse distance and krige estimates should be smoother than the NN estimate; the NN estimate should fluctuate around the inverse distance estimate on the plots or display a slightly higher grade. The composite line is generally located between the assay and the interpolated grade if there are a significant number of composites. A model with good composite distribution should show very few crossovers between the composite and the interpolated grade line on the plots. In the fringes of the deposits, as composite data points become sparse, crossovers are often unavoidable. The swath size also controls this effect to a certain extent; if the swaths are too small, then fewer composites will be encountered, which results in a very erratic line pattern on the plots.

Due to the elongated cylinder nature of the deposits, there are no preferred orientations for the swath plots. In general, the swath plots show agreement between the three interpolation methodologies used, with no major local bias. At the Colnic deposit, the resource model appears to return higher grades in the northeast corner of the deposit than the composite data, possibly indicating smearing of the high-grade values. This trend is like the 2008 mineral resource estimate, and more than likely resulted from the presence of the LD dyke in that area of the model, which would significantly lower the composite average in that region. The reduced data coverage in the satellite area of the model may also have contributed to this trend. Figure 14.11, Figure 14.12, and Figure 14.13 show examples of the swath plots for the main pay elements in the Z direction.

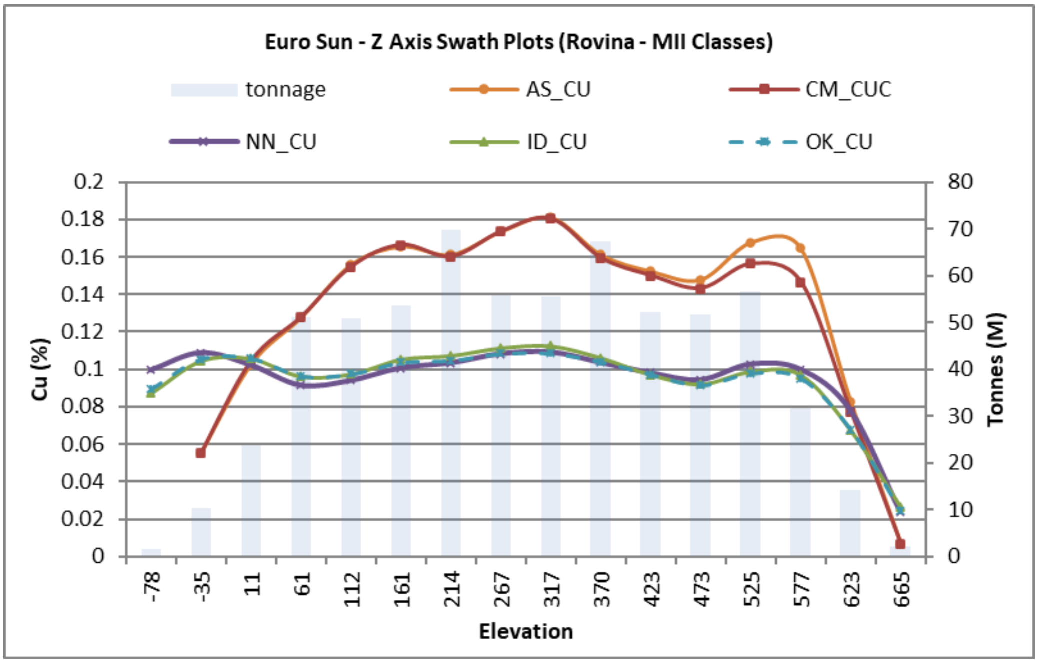


Figure 14.11: Rovina Cu Swath Plot – Z Direction

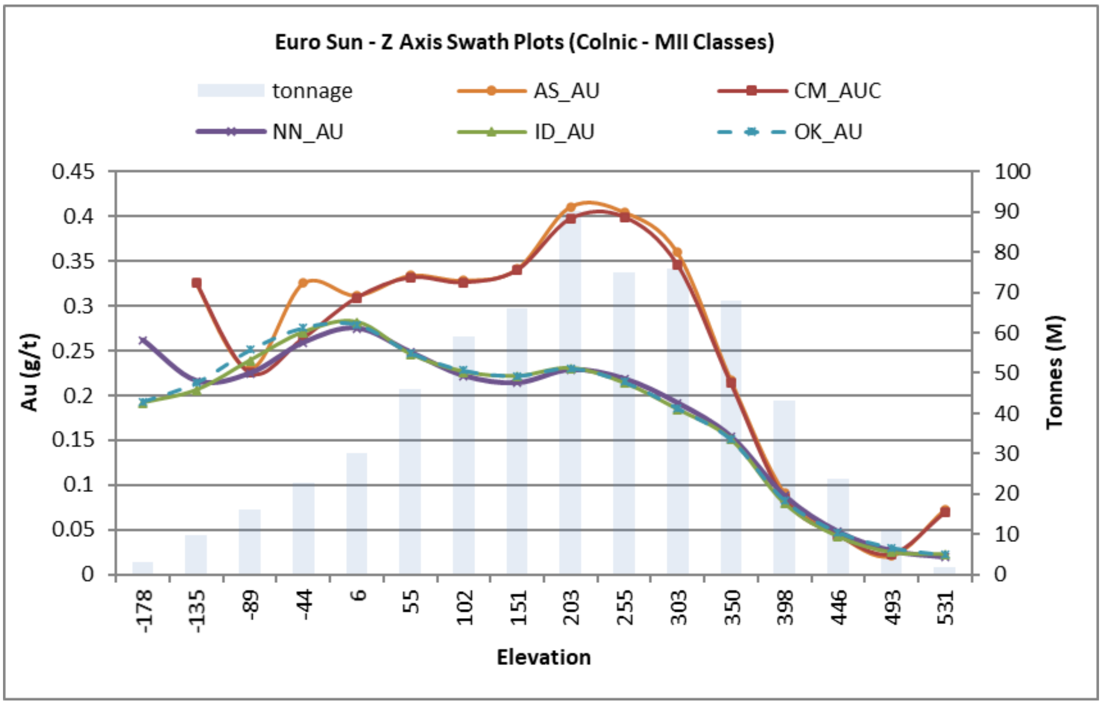


Figure 14.12: Colnic Au Swath Plot – Z Direction

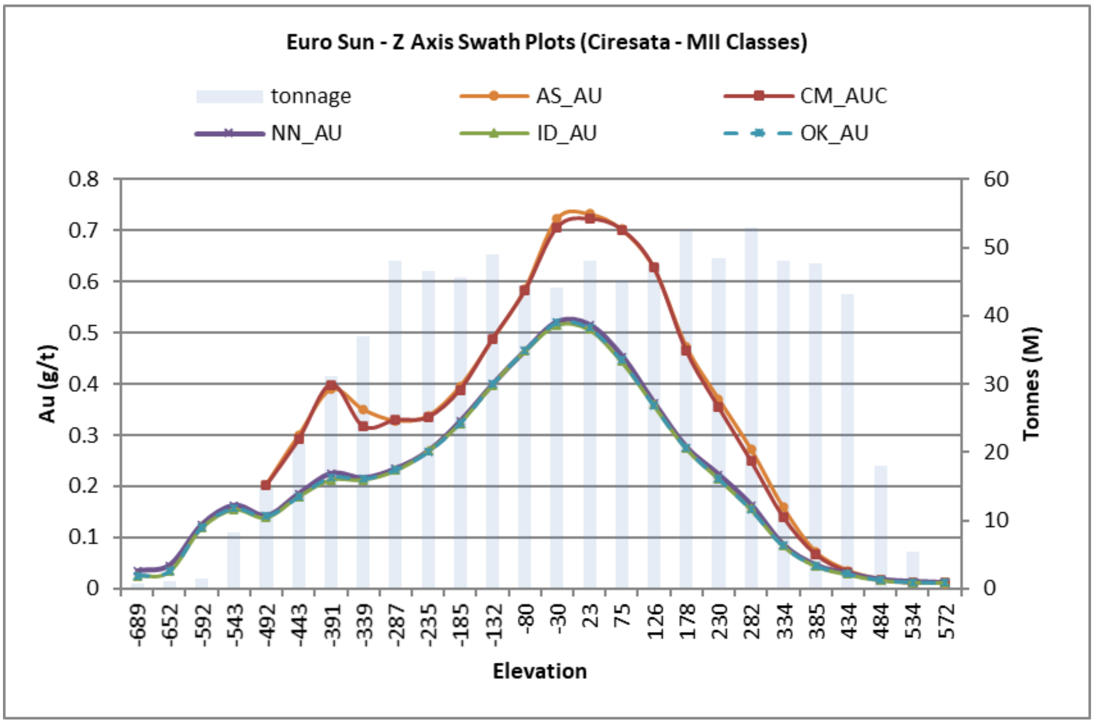


Figure 14.13: Ciresata Au Swath Plot – Z Direction

14.15 MINERAL RESOURCE CLASSIFICATION

Several factors are considered in the definition of a resource classification:

- Canadian Institute of Mining, Metallurgy and Petroleum (CIM) requirements and guidelines (2014)
- Experience with similar deposits
- Spatial continuity
- Confidence limit analysis
- Geology

No environmental, permitting, legal, title, taxation, socio-economic, marketing or other relevant issues are known to the QP that may currently affect the estimate of mineral resources. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Typically, the confidence level for a grade in the block model is reduced with the increase in the search ellipsoid size, along with the diminishing restriction on the number of samples used for the grade interpolation. This is essentially controlled via the pass number of the interpolation plan, as described in the previous section. A common technique is to categorise a model based on the pass number and distance to the closest sample.

For the RVP, classification of the mineral resource used the pass number from the interpolation plan with assistance from a drillholes density map and the actual position of the drillholes in order to minimise having blocks in the Measured category in close proximity to blocks in the Inferred category.

Two additional models were created and used in the categorisation, and are described as follows:

- A model in the block model matrix representing the drill position (Inside_DDH) was created first by assigning the percentage of the blocks inside a 50 m extruded drillhole trace. The model contains values from zero to 100 %, representing the volume occupied by the extruded drillhole trace, where 100 % means the block is fully within the trace of the drillhole.
- A second model representing the drill density called NB Holes was created in the block model matrix and contains the number of drillholes visible from a given block. The model contains values from 0 to 15, representing the number of drillholes visible within a 75 m search bubble from the block centre.
- The pass number was the primary driver to define the Measured, Indicated, and Inferred category. Drill density and drill position models were used to adjust the categorisation according to the parameters in Table 14.18.

Table 14.18: Mineral Resource Classification Parameters

Pass Number	Retained As	Downgraded To
Pass 1	Measured if number of DDH ≥ 7	Indicated if number of drillholes within a 75 m search was less than 7
Pass 2	Indicated if number of DDH ≥ 2 and block is within a drillhole trace	Inferred if number of DDH < 3 and block is outside a drillhole trace
Pass 3	Inferred if the number of DDH between 0 and 2	Code 4 if no drillhole was present within a 75 m search. These blocks do not contribute to the resources.

Adjustments to the classification of individual block values were required to create areas suitable for mine planning. This is accomplished by adjusting the confidence values of isolated blocks to create contiguous resource blocks with reasonably smooth class values and also to eliminate any small pockets of Inferred mineralisation within the most densely drilled portion of the deposit where, in the QP's opinion, additional drilling would not materially improve the estimate. A GEMS™ Cypress-enabled script adjusts or "grooms" isolated blocks. Individual blocks were upgraded or downgraded, depending on the category value of the 26 surrounding blocks. Small pockets of Inferred resource were upgraded to Indicated if they were in the most densely drilled portion of the deposit using a core area model.

Three confidence categories exist in the model. The usual CIM guideline classes of Measured, Indicated, and Inferred are coded 1, 2, and 3, respectively. A special Code 4 represents material outside of the criteria used to classify the resources. The assigned Code 4 was kept in the resource model files solely to aid ESM staff in conducting its exploration activity.

The classified model was verified by visual checks in 3D. Histograms were also generated of the distance to the closest composites versus the class model value to evaluate the class model for reasonableness. Table 14.19 shows the statistical distribution of the distance to the nearest composite by class for all three deposits.

Based on the classification criteria outlined above, of the blocks estimated for Rovina, Colnic, and Ciresata, on average 2 % are classified as Measured, 14 % as Indicated, and 13 % as Inferred. The remaining blocks are either non-interpolated or flagged as Code 4. Table 14.20 shows the distribution of the resource classification by deposits within the block model.

Table 14.19: Distance to the Nearest Composite Distribution

	Measured (m)	Indicated (m)	Inferred (m)
Mean	27	42	84
25th percentile	15	25	59
Median	24	37	78
75th percentile	36	55	103

Table 14.20: Resource Classification by Deposit

	Rovina	Colnic	Ciresata
Measured	2 %	2 %	2 %
Indicated	13 %	19 %	11 %
Inferred	18 %	14 %	7 %
Code 4 or non-interpolated	56 %	65 %	80 %

14.16 METAL EQUIVALENT FORMULA

The metal equivalent value is calculated as follows:

- For each element, a factor is calculated:

  copper (expressed as a percentage):

  Factor = Net Metal Price * Recovery * (Conversion percentage to troy pound * 100)

  gold (expressed in grams per tonne):

  Factor = Net Metal Price * Recovery * Conversion gram to troy ounce.

  Recoveries were assumed at 100 %

- For all elements, the value per tonne is calculated in US dollars

  Value/tonnes = grade * factor

  Total MV is the addition of the value per tonne expressed in US dollars for each element:

  MV = Value/tonnes Cu + Value/tonnes Au.

The copper equivalent for Rovina is calculated by dividing the MV by the copper price and multiplying by the percentage-to-pound conversion factor.

The gold equivalent for Colnic and Ciresata is calculated by dividing the MV by the gold price and then dividing by the grams-to-ounces conversion factor.

Table 14.21 lists the 2021 prices and recoveries used in the calculation.

The conversion factors used are as follows:

- Percentage to pounds, multiplied by 22.04622
- Parts per million to percentage, multiplied by 0.0001
- Grams to troy ounces, multiplied by 0.03215074 or (1/ 31.1034768)

Table 14.21: MV Input Parameters (2021)

Metal in Model	Metal Price Unit	Metal Price	Recovery
Copper (%)	US$/lb	US$3.50	100 %
Gold (g/t)	US$/oz	US$1,700	100 %

14.17 CCIC MINRES AUDIT

The 2021 DFS focused on the open-pit Rovina and Colnic deposits. The Ciresata deposit is envisioned as a bulk underground mining operation and will be evaluated for its economic potential in a separate study. As part of the DSF, CCIC MinRes completed a technical audit of the mineral resources at the Rovina and Colnic deposits. Parts of the audit were done using Leapfrog Geo ® software for validation of the geological wireframe models and Datamine Studio RM® software for validation of the mineral resource block models.

Mr Subramani visited site from the 9 to 12 November 2020 and completed the following:

- An overview of the RVP geology and the controls for mineralisation
- Verification of the drilling, core handling, core logging, and sampling procedures
- A review of the QA/QC sampling programme and monitoring of the results
- Verification of the core storage facilities and security
- Review of the density measurements
- A visit to the field and outcrops
- A visit to the ALS Chemex laboratory in Gura Rosiei to verify the sample preparation and analysis procedures

The audit included a review of the interpreted lithology and alteration wireframes against the drillholes. ESM geologists created the original wireframes in 2008, from a series of hand-drawn cross-sectional and plan-view interpretations by exploration geologists with in-depth knowledge and understanding of the deposits. In 2012, these wireframes were adjusted in Micromine to incorporate new drilling up to 31 May 2012. CCIC MinRes interrogated the wireframes for plausibility of interpretations and checked that the drillhole intercepts are honoured in 3D.

CCIC MinRes completed its own sample flagging, domain coding, sample compositing, statistical and geostatistical analyses, using Datamine Studio RM. The results of the statistical analysis were similar to the sample composites of AGP, with an average grade difference of + 2 % and +2 % for Au and Cu, respectively, at the Rovina deposits, and + 3 % and +2 % for Au and Cu, respectively, at the Colnic deposit. The slightly positive difference in the mean values is thought to be because of the differences in the approach to sample outlier capping and compositing between AGP and CCIC MinRes.

CCIC MinRes used grade iso-surfacing in Leapfrog Geo to guide the anisotropic trends during variography, which was done in Datamine Studio RM. Results of the variogram modelling showed similar anisotropic orientations, nugget values and spatial range values to those of AGP.

A Quantitative Krige Neighbourhood Analysis, using the CCIC MinRes variogram models, was done to determine the optimum estimation parameters. CCIC used ordinary kriging as the interpolation method for domains with conclusive variogram models and inverse distance squared for the remaining domains; the same approach employed by AGP. CCIC MinRes maintained the same parent block size of 10 m × 10 m × 12 m, in the X, Y and Z directions, respectively. CCIC MinRes also used the same mineral resource classification methodology, to maintain consistency in the comparisons between the three mineral resource categories.

14.17.1 Audit Findings and Comments

The main aim of the audit was to validate the Measured and Indicated mineral resources, because these resource categories were the subject of the DFS, and estimation of Proven and Probable mineral reserves. CICC MinRes, therefore, used the 2019 Lerchs-Grossmann mineral resource constraining shell as the limits for the comparisons as shown in Figure 14.14, which illustrates the limits of both the mineral resource constraining shell (black) and the mine design shell (red).

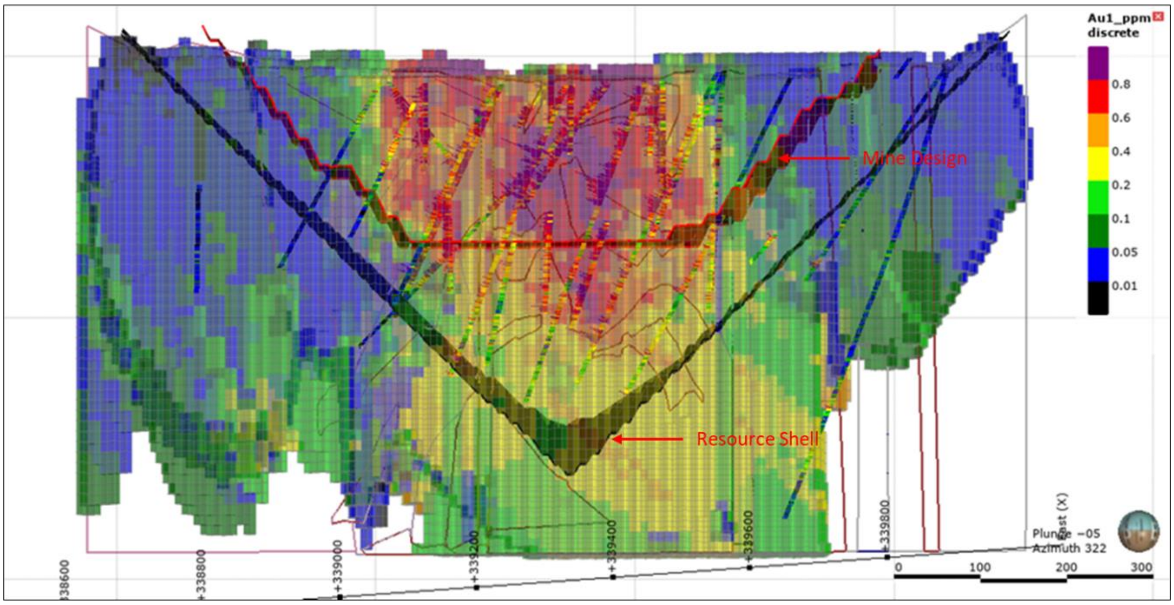


Figure 14.14: Colnic Deposit Showing Mine Design and Resource Shells

Summary comparisons between the AGP mineral resource model and the CCIC MinRes audit model for the Rovina and Colnic deposits are given in Table 14.22 and Table 14.23, respectively, with the corresponding grade/tonnage graphs for Cu and Au in Figure 14.15 and Figure 14.16. For the combined Measured and Indicated mineral resource categories at the Rovina deposit, there is 0 % difference in the tonnage, 3 % increase in the Au grade (0.29 g/t vs 0.30 g/t) and 1 % increase in the Cu grade (0.24 % vs 0.25 %). The Cu grade/tonnage graphs for the Rovina deposit show that the two models are very similar across the range of grade cut-offs. The CCIC MinRes model has higher average grades at cut-off grades above 0.6 % Cu; thought to be related to the differences in approaches to dealing with the outlier sample grade capping and search restrictions. This, however, affects a very small percentage of the overall model tonnage.

For the Colnic deposit, the combined Measured and Indicated mineral resource categories show a 2 % decrease in the tonnage, 4 % increase in Au grade (0.28 g/t vs 0.29 g/t) and 2 % decrease in Cu grade (0.06 % vs 0.05 %). The Au grade/tonnage graphs for the Colnic deposit show that the two models are very similar across the range of grade cut-offs. The CCIC MinRes model has higher average grades at cut-off grades above 1.7 g/t Au, thought to be related to the differences in outlier sample grade capping strategies. This, however, affects a very small percentage of the overall model tonnage.

The outcomes of the technical audit confirm the robustness of the AGP mineral resource model for the Rovina and Colnic deposits. CCIC MinRes is, therefore, of the opinion that these mineral resource models are sufficiently reliable for use in mineral resource and reserve estimation.

Table 14.22: Rovina Resource Model Comparison, AGP vs CCIC MinRes

AGP resource model inside resource shell

RES CLASS	TONNES	Au (g/t)	Cu (%)
Measured	32,721,691	0.36	0.29
Indicated	76,418,970	0.27	0.22
Inferred	15,565,758	0.19	0.19
Measured & Indicated	109,140,661	0.29	0.24

CCIC MinRes resource model inside resource shell

RES CLASS	TONNES	Au (g/t)	Cu (%)
Measured	32,700,451	0.38	0.30
Indicated	76,388,107	0.27	0.22
Inferred	15,565,023	0.16	0.15
Measured & Indicated	109,088,557	0.30	0.25

Percentage difference

TONNES	Au (g/t)	Cu (%)
0 %	6 %	3 %
0 %	2 %	-1 %
0 %	-15 %	-23 %
0 %	3 %	1 %

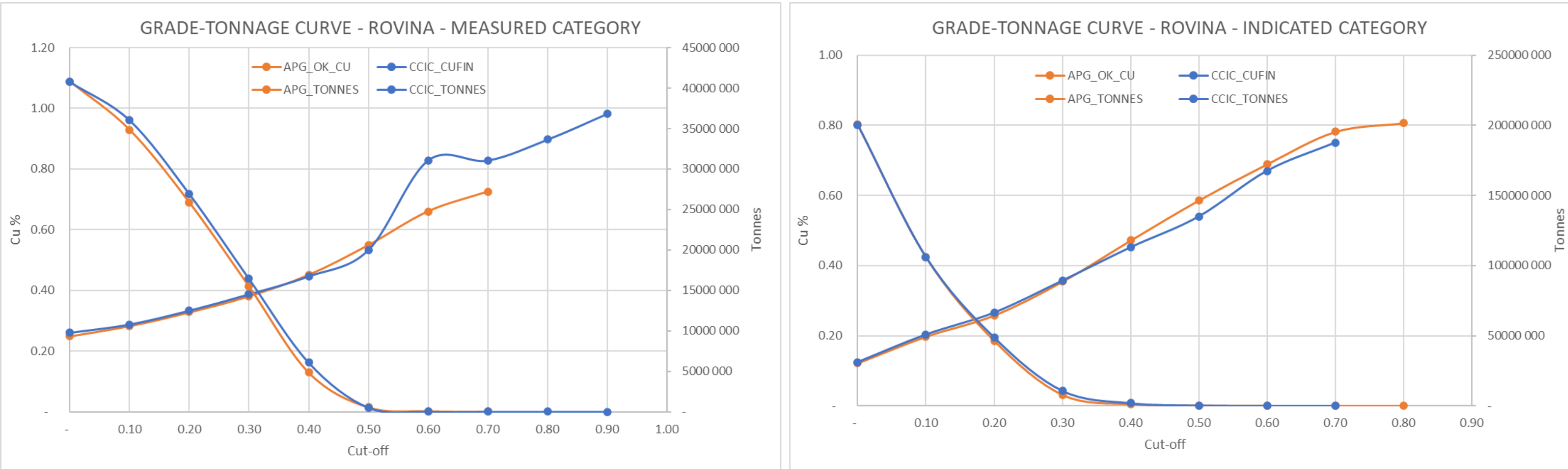


Figure 14.15: Rovina Grade-Tonnage Graphs, AGP vs CCIC MinRes

Table 14.23: Colnic Resource Model Comparison, AGP vs CCIC MinRes

AGP resource model inside resource shell

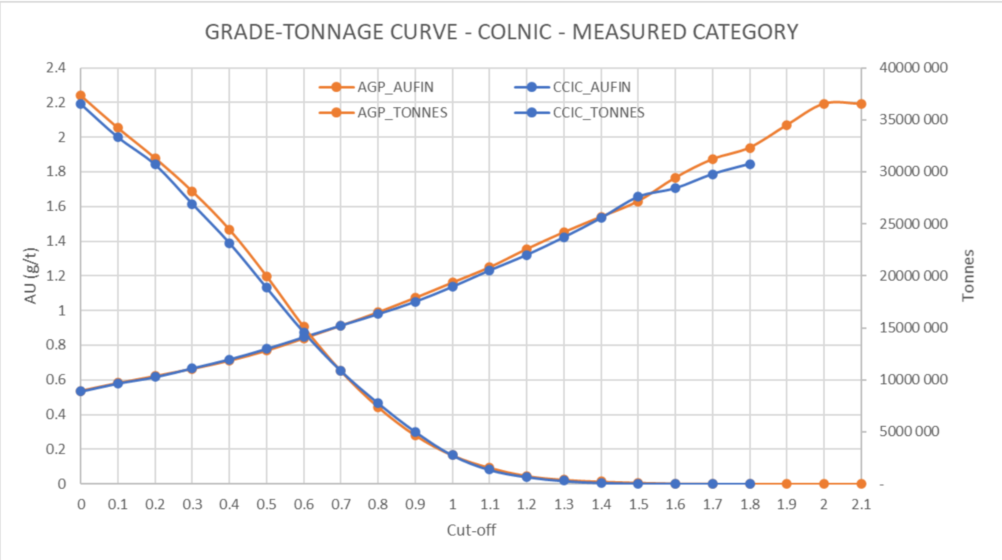
RES CLASS	TONNES	Au (g/t)	Cu (%)
Measured	37,330,220	0.54	0.10
Indicated	279,504,400	0.24	0.05
Inferred	64,627,258	0.07	0.02
Measured & Indicated	316,834,620	0.28	0.06

CCIC MinRes resource model inside resource shell

RES CLASS	TONNES	Au (g/t)	Cu (%)
Measured	36,506,206	0.53	0.09
Indicated	275,089,148	0.25	0.05
Inferred	63,709,332	0.07	0.02
Measured & Indicated	311,595,353	0.29	0.05

Percentage difference

TONNES	Au (g/t)	Cu (%)
-2 %	-1 %	-2 %
-2 %	5 %	-1 %
-1 %	5 %	0 %
-2 %	4 %	-2 %



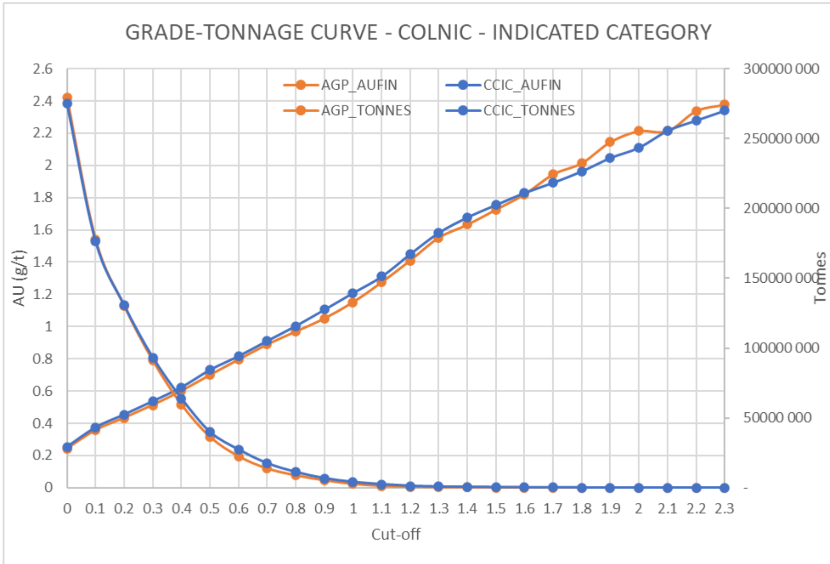


Figure 14.16: Colnic Grade-Tonnage Graphs, AGP vs CCIC MinRes

14.18 MINERAL RESOURCE STATEMENT

Effective 01 March 2021, CCIC MinRes has updated the mineral resource estimates for the Rovina and Colnic deposits to reflect the current metal prices, updated operating costs and operating parameters of the DFS. The geological and mineral resource block models remain unchanged from those used by AGP in 2019. In this current estimate, the Lerchs-Grossmann mineral resource constraining pit shell was updated using the updated input parameters. A summary of the input parameters that were used in 2019 and in 2021 are shown in Table 14.24. Input parameters that have changed positively are coloured in green, those that have changed negatively are coloured in red, and those that remained unchanged from 2019 are coloured in black. Metal prices have increased by approximately 13 % and 6 % for Au and Cu, respectively. However, Au recovery has decreased and Cu recovery for the Rovina deposit has increased by 4.5 %. The significant negative changes are the incremental mining costs (per 12 m bench) and processing costs that have increased by 19 %.

Table 14.24: Input Parameters – Lerchs-Grossmann Constraining Pit Shell

Design Parameter	Unit	2019		2021	
		Rovina	Colnic	Rovina	Colnic
Metal Prices					
Copper (World Price)	US$/lb	3.30	3.30	3.50	3.50
Copper (Net Price after charges)	US$/lb	2.61	2.61	2.75	2.75
Gold (World Price)	US$/oz	1500	1500	1700	1700
Gold (Net Price after charges)	US$/oz	1,384	1,384	1449	1449
Recoveries					
Copper Recovery	%	88.5	88.5	92.5	88.5
Gold Recovery	%	81.5	81.5	75.5	80.6
Operating Costs					
Base Elevation for Costs	masl	496	365	496	370
Mining Cost – Waste (Base Cost)	US$/t material	1.50	1.50	1.40	1.50
Incremental Cost above Base	US$/t/12 m bench	0.01	0.01	0.03	0.03
Incremental Cost below Base	US$/t/12 m bench	0.02	0.02	0.03	0.03
Mining Cost – Plant Feed	US$/t material	1.50	1.50	1.50	1.60
Incremental Cost above Base	US$/t/12 m bench	0.01	0.01	0.03	0.03
Incremental Cost below Base	US$/t/12 m bench	0.02	0.02	0.03	0.03
Processing Cost	US$/t feed	6.32	6.32	7.55	7.55
General and Administrative Cost	US$/t feed	0.25	0.25	0.22	0.22
Geotechnical					
Overall Wall Slope Angle	degrees	45	45	45	45
Green positive change Red negative change Black unchanged from 2019					

The net effect of changes to the input parameters has resulted in the Lerchs-Grossmann mineral resource constraining pit shell shrinking in volume for both the Rovina and Colnic deposits. Table 14.25 summarises the mineral resource estimates for the Rovina and Colnic deposits, stated above a 0.25 % Cu equivalent grade cut-off for the Rovina deposit, and above a 0.35 g/t Au equivalent grade cut-off for the Colnic deposit. The total Measured mineral resources for the Rovina and Colnic deposits amount to 62.2 Mt grading at 0.49 g/t Au and 0.21 % Cu, containing 0.99 Moz Au and 287 Mlb Cu; with the Au equivalent grading of 0.79 g/t. The total Indicated mineral resources for the Rovina and Colnic deposits amount to an additional 175.6 Mt grading at 0.39 g/t Au and 0.15 % Cu, containing 2.19 Moz Au and 589 Mlb Cu; with the Au equivalent grading of 0.60 g/t.

Table 14.25: 2021 Mineral Resource Estimate – Rovina and Colnic Deposits

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Colnic	Measured	29.1	0.65	0.12	0.61	74	0.81	0.76
	Indicated	97.5	0.49	0.10	1.53	210	0.62	1.96
	Inferred	1.6	0.41	0.09	0.02	3	0.49	0.03
Rovina	Measured	33.1	0.36	0.29	0.38	212	0.77	0.82
	Indicated	78.1	0.26	0.22	0.66	379	0.57	1.44
	Inferred	16.0	0.18	0.19	0.09	66	0.44	0.23
Total	Measured	62.2	0.49	0.21	0.99	287	0.79	1.58
	Indicated	175.6	0.39	0.15	2.19	589	0.60	3.40
	Inferred	17.6	0.20	0.18	0.11	69	0.45	0.26
Grand Total	Measured and Indicated	237.7	0.42	0.17	3.18	875	0.65	4.97

NOTES:

1. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are contained within conceptual pit shells that are generated using the same economic and technical parameters used for Mineral Reserves but at a gold price of US$1,700/oz and a copper price of US$3.50/lb.
3. The Colnic and Rovina deposits are amenable to open-pit mining and Mineral Resources are pit constrained and tabulated at a base case cut-off grade of 0.35 g/t AuEq for Colnic and 0.25 % CuEq for Rovina.
4. Minor summation differences may occur as a result of rounding.

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,700/oz and a copper price of US$3.50/lb with metallurgical recoveries not taken into account.

The Ciresata underground mineral resource estimate remains unchanged from the 20 February 2019 estimate by AGP. Table 14.26 summarises the mineral resource estimate for Ciresata, stated at above a 0.65 g/t Au equivalent grade cut-off. The Measured mineral resources amount to 28.5 Mt grading at 0.88 g/t Au and 0.16 % Cu, containing 0.81 Moz Au and 102 Mlb Cu, with the Au equivalent grading of 1.13 g/t. The Indicated mineral resources amount to an additional 125.9 Mt grading at 0.74 g/t Au and 0.15 % Cu, containing 3.01 Moz Au and 413 Mlb Cu, with the Au equivalent grading of 0.97 g/t.

Table 14.26: 2019 Mineral Resource Estimate – Ciresata Deposit

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Ciresata	Measured	28.5	0.88	0.16	0.81	102	1.13	1.03
	Indicated	125.9	0.74	0.15	3.01	413	0.97	3.92
	Inferred	8.6	0.70	0.14	0.19	26	0.94	0.25
Total	**Measured & Indicated**	**154.4**	**0.77**	**0.15**	**3.82**	**515**	**1.00**	**4.95**

NOTES:
1. The Ciresata deposit is amenable to bulk underground mining and resources are tabulated at a base case 0.65 g/t AuEq.
2. No Mineral Reserves have been defined at the Ciresata deposit. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Minor summation differences may occur as a result of rounding.

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,500/oz and a copper price of US$3.50/lb.

Source: From Table 14-20, AGP PEA NI 43-101 2019 Report (available on SEDAR)

It must be noted that the quantity and grade of Inferred resource reported above are conceptual in nature and are estimated based on limited geological evidence and sampling. Geological evidence is sufficient to imply, but not verify, geological and grade or quality continuity. For these reasons, an Inferred mineral resource has a lower level of confidence than an Indicated mineral resource, and it is reasonably expected that the majority of Inferred mineral resources could be upgraded to an Indicated mineral resource with continued exploration. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Rounding of tonnes as required by reporting guidelines may result in apparent differences between tonnes, grade, and contained metal content.

14.18.1 Reconciliation with Previous Mineral Resource Estimate

The 2019 mineral resource estimates for the Rovina and Colnic deposits are summarised in Table 14.27, with a summary of the input parameters for the Lerchs-Grossmann mineral resource constraining pit shell for 2019 and 2021 contained in Table 14.24. Changes in the Lerchs-Grossmann input parameters resulted in a shrinkage of the mineral resource constraining shell and, therefore, a reduction in the overall mineral resource estimates for the Rovina and Colnic deposits. The total Measured mineral resource tonnage increased by 1.4 %, with the Au and Cu grades remaining the same. This minor increase in tonnage is thought to be because of the increased metal prices, pushing some blocks above the Au equivalent grade cut-off. The total Indicated mineral resource tonnage decreased by 2.8 %, from 180.7 Mt to 175.6 Mt, with the Au and Cu grades remaining the same. This decrease in tonnage is thought to be a result of the shrinkage in the mineral resource constraining shells. The total Inferred mineral resource tonnage decreased by 10.4 %, from 19.6 Mt to 17.6 Mt, with the Au and Cu grades remaining the same.

Table 14.27: 2019 Mineral Resource Estimate – Rovina and Colnic Deposits

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Colnic	Measured	29.2	0.65	0.12	0.61	74	0.82	0.77
	Indicated	106.5	0.47	0.10	1.62	228	0.62	2.12
	Inferred	4.7	0.34	0.10	0.05	10	0.46	0.07
Rovina	Measured	32.1	0.36	0.29	0.37	208	0.80	0.83
	Indicated	74.2	0.27	0.22	0.64	365	0.60	1.44
	Inferred	14.9	0.19	0.19	0.09	62	0.46	0.22
Total	Measured	61.3	0.50	0.21	0.98	282	0.81	1.60
	Indicated	180.7	0.39	0.15	2.26	593	0.61	3.56
	Inferred	19.6	0.22	0.17	0.14	72	0.46	0.29
Grand Total	Measured & Indicated	242.0	0.42	0.16	3.24	875	0.66	5.16

NOTES:

1. Mineral Resources are contained within conceptual pit shells that are generated using the same economic and technical parameters as used for Mineral Reserves but at gold price of US$1,500/oz and a copper price of US$3.30/lb.
2. The Colnic and Rovina deposits are amenable to open-pit mining and Mineral Resources are pit constrained and tabulated at a base case cut-off grade of 0.35 g/t AuEq for Colnic and 0.25 % CuEq for Rovina.
3. *Minor summation differences may occur as a result of rounding.*

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,500/oz and a copper price of US$3.30/lb with metallurgical recoveries not taken into account.

Source: From Table 14-20, AGP PEA NI 43-101 2019 Report (available on SEDAR)

15 MINERAL RESERVE ESTIMATES

The process to develop the Mineral Reserve estimate for the RVP was as follows:

- The open-pit optimisation has been undertaken on the Measured and Indicated Resources only.
- The geological losses in the block model allow for a 2.5 % loss on Measured resources and a 3.5 % on Indicated resources.
- The grades and tonnes of the mineral resource model have been modified by mining/geological recovery and mining dilution based on orebody geometry and mining methodology. The mining model contains undiluted ore tonnes and ore grade. Owing to the massive nature of both the Colnic and Rovina orebodies, fixed dilution and recovery percentages of 2 % and 97.5 %, respectively, were applied in the Whittle optimisations.
- The Whittle suite of optimisation software was used to perform the pit optimisations. Whittle is an accepted industry optimisation tool. A range of operating costs and production parameters were applied. The source of the parameters is summarised below, along with the source of the information:
    - A base gold price of US$1,500/oz (Sidus Consulting) with a government royalty of 6.0 % of the revenue. This resulted in a net gold price of ~US$1,382.50/oz at the Colnic pit and ~$1,277.88/oz at the Rovina pit. The difference is due to the net smelter return (NSR) calculations, which differ for each area. (Ref: P20_015 Rovina Valley Market (RVM) report. V001)
    - A base copper price of US$3.00/lb (Sidus Consulting), with a government royalty of 5.0 % of the revenue. This resulted in a net copper price of ~US$5,089.83/t for both pits.
    - Pit slope inter-ramp angles ranging from 44.0° to 49.9°, depending on the pit slope geometry. The resulting overall pit slope angles account for access ramps where applicable. (KCB)
    - Gold recovery ranging from 77.6 % to 85.5 %, depending on the mining area and the ore type being processed. (Metallurgical test work)
    - Processing throughput of 7.2 Mt/a. (RVM)
    - The Owner's mining fleet capital and operating costs are based on budget pricing submissions from various OEMs, and all costs have been converted to United States dollars.
    - An average annual processing cost per tonne of ore, inclusive of general and administration costs and overland haulage. (RVM)

A sensitivity assessment was done on gold prices of US$1,400/oz and US$1,700/oz. A scenario assessment with a gold price of US$2,500/oz was also done to determine the surface infrastructure boundaries required to ensure that no potential future resource is sterilised. This indicated that the optimal shell inventory (i.e. the size and shape of the optimal shell and, therefore, the ore and waste generated) was robust for all mining areas.

Optimal shells were selected for each deposit, and these were then used as the basis for the ultimate pit designs. The shell selection criteria were conservative and were based on a gold price of US$1,500/oz and a copper price of US$3.00/lb (see Appendix 1 of the RVM Report. for more details)

Various cut-off grades were applied, based on projected incremental revenue per tonne and high-grade and low-grade stockpiles were generated based on a two-stage pit design with pushbacks to ensure that optimal revenue is delivered in the early life of mine (LOM).

Table 15.1 summarises the Mineral Reserve Statement based on the work detailed above, undertaken as part of the RVP.

Table 15.1: RVP Mineral Reserve Statement Summary

Deposit	Classification	Tonnage (Mt)	Au Grade (g/t)	Cu Grade (%)	Au (koz)	Cu (t)
Colnic	Proven	24.27	0.64	0.11	500.5	26,860.9
	Probable	49.49	0.52	0.08	828.7	41,004.7
Rovina	Proven	24.01	0.32	0.28	247.8	67,469.3
	Probable	35.62	0.22	0.20	249.5	72,896.1
Total	Proven	48.28	0.48	0.20	748.3	94,330.2
	Probable	85.11	0.39	0.13	1,078.2	113,900.8
Grand Total	**Proven + Probable.**	**133.40**	**0.43**	**0.16**	**1,826.5**	**208,231.0**
NOTE: All tonnes quoted are dry tonnes. Differences in the addition of deposit tonnes to the total displayed is due to rounding.						

The Mineral Reserve estimate has been classified and reported in accordance with the Canadian National Instrument 43-101, "Standards of Disclosure for Mineral Projects" of June 2011 (NI 43-101), and the classifications adopted by the CIM Council in November 2010. Furthermore, the Mineral Reserve classifications are also consistent with the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" of 2012 (JORC Code) as prepared by the Australasian Joint Ore Reserves Committee comprising representatives from the Australasian Institute of Mining and Metallurgy, the Australian Institute of Geoscientists, and the Minerals Council of Australia, with the minor exception that the JORC Code refers to Ore Reserves while NI 43-101 refers to Mineral Reserves.

The RVP Mineral Reserve estimate is not at this stage materially affected by any known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant issue. Furthermore, the estimate of Project Reserves is not materially affected by any known mining, metallurgical, infrastructure, or other relevant factor.

DRA is confident that sufficient geological work has been undertaken, and sufficient geological understanding gained, to enable the construction of an orebody model suitable for the derivation of Mineral Resource and Mineral Reserve estimates. DRA considers that both the modelling and the grade interpolation have been carried out in an unbiased manner and that the resulting grade and tonnage estimates should be reliable within the context of the classification applied. In addition, DRA is not aware of any metallurgical, infrastructural, environmental, legal, title, taxation, socio-economic, or marketing issues that would impact on the Mineral Resource or Mineral Reserve statements as presented.

16 MINING METHODS, MINING EQUIPMENT AND INFRASTRUCTURE

16.1 INTRODUCTION

This section presents the main conclusions given in the mining methods selection and their application to the Rovina Valley DFS.

A LOM schedule has been developed to supply one processing plant for the full LOM. The processing plant has a planned throughput of 7.2 Mt/a (Colnic pit and then Rovina pit) and a LOM of 16 years, excluding the Colnic low-grade stockpile. The LOM schedule considers the in-pit and stockpile blending requirements during the life of each pit, as well as the changeover from Colnic to Rovina ore supply to the processing plant.

The Owner’s mining fleet will be responsible for all mining-related earthmoving activities.

All deposits will be mined utilising conventional truck and shovel methods to supply ore to the ROM tip and waste to the respective pit’s waste crushing and conveying station, which will transport the waste to the Colnic stockpile area and backfill the Colnic pit.

Most of the ore and waste materials will be drilled and blasted as there is a nominal amount of free dig/oxidised materials.

Free-dig and blasted waste will be loaded with 200 t class hydraulic backhoe excavators, hauled with 90 t haul trucks, and stockpiled at designated waste stockpile locations, which will be systematically dozed and levelled to allow the stockpiles to be raised in accordance with the design parameters.

Free-dig and blasted ore will be loaded with 200 t class hydraulic backhoe excavators and hauled with 90 t haul trucks to the plant feed ROM pad. There the ore will either be direct tipped into the crushing facility or placed on the ROM pad stockpile areas, depending on the grade control strategy being applied.

The project is to be mined utilising proven drilling, blasting and earthmoving equipment operated by an experienced Owner’s team and well-trained workforce.

Owing to the significantly higher elevation and limited capacity of the Colnic waste storage facility, a waste-only crushing and conveying system will be installed for waste transportation only during the operational life of the Colnic pit. This crushing and conveying system will be reclaimed at the end of the Colnic pit’s life and reused for both waste and ore batched transportation to the Colnic backfill site and the Colnic ROM pad position from Year 9 till the end of the Rovina pit life.

The Colnic pit will commence production first as it has the best incremental value per tonne and as such will assist in delivering higher mill feed grades early in the project life. The option of mining the Colnic and Rovina pits simultaneously was investigated but rejected due to the following rationale:

- The Colnic waste crushing and conveying cost savings would not be fully realised.
- All the waste and ore from Rovina would not have access to the crushing and conveying infrastructure, so mining costs and fleet requirements would be significantly higher due to the longer haul distances.

- The Colnic waste facility would be filled in the first 10 to 12 years and then high-value ore in either the Colnic pit or Rovina pit would be lost when backfilling of either pit became necessary.
- The simultaneous mining option would have a negative impact on the total ore reserves that could be mined.

Approximately six to seven months of waste stripping will be required to expose sufficient ore to maintain a constant ore feed of 7.2 Mt/a post commissioning and the planned process plant feed build-up.

The mining of the two deposits runs for a period of approximately 16 years based on the current production schedule and design parameters.

The peak production requirements of the total mining fleets have been capped at an estimated 27.4 Mt/a (total material movement).

16.2 PROJECT DESIGN AND OPERATION

16.2.1 Optimisation Parameters

The optimisation parameters used are as follows:

- Mining costs
- Slope design
- Processing recoveries and costs
- Financial parameters

16.2.1.1 Mining Costs

Mining costs are based on the October 2020 OEM budget price estimates. The cost estimates comprise the following elements:

- Transport, on-site assembly, commissioning, and training of operators
- OEM training on maintenance of all necessary equipment
- Construction and commissioning planning of the works
- Clearing and grubbing, topsoil stripping and stockpiling
- Drilling and blasting of all relevant ore and waste material, including pre-splitting.
- Excavating and loading of all materials
- Hauling, tipping, and stockpiling of all materials to the designated destinations
- Construction, or reconstruction and maintenance of all necessary haul roads
- Grade control drilling and sampling from dedicated drill rigs as per bill of quantities (BOQ)
- Rehandling of the ROM stockpile to the ROM tip as per the plant feed schedule
- Provision and control of surface drainage
- Management and removal of all water within the open-pit area and associated surface activities, including removal of storm water and ground water
- Provision of all pit and stockpile lighting facilities if required
- Profiling of final stockpiles and other disturbed areas as directed by the superintendent.

- Carrying out secondary breaking of ore as required
- Provision and management of all personnel for the mining activities
- Provision of safety, environmental and quality assurance plans
- Strict safety management and reporting of progress of the works

16.2.1.2 Bench Slope Design Parameters

The preliminary pit slope design parameters were provided by KCB. The slopes were provided based on the weathering codes within the block model (i.e. oxide/transition/fresh). The preliminary design parameters for both the Rovina and Colnic Pits are detailed in Table 16.1.

Table 16.1: Colnic and Rovina Slope Design Parameters

Area	Bench Face Angle (BFA)	Bench Height	Berm Width	Maximum Inter-Ramp Height	Maximum Height Width Geotechnical Berm if No Ramp	Geotechnical Berm Width	Ramp Width	Inter-Ramp Angle
	degrees	m	m	m	m	m	m	degrees
Controlled Blasting Area	65	24	9	120	120	25	28	49.9
Overburden	33	3	9					

It is important to note that these preliminary slope angle design parameters may be subject to changes based on the 2020 geotechnical and geohydrological studies currently being undertaken by KCB.

KCB Final Bench Face Angles

The Colnic pit optimisation, design and production schedule used a preliminary BFA of 65°. Post the updated geotechnical investigation, the rock mass strengths showed significant improvement, and an improvement in the BFA has been recommended for the Colnic Pit.

The final summary results for most of the Colnic pit BFA increased from 65° to 70°. This is for all areas except C1 and C2 (see Figure 16.1).

The Rovina pit design BFA parameters did not change.

For more details, refer to the following:

- Klohn Crippen Berger, 2021. Rovina Valley Project – Feasiblity Study – Geotechnical, Hydrotechnical, and Hydrogeological Report.

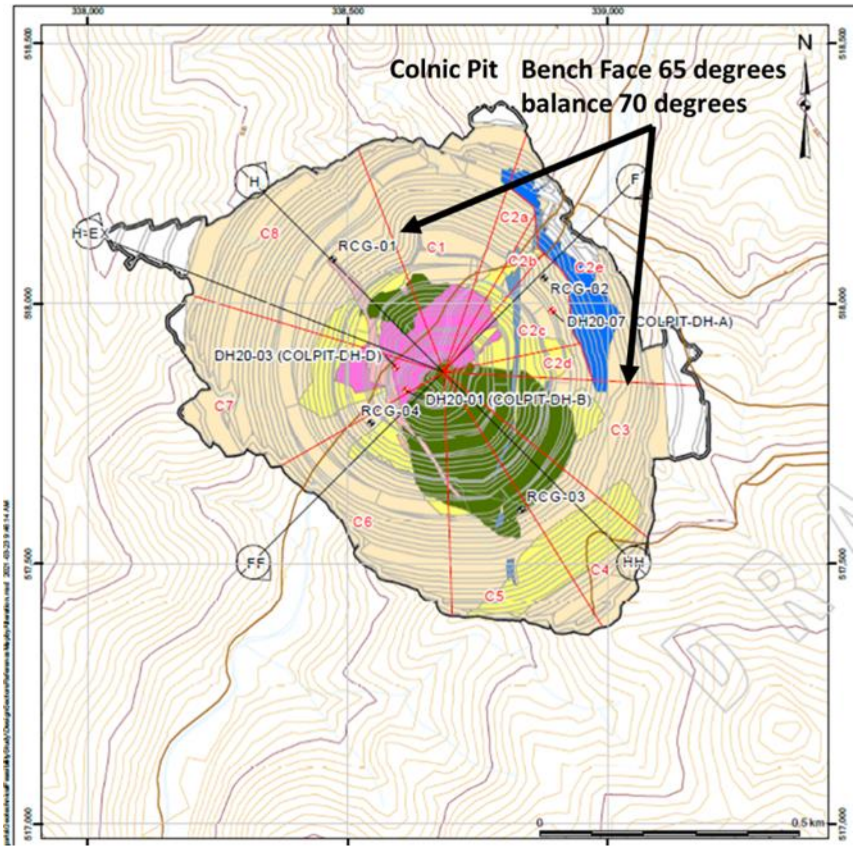


Figure 16.1: Colnic Pit BFA Outcome Summary

A re-optimisation, redesign and reschedule will improve the project economics by reducing the strip ratio and lowering the total mining cost in the early RVP LOM.

16.2.1.3 Processing Recoveries and Costs

Processing recoveries and costs for each metallurgical domain were received from DRA and modelled in 3D using the ESM geological model for each of the domains of the open-pit deposits.

16.2.1.4 Financial Parameters

The gold price and discount rate used in the optimisations are summarised in Table 16.2.

Table 16.2: Optimisation Financial Parameters

Parameter	Unit	Colnic	Rovina
Discount Rate	%	10	10
Base Price:			
Gold	US$/oz	1,500	1,500
Copper	US$/lb	3.00	3.00
Government Royalty:			
Gold	%	6.0	6.0

Parameter	Unit	Colnic	Rovina
Copper	%	5.0	5.0
Processing Cost	US$/t ROM	9.77	7.53
Net Price:			
Gold	US$/g	44.448	41.085
Copper	US$/t	5,089.83	5,089.83

16.2.2 Bench Height Selection

16.2.2.1 Motivation for Bench Height Analysis

The geological resource block model was generated on a parent block cell size of 10 m × 10 m × 12 m (X, Y and Z, respectively) for both the Rovina and Colnic pits.

16.2.2.2 Ore Dilution and Ore Loss

A 2 % dilution and 2.5 % loss were assumed because the ore domains are continuous and will be clearly delineated and marked. Sampling of blast holes would be the basis for grade control in this analysis. The accuracy of the resulting ore/waste boundary is limited by the resolution of the grade control, which is a function of the density of the drilling pattern. The lower the flitch height, the smaller the pattern, the smaller the distance between "ore holes" and "waste holes" and hence the smaller the potential for ore loss and/or ore dilution. These dilution and loss percentages are accepted as being in line with smaller flitch heights such as the 6 m flitches associated with this mining operation.

16.2.3 Optimisation Sensitivities

To gauge the project sensitivity to the gold price fluctuations, three scenarios were run at US$1,400/oz, US$1,700/oz and US$2,000/oz in addition to the base case scenario at US$1,500/oz.

The influence of these variations was tested on the following key project performance indicators:

- Indicative project discounted cash flow (specified case)
- Gold metal recovered
- Copper metal recovered
- Project life in years
- Ore tonnes mined

The resulting key performance indicator variance is indicated in percentage changes from the base case. The resulting project outcomes are indicated in Figure 16.2 to Figure 16.6.

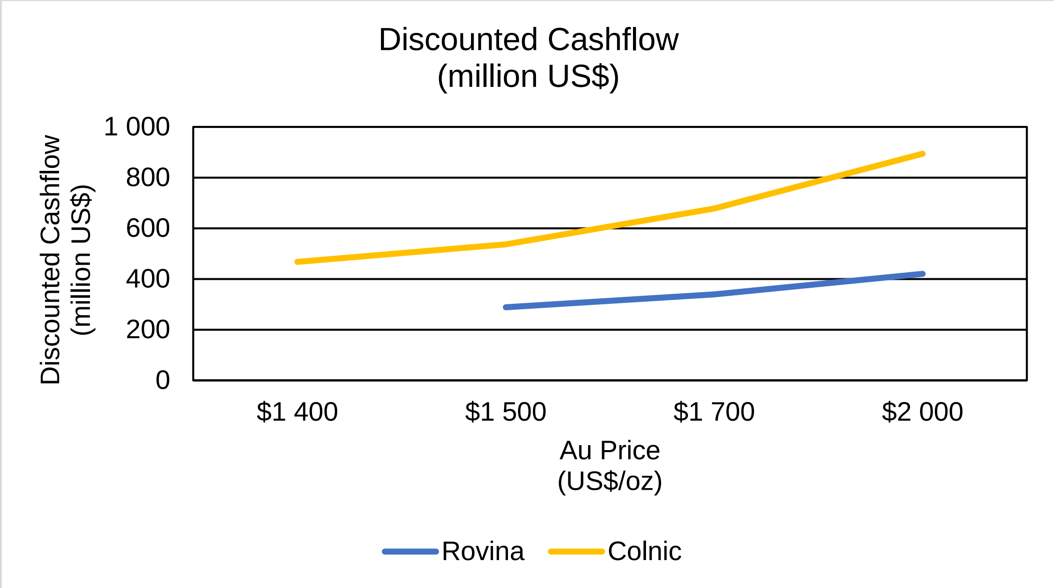


Figure 16.2: Discounted Cash Flow Sensitivities

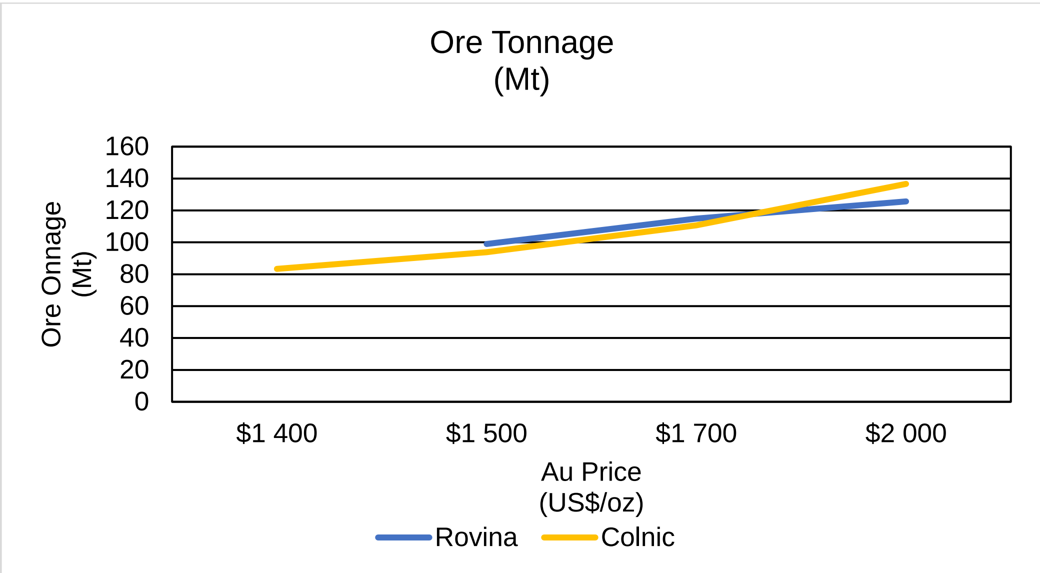


Figure 16.3: Ore Tonnes

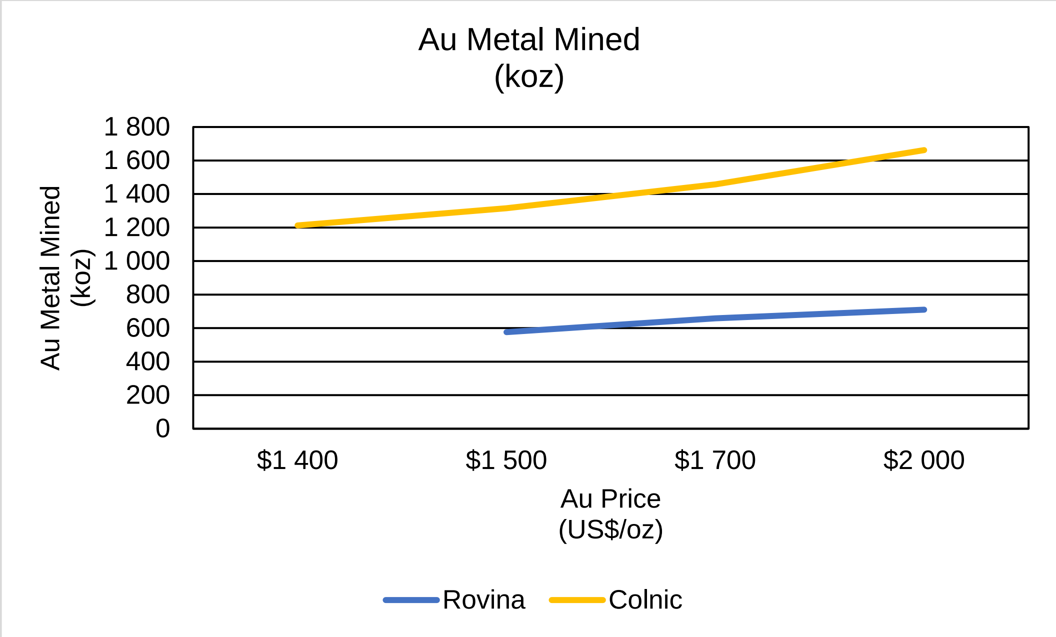


Figure 16.4: Mining Au Metal Output

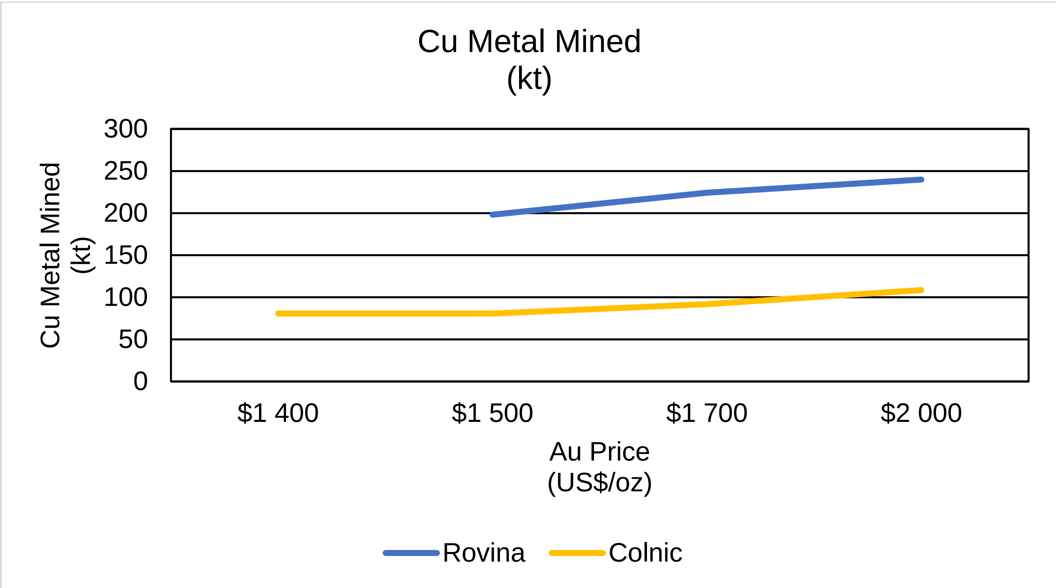


Figure 16.5: Mining Cu Metal Output

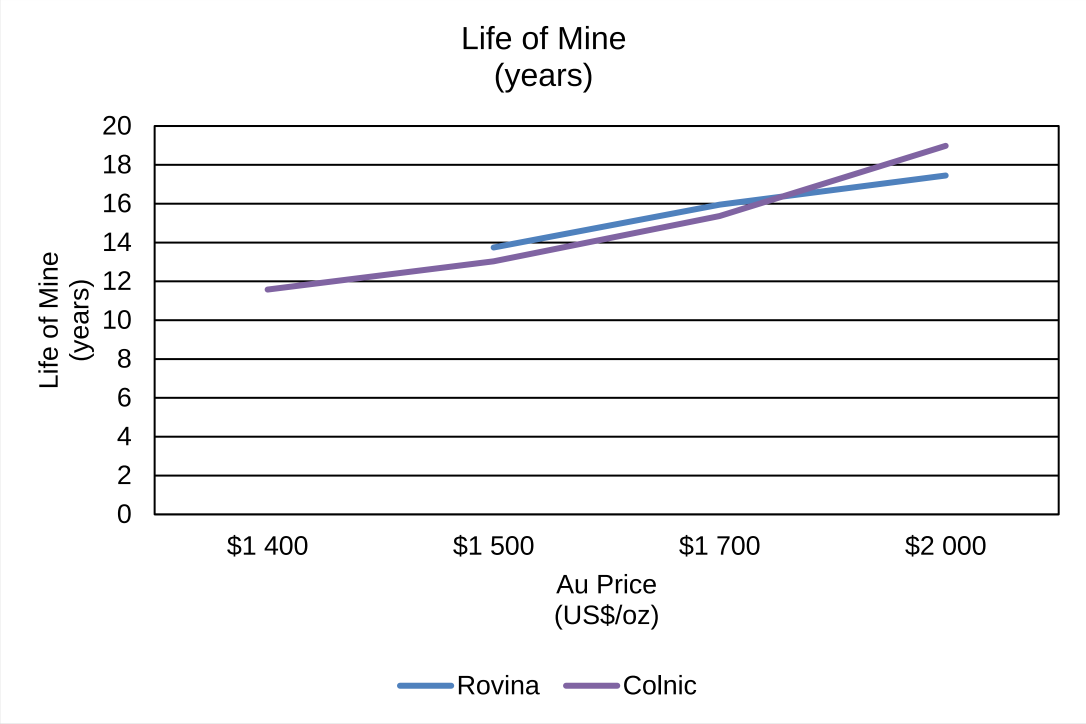


Figure 16.6: Project Life

Predictably, the project value indicates linear sensitivity behaviour as a result of commodity price variation. The metal mined, project life and ore tonnes mined sensitivities indicate that the resource is relatively robust and insensitive to commodity prices.

The above analyses confirmed the robustness of the optimal shells. Predictably, all discounted cash flow (DCF) sensitivities to gold price fluctuations exhibit similar linear behaviour.

16.2.4 Final Shell Selection

An indicative profit value was calculated from the Whittle optimisation results using the revenue and various total costs determined by Whittle for each pit shell.

The pit shell with the largest indicative incremental profit per tonne was selected as the preferred option for both mining areas depending on their waste/co-disposal facility capacities.

16.2.4.1 Colnic Shell Selection Run 29

By selecting Shell 26 in run 29, the total waste mined was limited to 108.6 Mt and ore to 73.2 Mt at an average gold grade of 0.57 g/t and an average copper grade of 0.09 %. This is necessary due to the limited capacity of the co-disposal storage facility, which at this stage was estimated at approximately 180 Mt.

Table 16.3: Colnic Optimisation Results

Pit Shell No.	DCF (Specified) US$ million	Indicative Profit US$ million	Waste Mt	Ore Mt	Au Mined t	Au Grade g/t	Cu Mined t	Cu Grade %
1	21.54	21.76	0.06	0.76	0.72	0.95	1,175	0.16
2	36.23	36.89	0.09	1.37	1.25	0.91	1,979	0.14
3	52.37	53.90	0.24	2.17	1.87	0.86	3,023	0.14
4	75.55	79.14	0.60	3.51	2.82	0.80	4,721	0.13
5	92.23	97.94	1.09	4.54	3.55	0.78	5,984	0.13
6	120.53	131.42	2.29	6.53	4.91	0.75	8,235	0.13
7	147.62	162.85	3.77	8.56	6.26	0.73	10,215	0.12
8	181.95	203.74	6.21	11.37	8.09	0.71	12,798	0.11
9	269.01	318.72	13.94	19.71	13.36	0.68	20,253	0.10
10	292.53	352.69	16.99	22.14	14.94	0.67	22,749	0.10
11	316.46	386.38	20.27	24.76	16.57	0.67	25,185	0.10
12	340.89	423.66	24.39	27.83	18.42	0.66	28,095	0.10
13	367.73	465.60	29.42	31.60	20.56	0.65	31,382	0.10
14	380.90	487.14	32.88	33.42	21.68	0.65	33,304	0.10
15	397.33	515.85	37.22	36.22	23.23	0.64	35,920	0.10
16	415.73	547.03	42.70	39.39	24.98	0.63	38,820	0.10
17	434.58	582.27	49.90	43.31	27.08	0.63	42,468	0.10
18	447.41	606.07	54.61	46.48	28.56	0.61	44,976	0.10
19	491.81	704.71	78.39	59.46	34.93	0.59	56,601	0.10
20	498.41	720.56	82.51	61.72	35.99	0.58	58,563	0.09
21	503.42	733.31	86.47	63.59	36.93	0.58	60,323	0.09
22	510.55	751.54	92.76	66.50	38.34	0.58	62,989	0.09
23	512.77	757.27	94.93	67.49	38.81	0.58	63,923	0.09
24	517.54	770.19	100.70	69.91	39.96	0.57	66,197	0.09
25	521.28	781.01	106.09	72.15	41.01	0.57	68,268	0.09
26	523.03	785.64	108.63	73.24	41.48	0.57	69,295	0.09
27	527.60	798.53	117.55	76.55	43.01	0.56	72,413	0.09
28	529.00	802.73	120.74	77.70	43.54	0.56	73,522	0.09
29	530.66	807.94	124.81	79.46	44.25	0.56	75,093	0.09
30	531.89	811.60	128.65	80.87	44.85	0.55	76,410	0.09
31	533.19	815.59	133.79	82.45	45.59	0.55	77,892	0.09
32	534.15	818.81	139.06	84.44	46.37	0.55	79,742	0.09
33	535.10	822.16	146.93	86.78	47.40	0.55	81,904	0.09
34	535.71	824.21	152.36	88.84	48.18	0.54	83,742	0.09
35	535.83	824.67	154.49	89.54	48.46	0.54	84,377	0.09
36	535.98	825.51	167.23	93.77	50.08	0.53	88,181	0.09
37	535.92	825.31	169.17	94.51	50.34	0.53	88,835	0.09
38	535.59	824.15	176.90	96.53	51.20	0.53	90,630	0.09
39	535.04	822.13	183.31	98.86	51.98	0.53	92,718	0.09
40	534.20	818.91	191.76	101.31	52.90	0.52	94,880	0.09
41	532.50	812.32	204.55	105.34	54.32	0.52	98,369	0.09
42	531.67	808.83	210.57	106.89	54.91	0.51	99,820	0.09
43	530.74	804.90	216.54	108.49	55.49	0.51	101,275	0.09

Pit Shell No.	DCF (Specified)	Indicative Profit	Waste	Ore	Au Mined	Au Grade	Cu Mined	Cu Grade
	US$ million	US$ million	Mt	Mt	t	g/t	t	%
44	528.54	795.40	228.92	112.06	56.72	0.51	104,423	0.09
45	526.06	783.67	243.16	115.63	58.01	0.50	107,684	0.09
46	525.48	780.97	245.92	116.49	58.28	0.50	108,470	0.09
47	522.84	768.07	259.23	120.10	59.49	0.50	111,684	0.09
48	521.22	759.48	267.48	122.21	60.21	0.49	113,480	0.09
49	518.03	743.64	282.01	125.76	61.41	0.49	116,636	0.09
50	515.75	731.55	292.94	128.16	62.28	0.49	118,644	0.09
51	514.70	725.66	297.37	129.38	62.65	0.48	119,786	0.09
52	510.56	703.13	315.67	133.37	64.02	0.48	123,312	0.09
53	508.51	691.10	325.15	135.39	64.73	0.48	125,100	0.09
54	507.79	686.81	328.24	136.10	64.95	0.48	125,688	0.09
55	506.51	678.94	334.34	137.23	65.38	0.48	126,636	0.09
56	502.72	654.61	351.57	140.66	66.56	0.47	129,672	0.09
57	502.69	654.43	351.66	140.68	66.57	0.47	129,689	0.09
58	500.95	642.53	359.01	142.30	67.08	0.47	131,063	0.09
59	498.23	623.42	372.15	144.56	67.88	0.47	133,054	0.09
60	495.47	604.21	384.59	146.83	68.64	0.47	135,056	0.09
61	495.32	603.06	385.04	146.95	68.67	0.47	135,167	0.09
62	492.86	585.03	396.61	148.86	69.35	0.47	136,803	0.09
63	491.98	578.27	400.45	149.56	69.56	0.47	137,486	0.09
64	489.30	557.51	413.03	151.61	70.27	0.46	139,198	0.09
65	488.00	547.76	418.63	152.58	70.58	0.46	140,063	0.09
66	486.53	536.40	425.79	153.52	70.94	0.46	140,932	0.09
67	483.86	514.97	437.36	155.44	71.53	0.46	142,627	0.09
68	483.69	513.58	437.89	155.55	71.56	0.46	142,728	0.09
69	481.47	495.00	448.71	157.06	72.10	0.46	144,034	0.09
70	480.74	488.71	452.26	157.54	72.27	0.46	144,495	0.09
71	480.20	483.99	454.72	157.92	72.39	0.46	144,748	0.09
72	479.81	480.62	456.49	158.17	72.47	0.46	144,978	0.09
73	477.06	458.02	468.60	159.74	73.01	0.46	146,476	0.09
74	476.43	452.83	471.25	160.12	73.13	0.46	146,798	0.09
75	475.58	445.64	475.10	160.56	73.28	0.46	147,186	0.09
76	474.31	434.81	481.12	161.25	73.54	0.46	147,742	0.09
77	473.73	429.79	483.70	161.58	73.65	0.46	147,998	0.09
78	471.62	411.27	493.36	162.75	74.05	0.46	148,943	0.09
79	470.97	405.48	495.76	163.12	74.15	0.45	149,256	0.09
80	469.79	394.66	501.14	163.79	74.36	0.45	149,895	0.09
81	468.85	386.02	505.66	164.30	74.54	0.45	150,288	0.09
82	468.17	379.75	508.69	164.65	74.65	0.45	150,554	0.09
83	467.14	370.01	513.61	165.18	74.83	0.45	151,004	0.09
84	465.53	355.13	520.68	166.01	75.09	0.45	151,702	0.09
85	465.18	351.96	522.10	166.19	75.14	0.45	151,810	0.09
86	464.32	344.15	525.37	166.64	75.26	0.45	152,150	0.09
	Optimum Specified Case							
	Selected Shell							

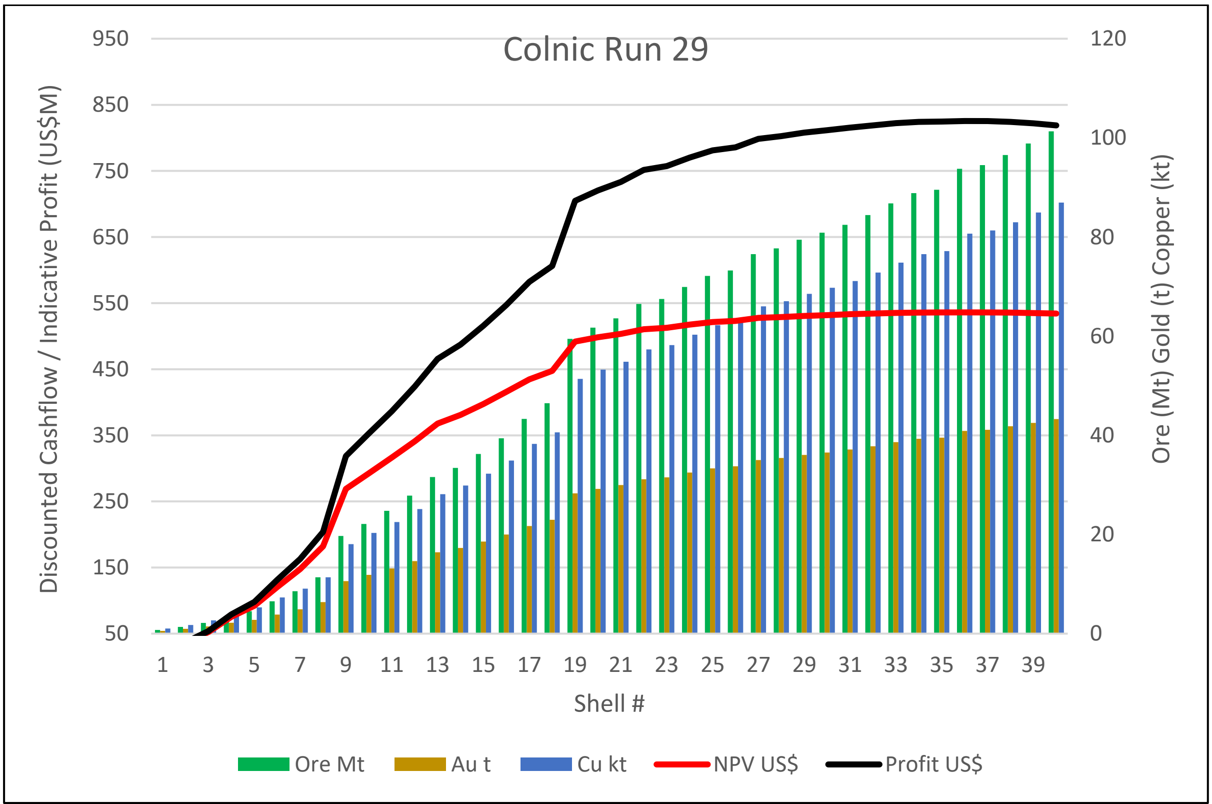


Figure 16.7: Colnic Whittle Analysis

16.2.4.2 Rovina Shell Selection Run 8

By selecting Shell 32 of run 8, the total waste mined was limited to 125.9 Mt and ore to 64.3 Mt at an average gold grade of 0.25 g/t and an average copper grade of 0.22 %. This is necessary due to the limited capacity of the Colnic pit backfill co-disposal storage facility, which at this stage was estimated at approximately 190 Mt.

Table 16.4: Rovina Optimisation Results

Pit Shell No.	DCF (Specified)	Indicative Profit	Waste	Ore	Au Mined	Au Grade	Cu Mined	Cu Grade
	US$ million	US$ million	Mt	Mt	t	g/t	t	%
1	25.47	25.93	0.09	1.34	0.54	0.40	4,549	0.34
2	37.41	38.45	0.24	2.08	0.81	0.39	6,914	0.33
3	49.70	51.65	0.50	2.90	1.10	0.38	9,473	0.33
4	60.56	63.56	0.91	3.66	1.37	0.37	11,835	0.32
5	73.72	78.47	1.34	4.72	1.70	0.36	14,967	0.32
6	84.27	90.84	1.88	5.67	2.00	0.35	17,628	0.31
7	95.49	104.47	2.53	6.79	2.33	0.34	20,727	0.31
8	107.83	118.77	3.62	8.03	2.72	0.34	23,942	0.30

Pit Shell No.	DCF (Specified)	Indicative Profit	Waste	Ore	Au Mined	Au Grade	Cu Mined	Cu Grade
9	119.28	131.96	4.38	9.32	3.06	0.33	27,235	0.29
10	133.61	149.23	5.57	11.12	3.52	0.32	31,703	0.29
11	148.21	167.75	7.55	13.07	4.07	0.31	36,437	0.28
12	175.03	202.55	11.48	17.02	5.10	0.30	45,895	0.27
13	185.02	216.07	13.45	18.59	5.50	0.30	49,813	0.27
14	195.06	230.44	15.69	20.42	5.97	0.29	54,034	0.26
15	205.19	245.05	18.30	22.37	6.44	0.29	58,632	0.26
16	208.36	249.45	19.15	22.98	6.58	0.29	60,117	0.26
17	219.47	265.55	22.98	25.38	7.18	0.28	65,495	0.26
18	221.58	268.75	23.86	25.88	7.29	0.28	66,677	0.26
19	228.79	280.11	27.31	27.81	7.76	0.28	70,891	0.25
20	232.11	285.57	29.29	28.76	7.99	0.28	73,028	0.25
21	239.44	296.75	33.31	30.90	8.50	0.27	77,553	0.25
22	244.21	304.37	36.83	32.43	8.84	0.27	81,028	0.25
23	252.84	319.31	43.87	35.88	9.67	0.27	87,838	0.24
24	258.75	329.39	49.83	38.42	10.26	0.27	92,993	0.24
25	262.23	335.72	53.69	40.24	10.69	0.27	96,416	0.24
26	266.06	343.11	59.38	42.43	11.17	0.26	100,967	0.24
27	270.23	351.23	66.32	45.15	11.79	0.26	106,316	0.24
28	275.05	361.27	76.49	48.88	12.64	0.26	113,628	0.23
29	278.62	369.21	86.62	52.16	13.41	0.26	120,161	0.23
30	281.26	375.46	96.17	55.09	14.10	0.26	126,029	0.23
31	284.12	383.07	110.68	59.59	15.15	0.25	134,723	0.23
32	286.36	389.79	125.92	64.29	16.22	0.25	143,800	0.22
33	287.22	417.97	202.56	87.97	21.82	0.25	188,029	0.21
34	288.08	423.06	226.21	94.66	23.43	0.25	200,895	0.21
35	288.42	425.13	242.52	98.96	24.48	0.25	209,224	0.21
36	288.41	425.83	256.81	102.39	25.33	0.25	216,209	0.21
37	287.97	424.93	274.82	106.43	26.34	0.25	224,407	0.21
38	287.54	423.51	284.51	108.56	26.87	0.25	228,689	0.21
39	286.94	421.14	297.86	111.02	27.53	0.25	233,972	0.21
40	286.85	420.77	299.42	111.30	27.60	0.25	234,570	0.21
41	286.19	417.93	308.34	113.05	28.05	0.25	238,004	0.21
42	285.27	413.71	319.94	115.30	28.61	0.25	242,552	0.21
43	284.67	410.97	326.49	116.38	28.90	0.25	244,881	0.21
44	283.69	406.24	337.05	118.25	29.38	0.25	248,512	0.21
45	282.95	402.56	344.19	119.49	29.68	0.25	251,047	0.21
46	282.01	397.71	352.65	121.01	30.06	0.25	254,169	0.21
47	280.50	389.90	365.84	123.23	30.60	0.25	258,564	0.21
48	279.13	382.99	376.41	124.98	31.04	0.25	262,055	0.21

Pit Shell No.	DCF (Specified)	Indicative Profit	Waste	Ore	Au Mined	Au Grade	Cu Mined	Cu Grade
49	277.56	374.64	388.66	126.88	31.50	0.25	265,936	0.21
50	276.79	370.39	394.73	127.80	31.73	0.25	267,793	0.21
51	275.13	360.94	406.00	129.79	32.17	0.25	271,635	0.21
52	275.06	360.54	406.48	129.87	32.18	0.25	271,783	0.21
53	273.37	350.96	418.88	131.44	32.60	0.25	275,041	0.21
54	272.75	347.34	422.92	132.10	32.74	0.25	276,295	0.21
55	272.40	345.26	424.69	132.48	32.81	0.25	276,986	0.21
56	271.82	341.89	428.41	133.00	32.92	0.25	278,102	0.21
57	269.96	330.38	440.35	134.64	33.31	0.25	281,332	0.21
58	267.72	315.92	454.74	136.71	33.76	0.25	285,361	0.21
59	267.17	312.59	458.30	137.12	33.86	0.25	286,196	0.21
60	266.95	311.33	459.36	137.32	33.89	0.25	286,507	0.21
61	264.79	297.67	472.29	138.82	34.25	0.25	289,497	0.21
62	264.00	292.55	476.93	139.36	34.38	0.25	290,616	0.21
63	261.95	278.98	487.56	140.87	34.67	0.25	293,472	0.21
64	261.87	278.47	487.92	140.92	34.68	0.25	293,552	0.21
65	261.59	276.53	489.45	141.12	34.73	0.25	293,894	0.21
66	259.14	259.68	504.21	142.71	35.09	0.25	297,058	0.21
67	258.14	252.58	509.70	143.44	35.21	0.25	298,537	0.21
68	257.60	248.67	512.57	143.82	35.29	0.25	299,224	0.21
69	257.03	244.59	515.05	144.18	35.36	0.25	299,848	0.21
70	256.53	241.30	517.66	144.47	35.42	0.25	300,385	0.21
71	255.46	233.82	522.75	145.18	35.54	0.24	301,595	0.21
72	252.81	215.24	537.26	146.71	35.87	0.24	304,542	0.21
73	252.65	214.07	538.21	146.81	35.89	0.24	304,732	0.21
74	251.98	209.23	541.81	147.20	35.97	0.24	305,472	0.21
75	251.85	208.29	542.52	147.27	35.98	0.24	305,629	0.21
76	251.32	204.41	545.19	147.58	36.03	0.24	306,206	0.21
77	251.09	202.75	546.43	147.71	36.06	0.24	306,455	0.21
78	247.52	175.68	564.78	149.72	36.44	0.24	310,172	0.21
79	246.96	171.42	568.17	150.01	36.51	0.24	310,748	0.21
80	246.83	170.33	568.80	150.09	36.52	0.24	310,870	0.21
81	245.10	156.75	578.09	151.02	36.70	0.24	312,555	0.21
82	244.91	155.11	579.02	151.14	36.72	0.24	312,752	0.21
83	244.43	151.48	581.38	151.39	36.76	0.24	313,212	0.21
	Optimum Specified Case							
	Selected Shell							

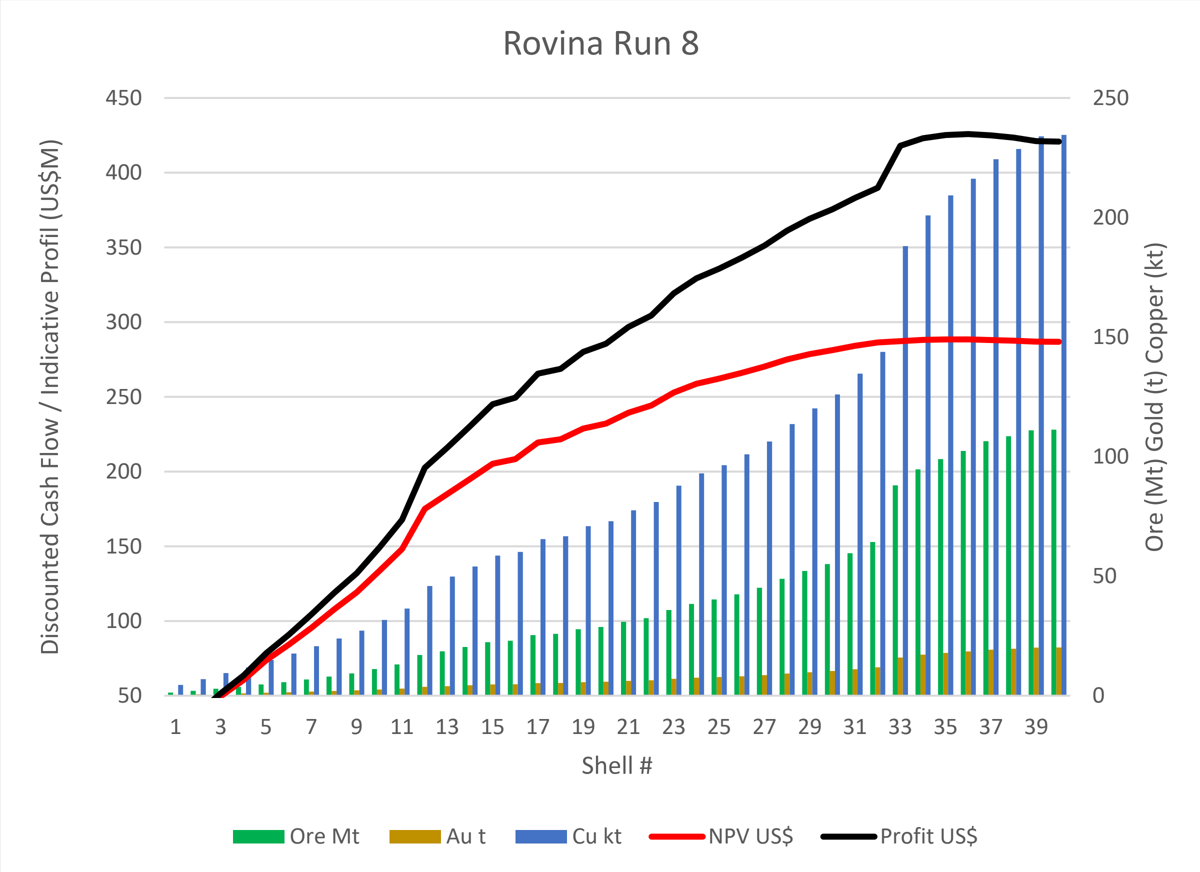


Figure 16.8: Rovina Whittle Analysis

16.2.4.3 Shell Selection Summary

The mining inventories of the selected shells are summarised in Table 16.5.

Table 16.5: Summary of Selected Shells

Area	Pit Shell No.	DCF (Specified)	Indicative Profit	Waste	Ore	Au Mined	Grade
		US$ million	US$ million	Mt	Mt	t	g/t
Colnic Run 29	26	523.03	785.64	108.63	73.24	41.48	0.57
Rovina Run 8	32	286.36	389.79	125.92	64.29	6.22	0.25

16.2.5 Project Design

All the pit designs were developed using the Deswik.CAD suite of software packages. They were based on the optimal shells selected as detailed in 16.2.4 and utilised the latest ESM resource block models that have been reviewed and signed off by CCIC MinRes.

The models were coded with the appropriate batter angles, berm widths, bench, and stack heights for the different rock/material types for each deposit. The slope design parameters (summarised in Table 16.1) were based on geotechnical design criteria provided by KCB as used in the open-pit optimisation.

The criteria for pit and waste stockpile ramp designs were based on the width and turning circle of 90 t dump trucks, as this size truck is likely to be used as the OEM trucking fleet. Ramp gradients are 10 %. Wherever possible, the ramp exits were located at the closest possible distance to the waste storage facilities to minimise ex-pit haulage.

The pit and waste stockpile design outlines recognised lease boundaries, neighbouring villages, and the local road infrastructure. An additional requirement to siting the stockpile locations was to avoid all identified future exploration zones so as not to sterilise these areas. This had a slightly negative impact of increasing the ex-pit hauling distances in some cases.

Within the pit designs, a minimum distance of 20 m is required between the pit edge and final stockpile toe, which is considered acceptable. However, this is only a minimum separation distance as the US$2,500/oz gold pricing scenario was used to demarcate the surface area to be left free of infrastructure that might sterilise the remaining open pit / underground resources. Hence, pit rim to waste rock dump (WRD) toe distances will exceed the minimum of 20 m in most cases. A minimum mining width of 20 m is maintained in the design to ensure representative grades in the mining blocks and to match the equipment suite used in the design phase.

16.2.5.1 Haul Roads

Where possible, existing haul roads will be used for ore and waste. However, a number of temporary haul roads will be required during the LOM of both pits. All haul roads have been laid out on the overall site plans. Refer to the Section 18 for further details regarding the haul road network.

16.2.5.2 RVP Pit Designs

The RVP area consists of two mining areas containing one pit each. These are the Colnic pit and the Rovina pit, and their designs are developed based on the final whittle shell selection and are shown in Figure 16.9 and Figure 16.10, respectively.

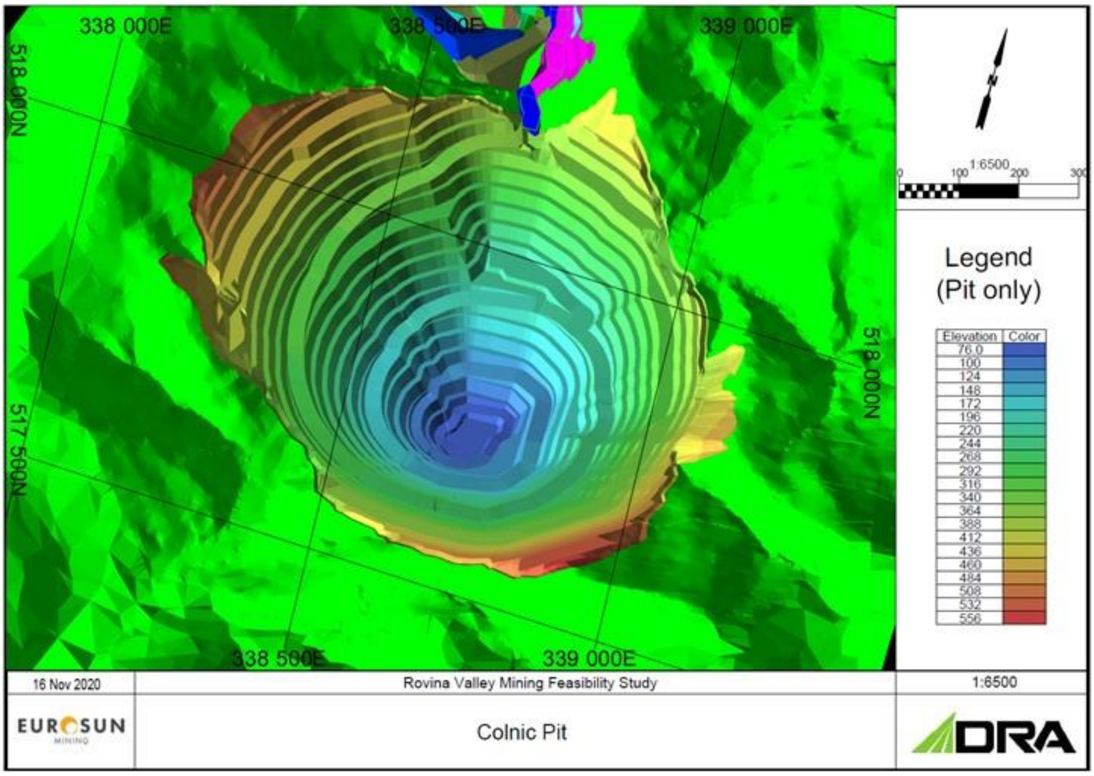


Figure 16.9: Colnic Pit Design

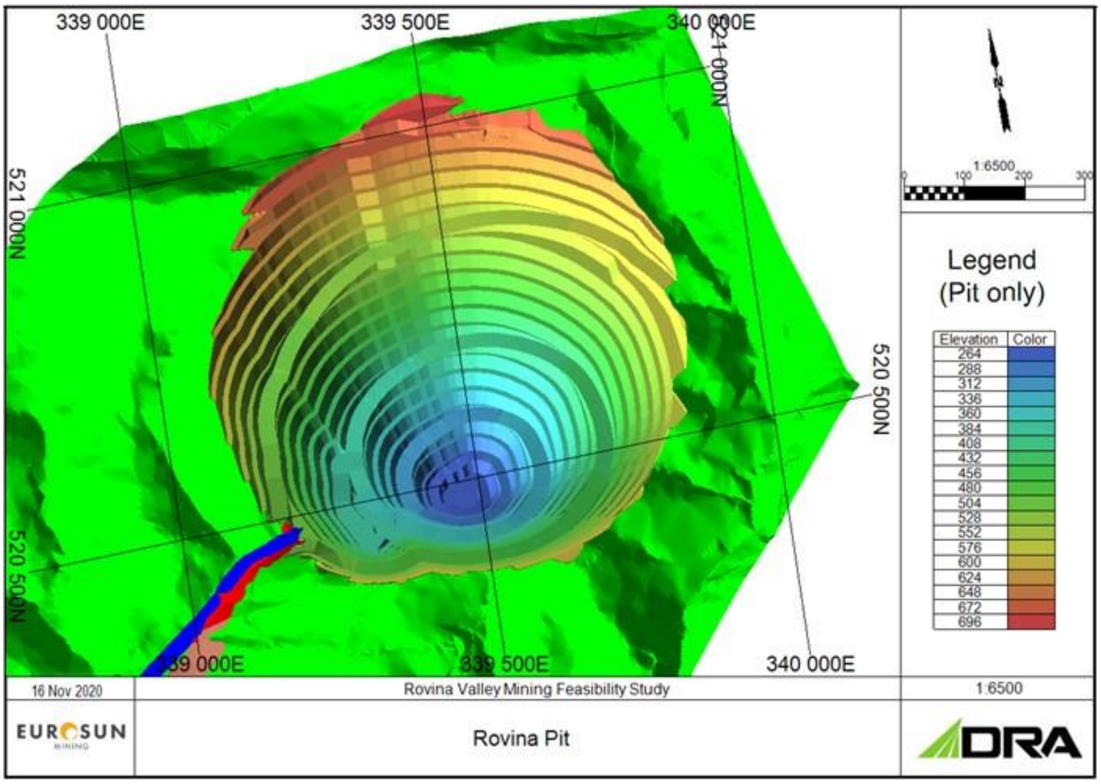


Figure 16.10: Rovina Pit Design

Two stages of mining are planned for the Colnic pit as depicted in Figure 16.11.

The two-stage design was a guideline to ensure that the requisite haulage ramps are available to progress the Stage 1 mining and simultaneously commence the Stage 2 pushback once spare equipment capacity is available.

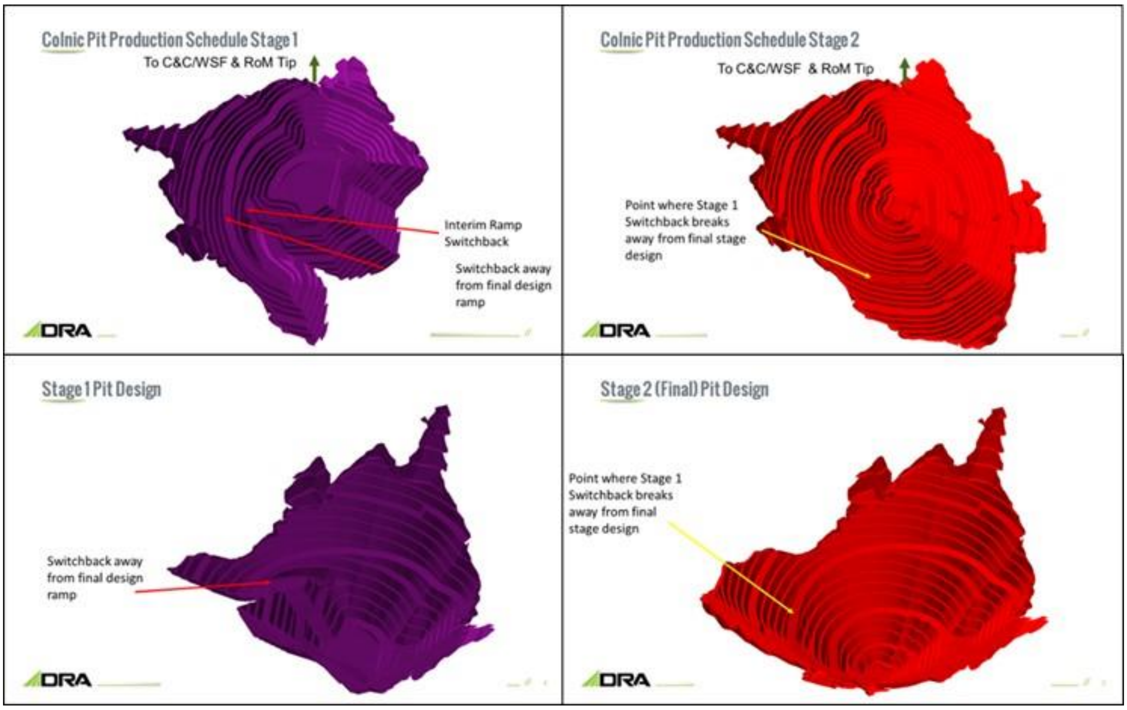


Figure 16.11: Colnic Pit Two-Stage Design

16.2.6 Project Production

The scheduling undertaken for the RVP is based on the ore and waste inventories for each of the pit designs. The aim of the scheduling component was to ensure that the mining process allowed for the following:

- Pre-stripping and stockpiling of sufficient ore for the commissioning of the processing plant and reaching and maintaining the annual processing plant feed rate with the application of a 0.45 g/t gold cut-off for the first 2.5 years, followed by a 0.4 g/t gold cut-off for the next 2.5 years of the Colnic pit LOM.
- A practical and realistically achievable schedule in terms of fleet deployment, equipment productivities, and bench turnover rates based on the total storage constraint of the Colnic co-disposal facility.

The mining schedule aims to maximise value by:

- Reducing waste mining during the early years as much as possible
- Mining high-grade materials during the early years of the LOM to enhance the project payback and overall business case
- Delaying mining of areas with higher ore mining costs
- Limiting the total waste and ore excavator (200 t) fleets to three

Further requirements for the mining schedule were as follows:

- The Colnic ore will be mined first due to its higher average grade and higher ore value per tonne.
- The Rovina ore will follow as soon as all the high-grade stockpiles from the Colnic pit are depleted.

Based on the process plant design capacity of 7.2 Mt/a, the mining schedule aims to meet the same capacity with the exception that the mining will also target higher grades to feed to the processing plant and also generate low-grade stockpiles.

Due to the relatively short life of each of the mining areas, a monthly production schedule was done for the full LOM. For reporting purposes, the monthly production schedule can also be aggregated into calendar and/or financial years.

All mining schedules were generated in XPAC, RPMGlobal's proprietary scheduling software.

16.2.7 Operating Assumptions

16.2.7.1 Available Operating Hours

The operation is planned to utilise four crews on a three eight-hour shift roster for 353 days (365 days per year minus 12 public holidays) of the year based on mining being carried out by an owner-operated production and support fleet.

16.2.7.2 Scheduling Results

The scheduling results are summarised in Figure 16.12 to Figure 16.15, which depict the RVP combined LOM schedule, the Colnic LOM schedule, and the Rovina LOM schedule, respectively. The results show that the schedule is a practical solution that targets value and meets all mining and processing goals on a monthly basis. The key features of the final base case schedule include the following:

- The requirement of 10.71 Mt of pre-strip material movement over six months of pre-stripping in the Colnic mining area.
- The requirement for the Rovina pit to be pre-stripped while the Colnic high-grade stockpiles are being depleted. The pre-strip requirement for the Rovina pit is 14.05 Mt.
- The requirement that a maximum annual materials movement of 27.3 Mt be maintained throughout each production year. The two FELs can be utilised to assist in achieving this production performance when planned maintenance and breakdowns would impede achieving this total productivity.

It is important to note that all Whittle optimisation, pit design and production scheduling were done using the preliminary pit slope design assumptions and preliminary Colnic co-disposal facility constraints. While these assumptions are deemed to be within the acceptable accuracy constraints of the Rovina Valley DFS, the interrogation of the final design parameters may allow for further optimisation of these production schedule results.

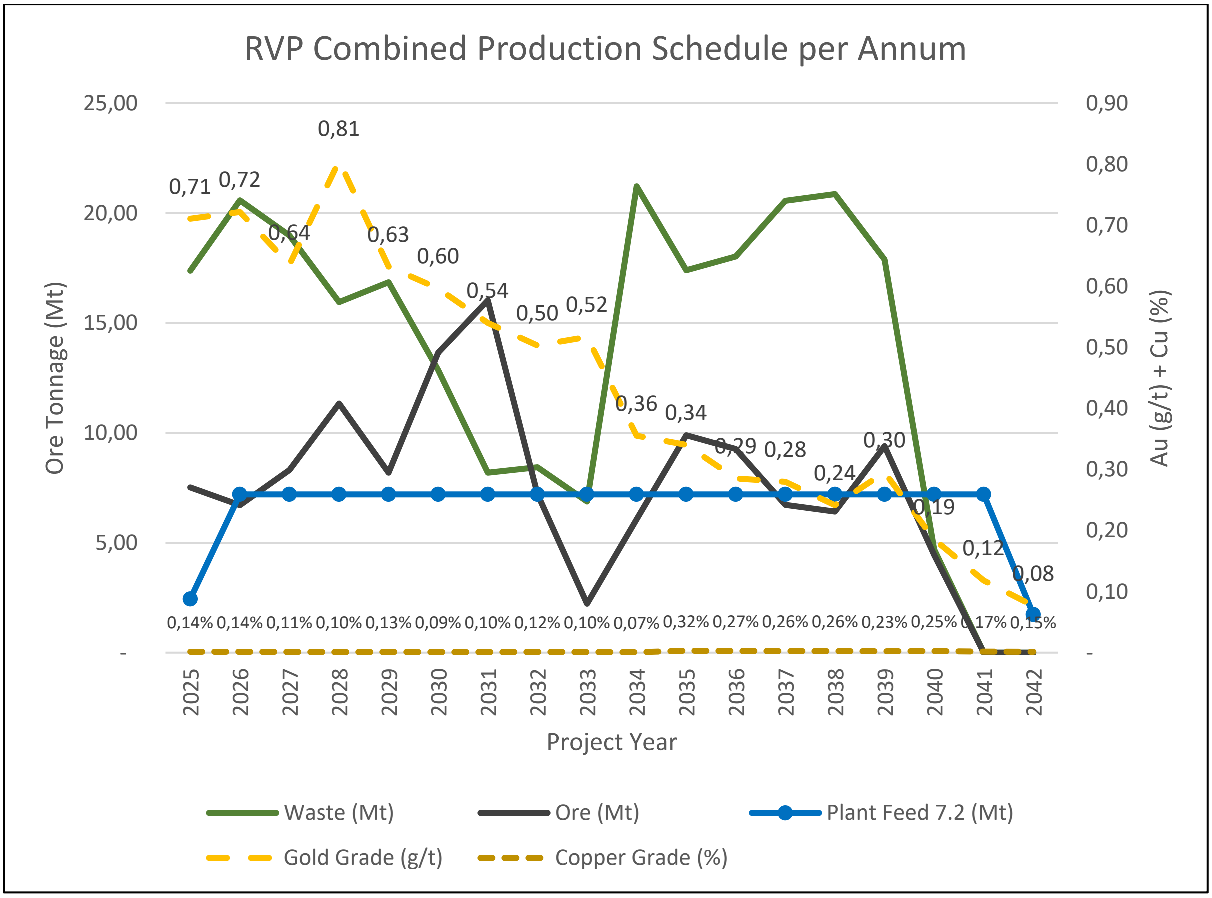


Figure 16.12: RVP Combined Production Schedule per Annum

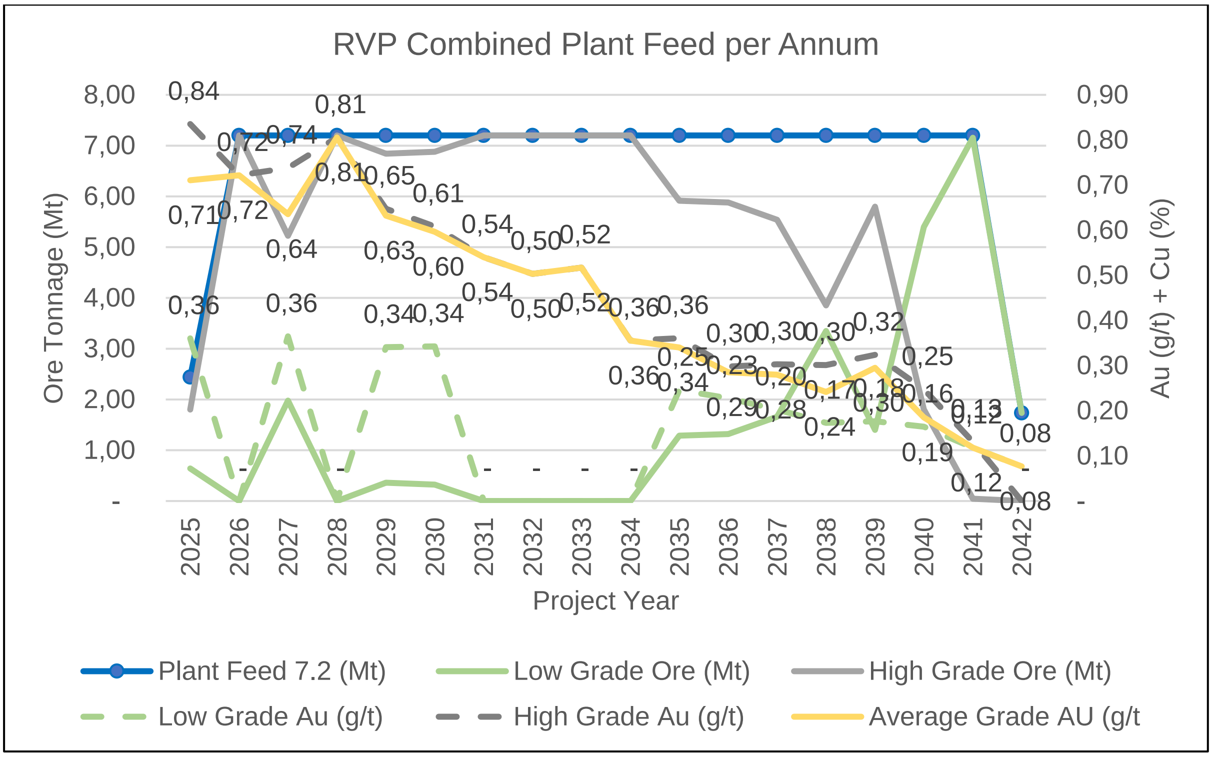


Figure 16.13: RVP Combined Plant Feed per Annum

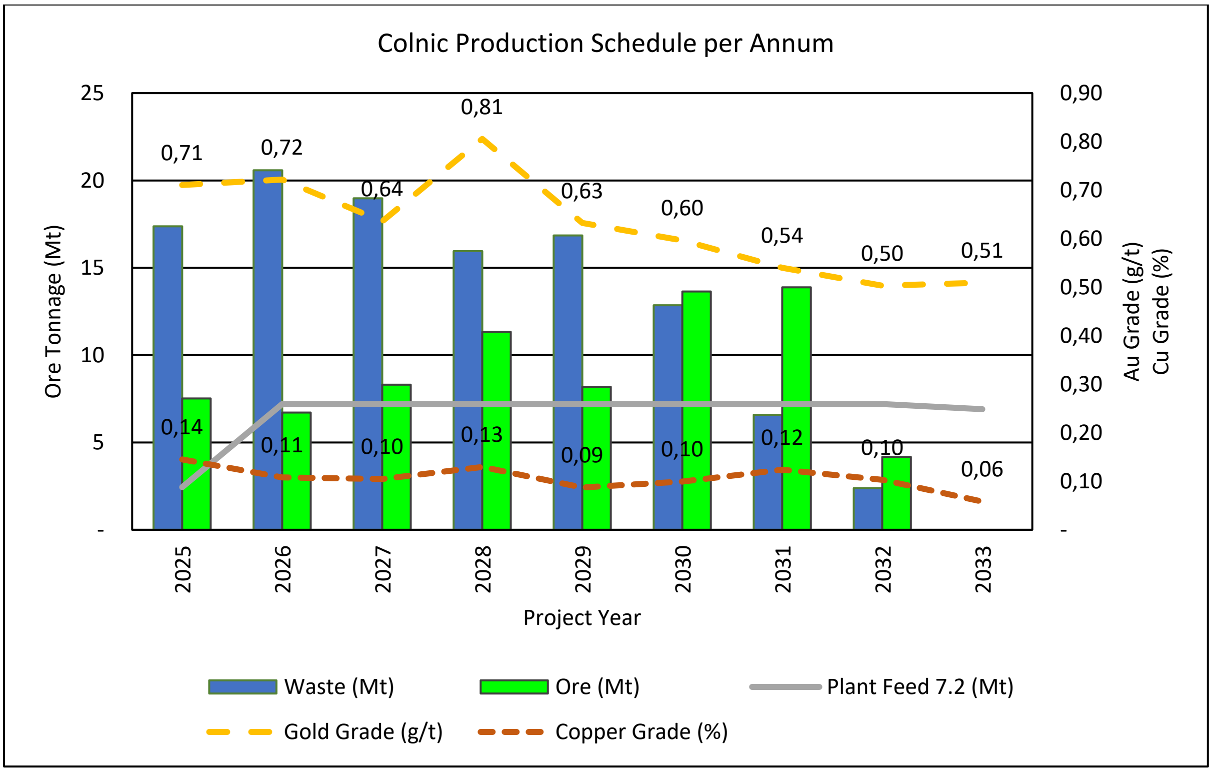


Figure 16.14: Colnic Production Profile

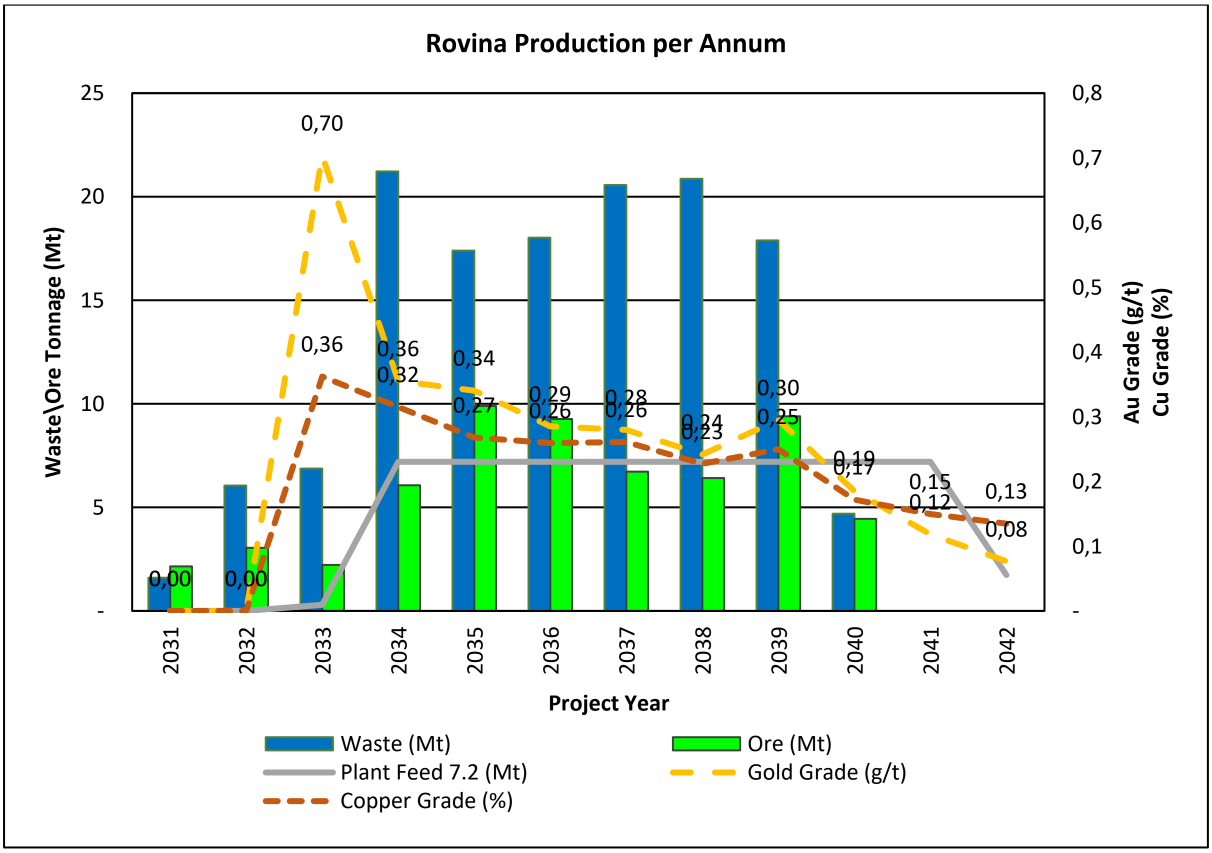


Figure 16.15: Rovina Production Profile

16.3 COLNIC PIT PERIOD PROGRESS PLOTS

The Colnic pit's annual period progress plots are shown in Figure 16.16 and Figure 16.17.

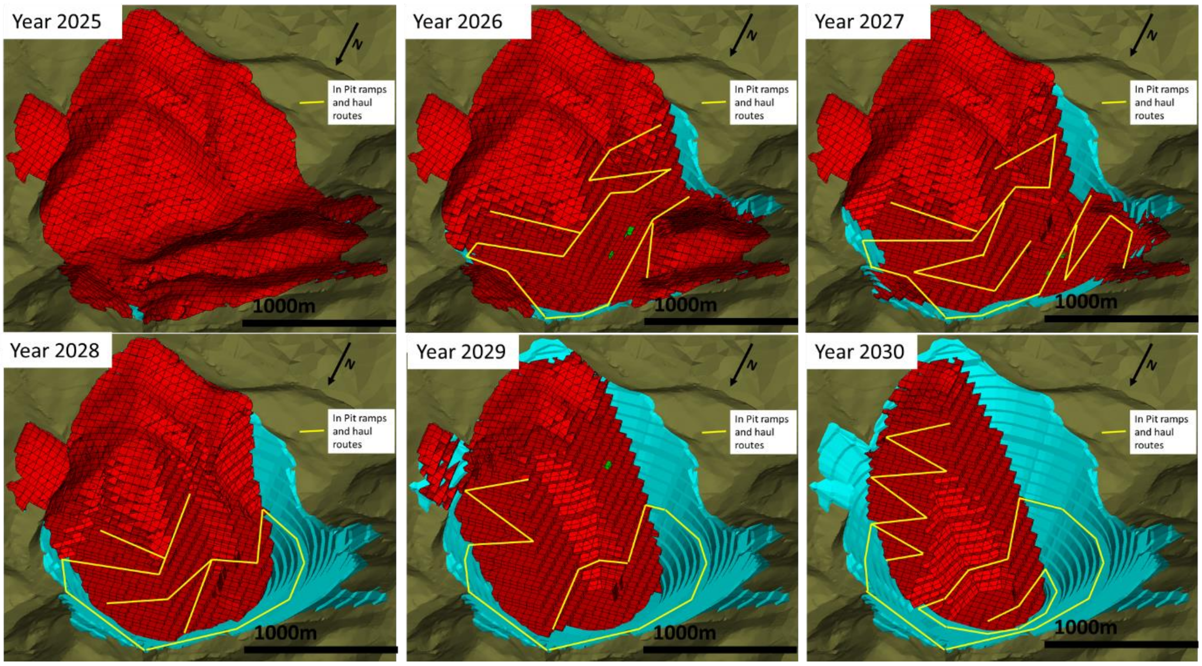


Figure 16.16: Colnic Pit (Mining Years 2025 to 2030)

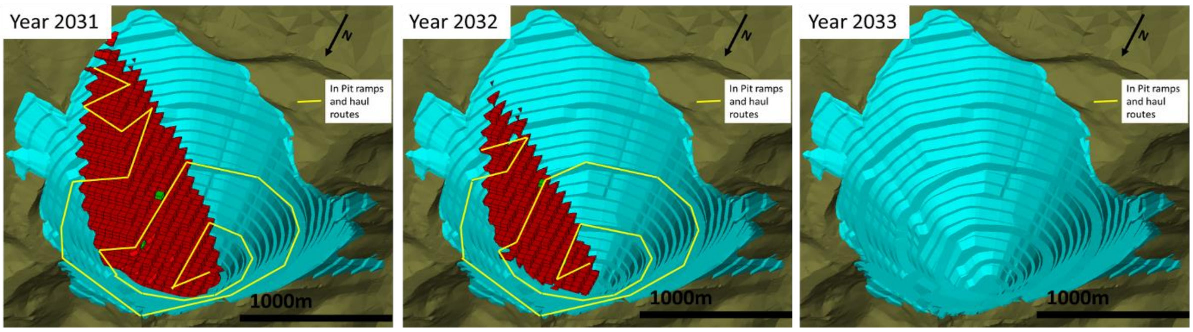


Figure 16.17: Colnic Pit (Mining Years 2031 to 2033)

16.4 ROVINA PIT PERIOD PROGRESS PLOTS

The Rovina pit's annual period progress plots are shown in Figure 16.18 and Figure 16.19.

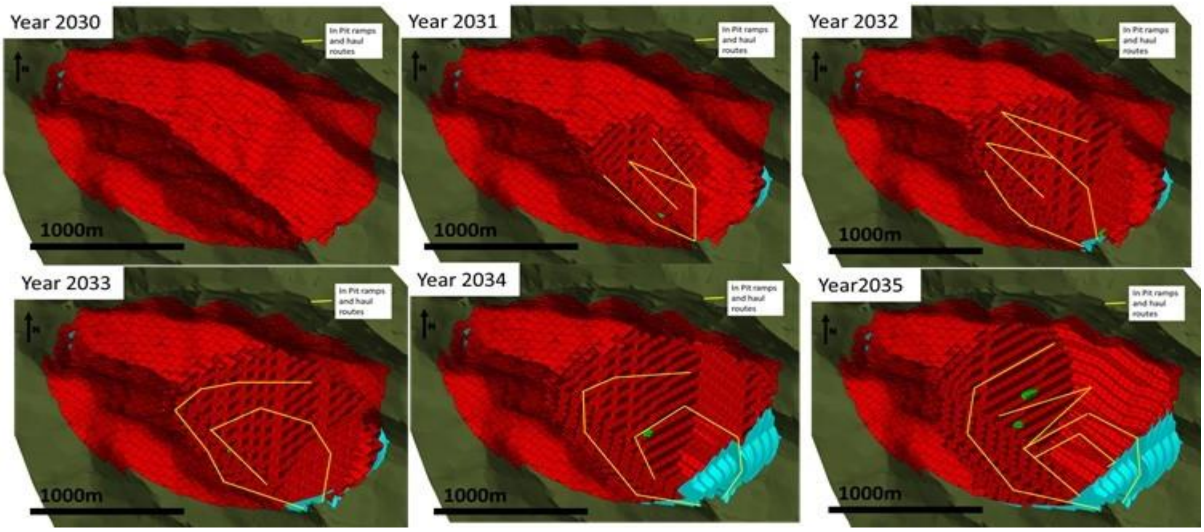


Figure 16.18: Rovina Pit (Mining Years 5 to 10)

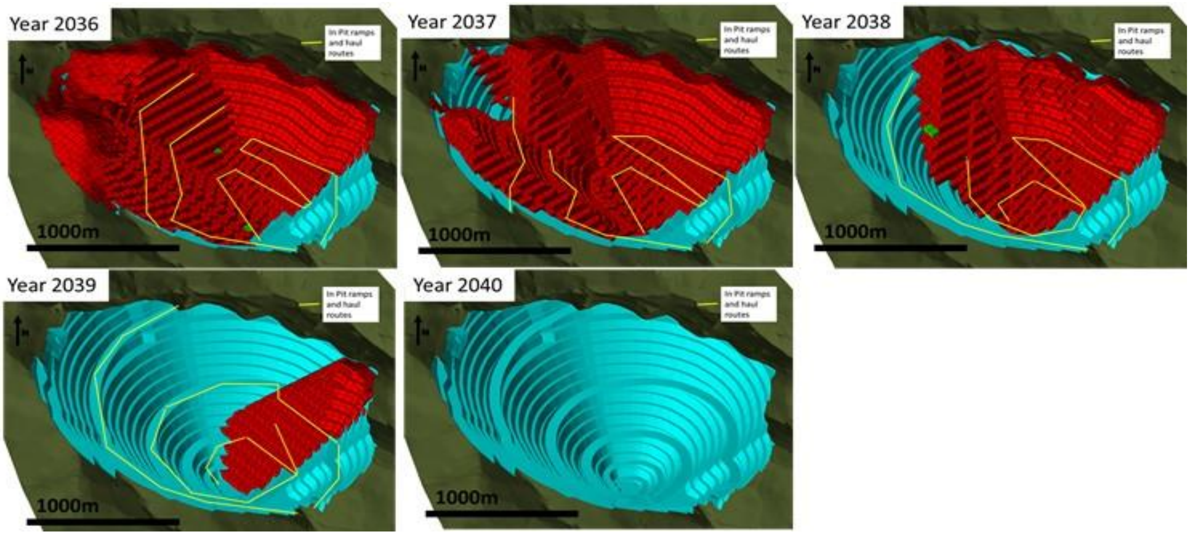


Figure 16.19: Rovina Pit (Mining Years 11 to 16)

16.5 FLEET REQUIREMENTS

16.5.1 Equipment Fleets

The scheduling in XPAC is driven by the excavating capability during each period (i.e. the product of the number of excavators and their productivity).

For scheduling purposes, it was assumed that three 200 t excavators with 12.5 m^3 buckets would be deployed on both waste and ore production. These excavators will be loading rigid dump trucks with a payload capacity of 90 t. The first principal productivity calculations for determining period-by-period material movement weas based on 200 excavators and 90 t rigid dump trucks, which will be allocated to each mining area, taking cognisance of the hauling routes that each fleet will use. The envisioned excavator fleet deployment is shown in Figure 16.20.

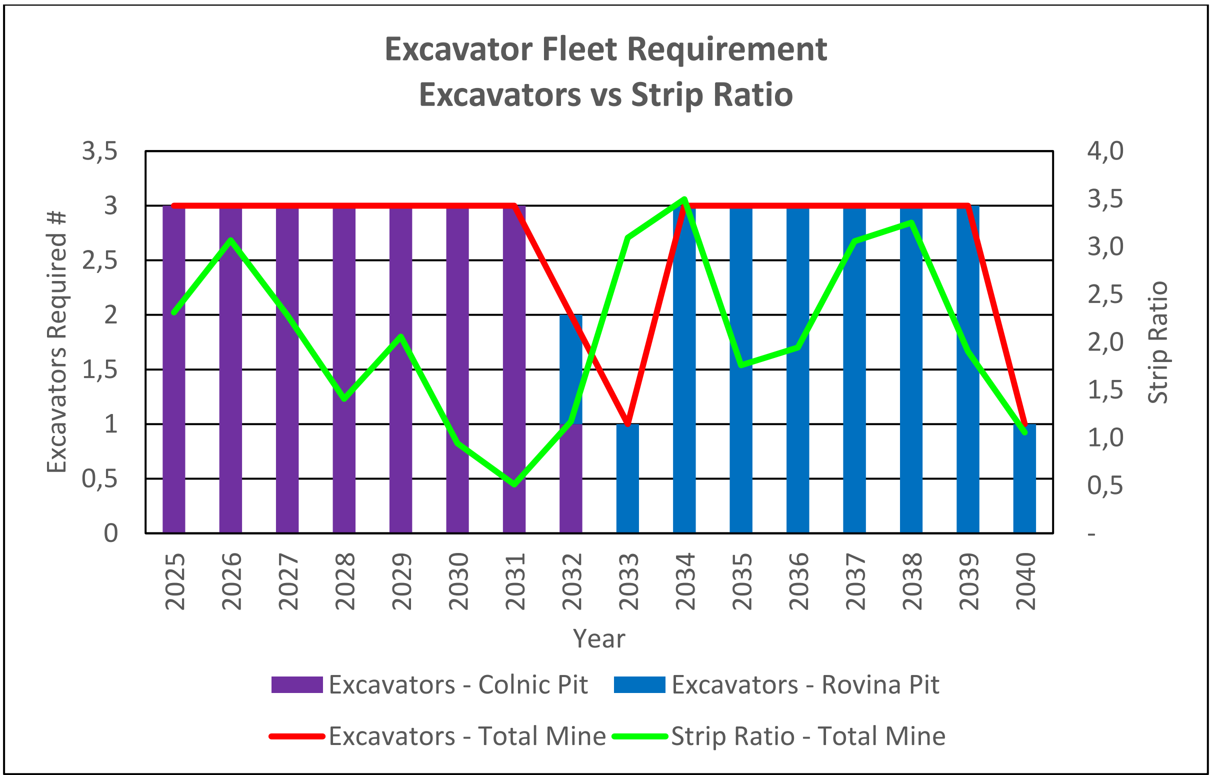


Figure 16.20: LOM Excavator Fleet

This selection was based on the results of a comprehensive request for budget pricing (RFBP) process that was conducted, involving most of the major earthmoving OEMs. The final capital and operating costs are for the selected equipment, which was optimal in terms of costs, size, flexibility, operability, and suitability to handle the conditions that are anticipated in the two planned open-pit mining operations that currently make up the RVP.

16.5.2 Excavator Productivity

Excavator productivity varies with the two loader types as indicated in Table 16.6.

Table 16.6: Loaders Productivity

Loader Type	Truck Type	Annual Capacity	Unit
Hydraulic Excavator 230 t – 12 m³	DT 90 t	9,874,851	t/a
FEL 10 m³	DT 90 t	4,240,989	t/a

During the first four months of operation, the loader productivity is reduced to reflect commissioning, shift ramp-up, and the ramping up of each operator's skills level.

16.5.3 Haul Roads

The ex-pit haul roads will be designed to minimise the haulage distance between the deposit and the processing plant. All haul roads for ore haulage must be designed using a minimum of three truck widths with longitudinal gradients and crossfall recommendations. All ramps and haul roads shall have a maximum gradient of 10 %. See Table 16.7 and Figure 16.21 to Figure 16.23 for haul road design specifications and typical drawings.

Waste storage facility access haul roads will be designed with less engineering work in areas where these roads will regularly be moving as the WMF develops. Permanent WMF haul roads must be designed to the same standards as the ore haul roads.

Table 16.7: Haul Road Design Parameters

Description	Code	Unit	Value		
Truck Type			Caterpillar 777G		
Operating Width	TOW	m	6.1		
Tyre Size			27R49		
Outside Diameter	OD	m	2.68		
Bund Height	BH	m	1.34		
Top Width	Top W	m	0.5		
Overall Width	OW	m	4		
Description	**Code**	**Unit**	**Road Type**		
			Two-Way Ramp	**One-Way Ramp**	**Two-Way Road**
Road Width Multiplier			3	1.5	3
Safety Bunds			1	1	2
Calculated Road Width	RW	m	18.3	9.1	18.3
Drain Width	DW	m	1	1	0
Total Road Width (rounded up)	TRW	m	24	14	26

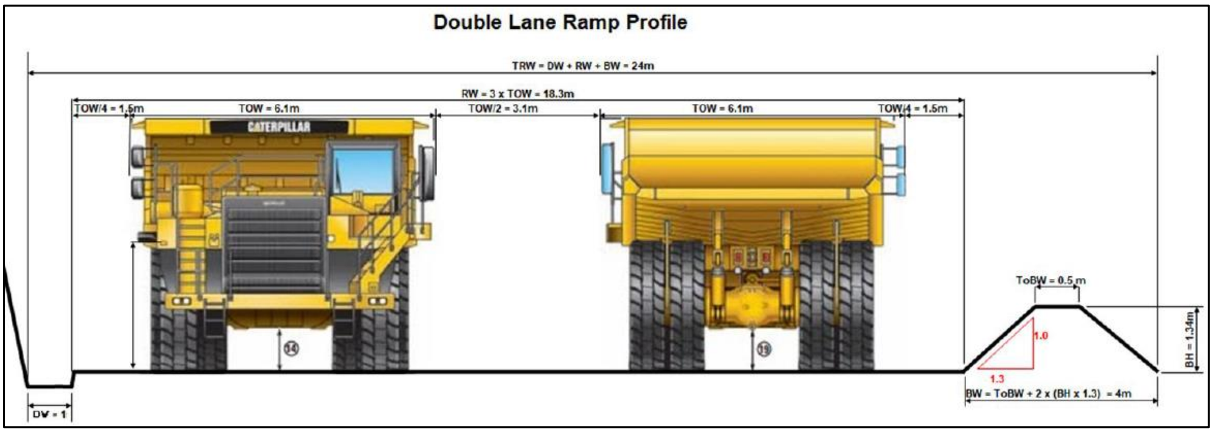


Figure 16.21: Double-Lane Ramp Profile

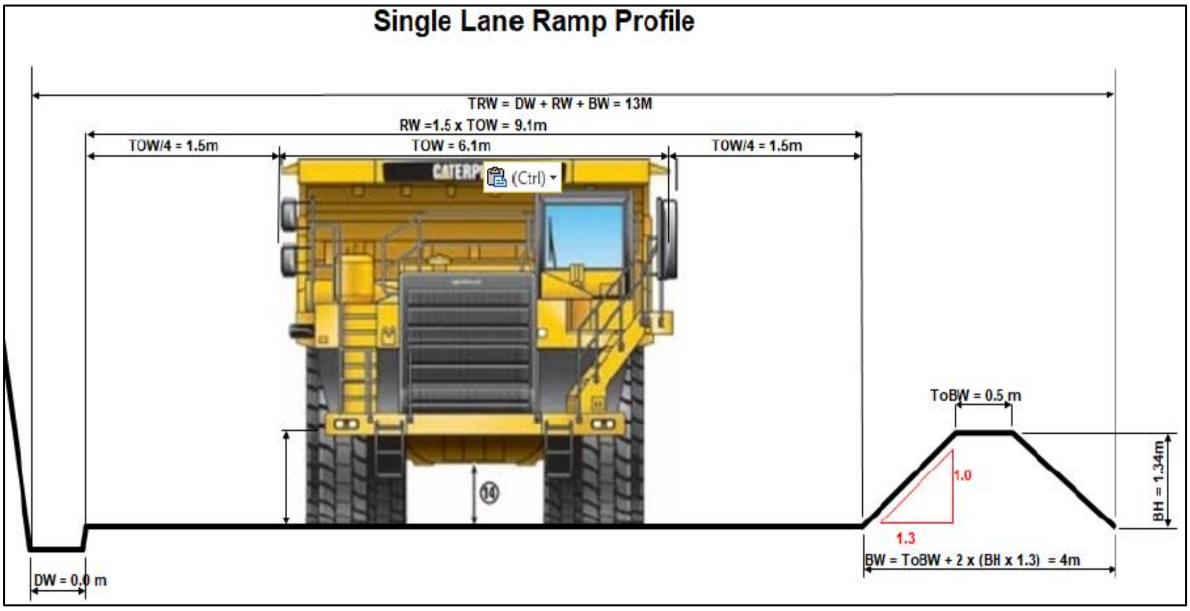


Figure 16.22: Single-Lane Ramp Profile

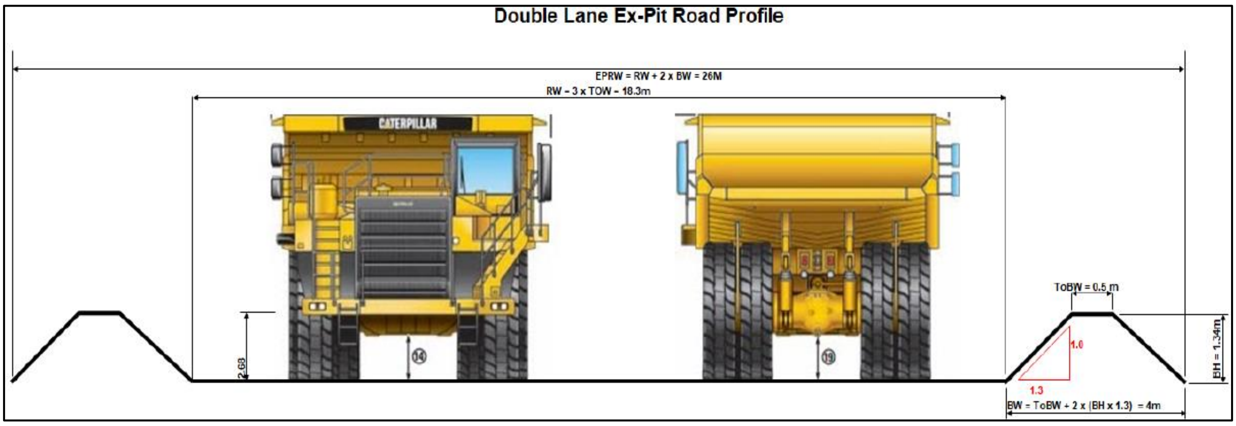


Figure 16.23: Double-Lane Ex-Pit Road Profile

16.5.4 Waste Rock Stockpiles

The waste stockpiling will be staged appropriately to minimise haul/conveyor distances throughout the LOM.

The waste stockpile construction and final landform are based on the following criteria:

- The maximum height of the waste storage facilities is currently set at 650 masl.
- A swell and recompacting factor of 50 % was utilised to calculate the material placement density of waste on the waste storage facilities.
- The stockpile BFA is designed at 30° during construction, with 10 m berms separating benches. During the rehabilitation phase, the WRD side slopes will be progressively dozed down into continuous slopes without benches, as required for agricultural use. After rehabilitation, the final landform slope will not exceed 20°in overall slope angle.

- The waste storage facilities will be built with a minimum 1:100 gradient on the top surface to ensure effective water shedding.
- All stockpile locations were selected outside the boundaries as indicated by the Whittle gold price scenario of US$2,500/oz. Future prospecting zones were also considered so as not to sterilise any potential resource.
- The minimum operating width on the waste stockpile is 40 m based on the stacking conveyors selected.
- All the waste storage facilities were designed with road ramps of 10 % gradient and conveyor operation at a maximum gradient of 15°.
- It has been assumed that all waste is benign and does not require any neutralising treatment or containment.

The placement of the waste rock stockpiles is constrained and should be optimised to limit hauling and conveying distances over the LOM. These optimisation results are shown in Figure 16.24 to Figure 16.27 for waste rock and ore.

Colnic ore will be transported to the Colnic ROM tip position for direct feed to the process plant or high- and low-grade stockpiles depending on short planning requirements.

Colnic waste will be transported to the Colnic waste crushing facility and then conveyed Colnic waste storage facility.

The Rovina ore will be batch-conveyed from the Rovina crushing station and conveyed to the Colnic ROM tip position for direct feed to the process plant or high- and low-grade stockpiles dependant on short planning requirements.

The Rovina waste will be trucked to the northern area of the Colnic waste storage facility and later batch-conveyed from the Rovina crushing station and conveyed to the Colnic Pit backfill waste storage facility.

The Rovina low-grade will use the depleted Colnic high-grade stockpile area once it is empty. Owing to the Rovina pit's location, road hauling of the Rovina low-grade is planned, but this may be optimised to conveying if spare capacity is available.

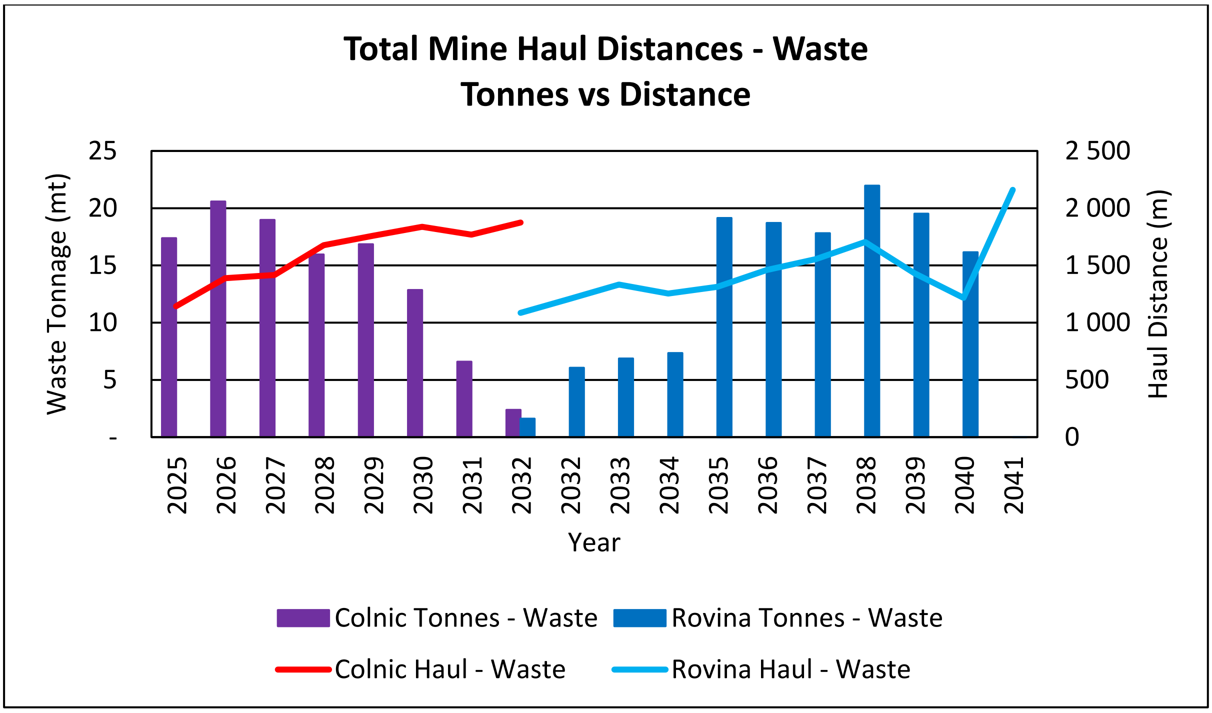


Figure 16.24: Weight Averaged One-way Haul Distances – Total Mine Waste

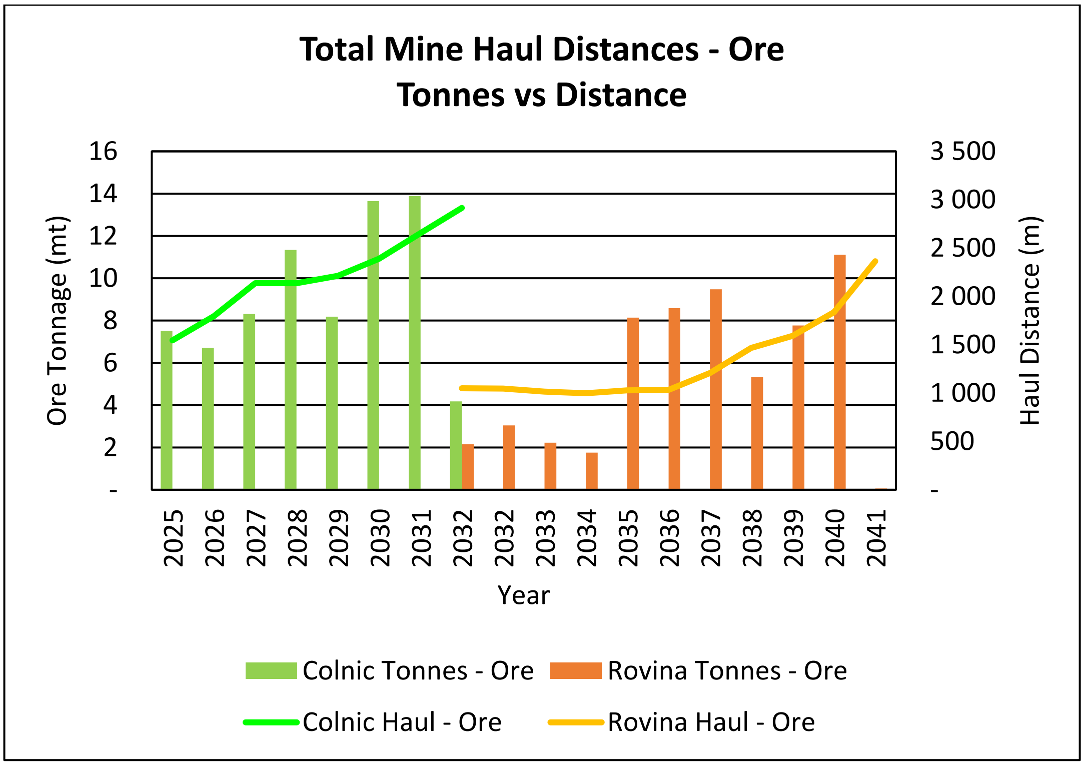


Figure 16.25: Weight Averaged One-Way Haul Distances – Total Mine Ore

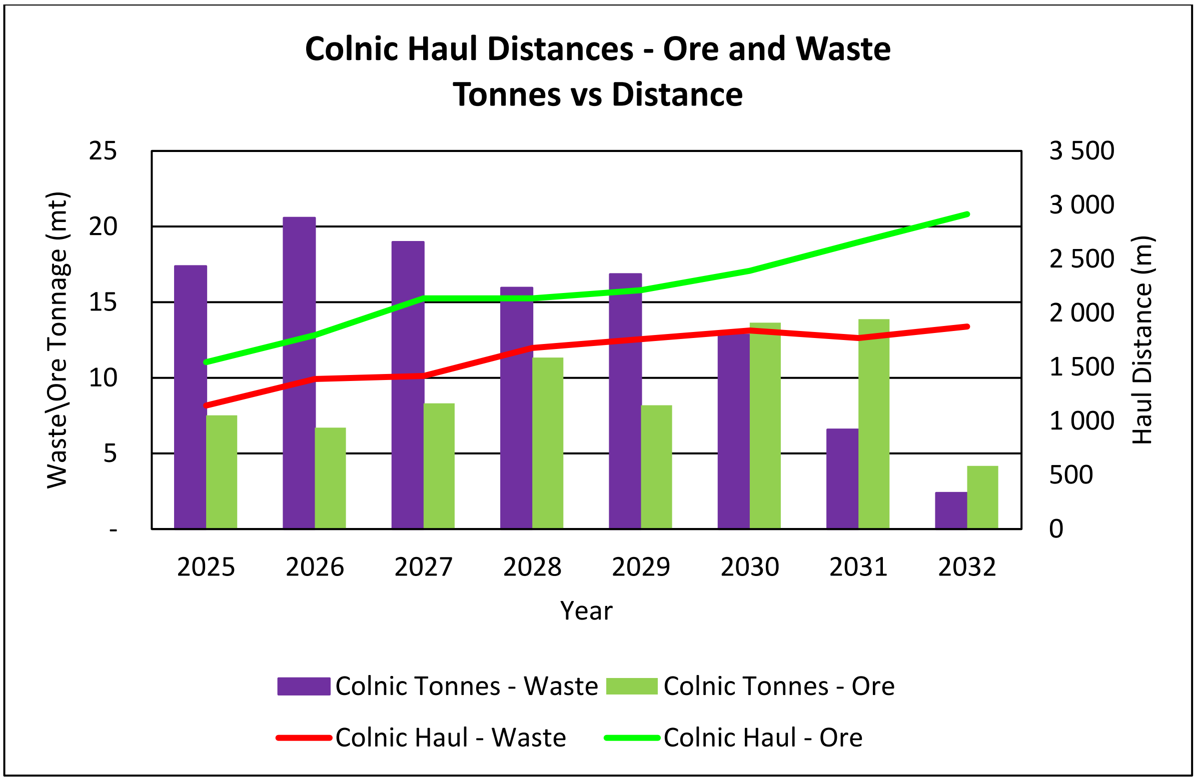


Figure 16.26: Weight Averaged One-Way Haul Distances for Ore and Waste – Colnic Pit

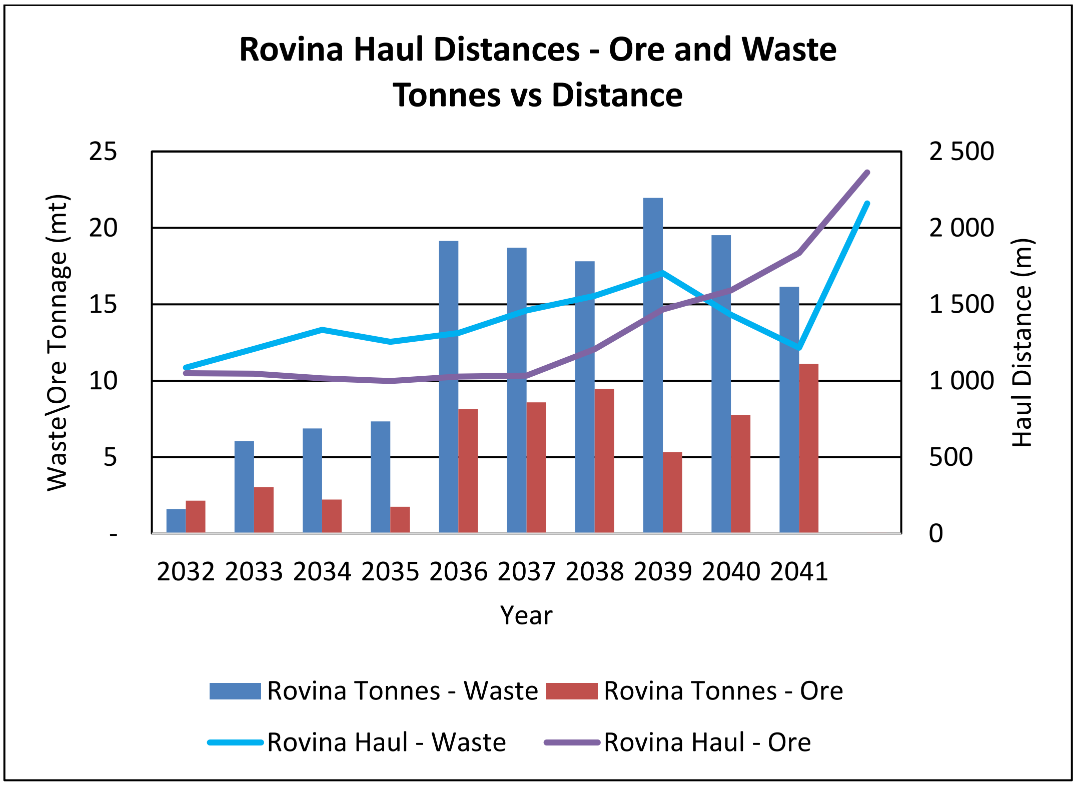


Figure 16.27: Weight Averaged One-Way Haul Distances for Ore and Waste – Rovina Pit

16.6 PROJECT PERIOD PROGRESS STAGES OF THE RVP

The LOM activities of the Colnic and Rovina open pits involve the sequential mining of waste and ore material that is initially hauled by 90 t rigid dump trucks and then crushed and conveyed to either the co-disposal facility or process plant. The process plant tailings are classified into rougher and cleaner tailings, which are sent either to the co-disposal waste facility or to a lined cleaner tails storage facility. The period progress plots show the ongoing filling of the Colnic waste storage facility and then the filling of the Rovina pit backfill waste storage facility as depicted in Figure 16.28 and Figure 16.29, respectively.

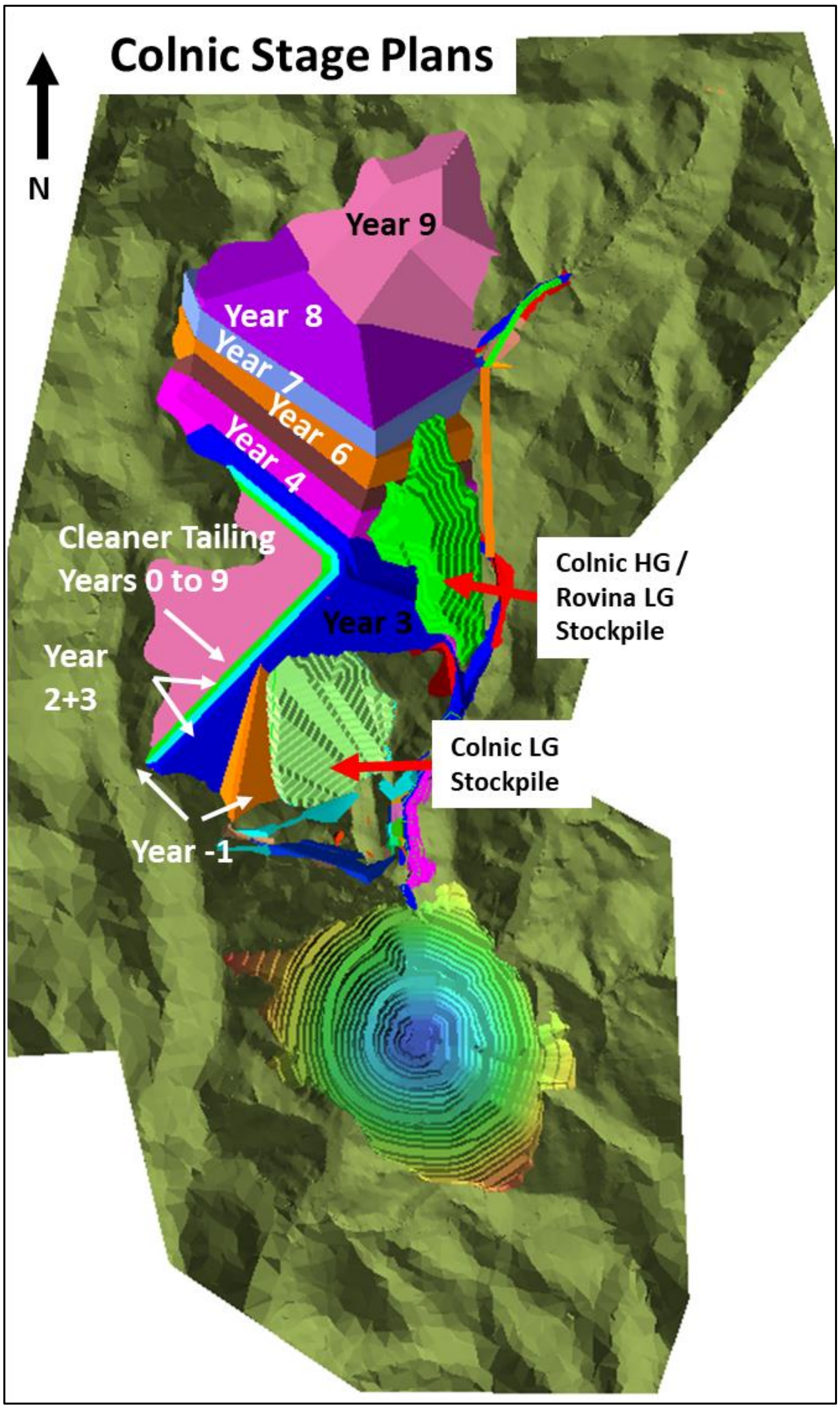


Figure 16.28: Colnic Pit and Co-Disposal Storage Facilities and Stockpiles

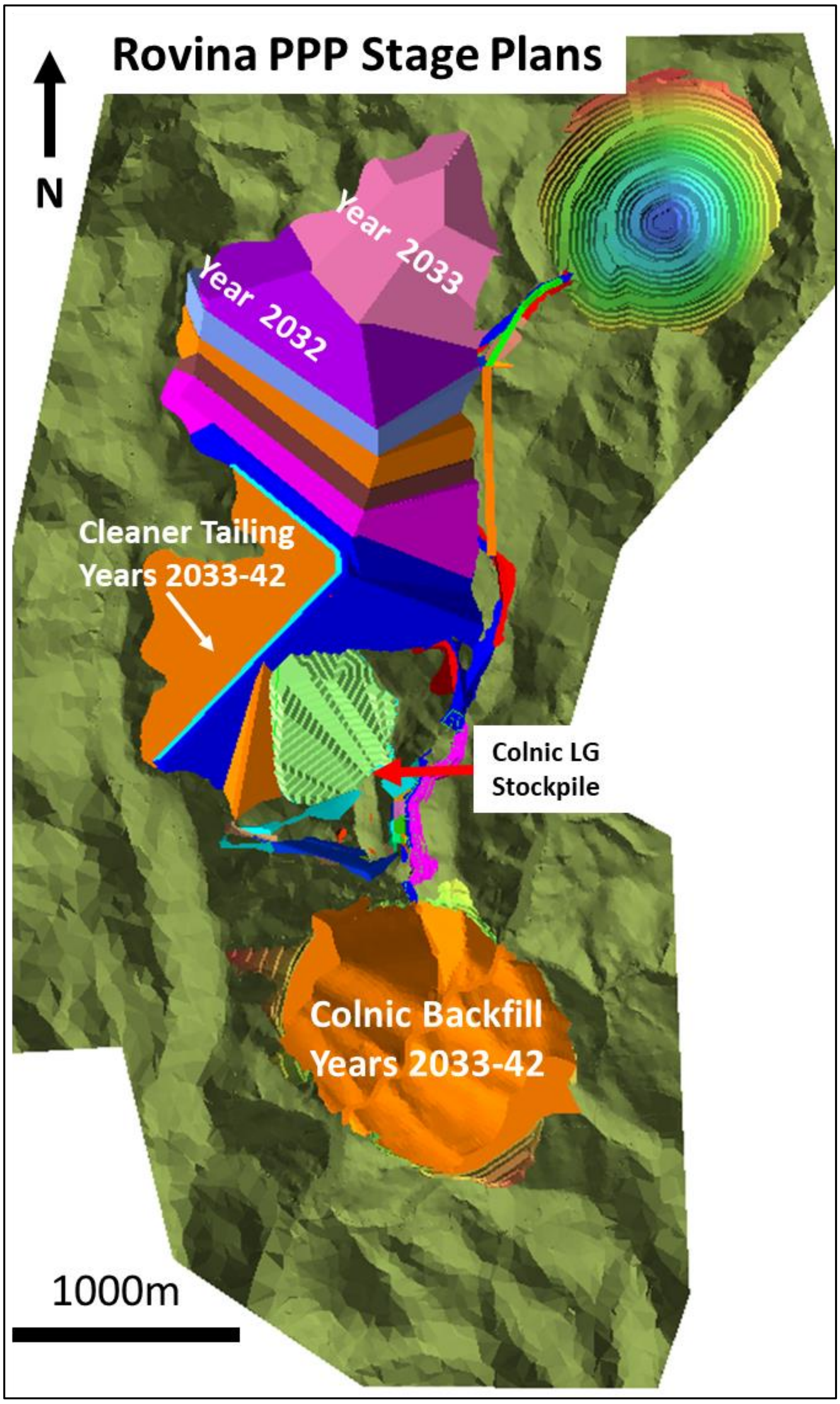


Figure 16.29: Rovina Pit and Co-Disposal Storage Facilities and Stockpiles

16.7 WASTE MANAGEMENT FACILITY – CONVEYORS AND INFRASTRUCTURE

The Colnic waste will be hauled to the waste crushing facility situated along the haul road to the ore crushing pad. Waste will be tipped into the receiving bin and fed from the bin via an apron feeder to the semi-mobile crushing station.

The waste passes over a vibrating grizzly feeder (VGF) cutting at a 450 mm particle size, with the oversize reporting to a C200 jaw crusher and the undersize reporting directly to the waste extraction conveyor. The jaw crusher product (−450 mm) also reports to the extraction conveyor.

The crushing system has an average capacity of 3,891 t/h and can handle up to 5,000 t/h peaks.

Table 16.8 highlights the key parameters included in the crusher selection process, and Figure 16.30 shows the general arrangement (GA) and flow of waste material through the tipping and crushing process.

Table 16.8: C200 Waste Crusher System Key Parameters

C200 Crusher Parameters Description	Unit	Value
Maximum product size	mm	450
Design percentage passing ± 5 %	%	77
VGF undersize	t/h	2,743
Crusher feed	t/h	817
Closed crusher size	mm	300
Operational efficiency	%	80%
C200 crushing throughput	t/h	1,148
C200 crusher production	t/h	3,891

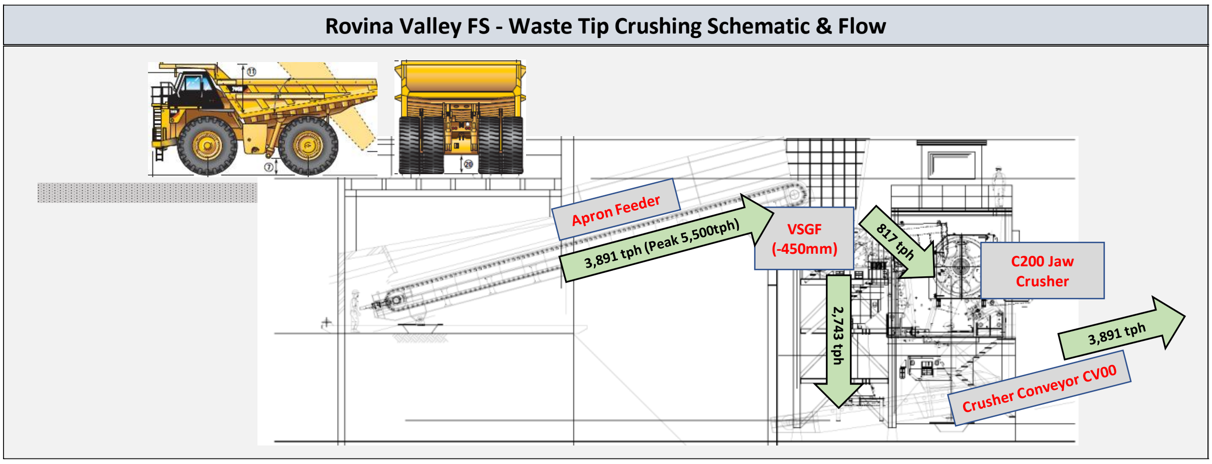


Figure 16.30: Waste Crusher GA and Flow

The crushing system waste stream (see Figure 16.31) is transferred via the extraction conveyor (CV-00) to the waste facility conveyor system. The waste facility conveyor system has a total of five conveyors and a telescopic spreader/stacker, scheduled in the capital costing according to the waste facility building process explained in the earlier section of this report.

The first three conveyors in the system (CV-01, CV-02 and CV-03) have an average design capacity of 3,891 t/h. The last conveyors in the system (CV-04 and CV-05), as well as the spreader/stacker, have an average design capacity of 4,800 t/h to cater for plant tailings added at the transfer point between CV-02 and CV-03.

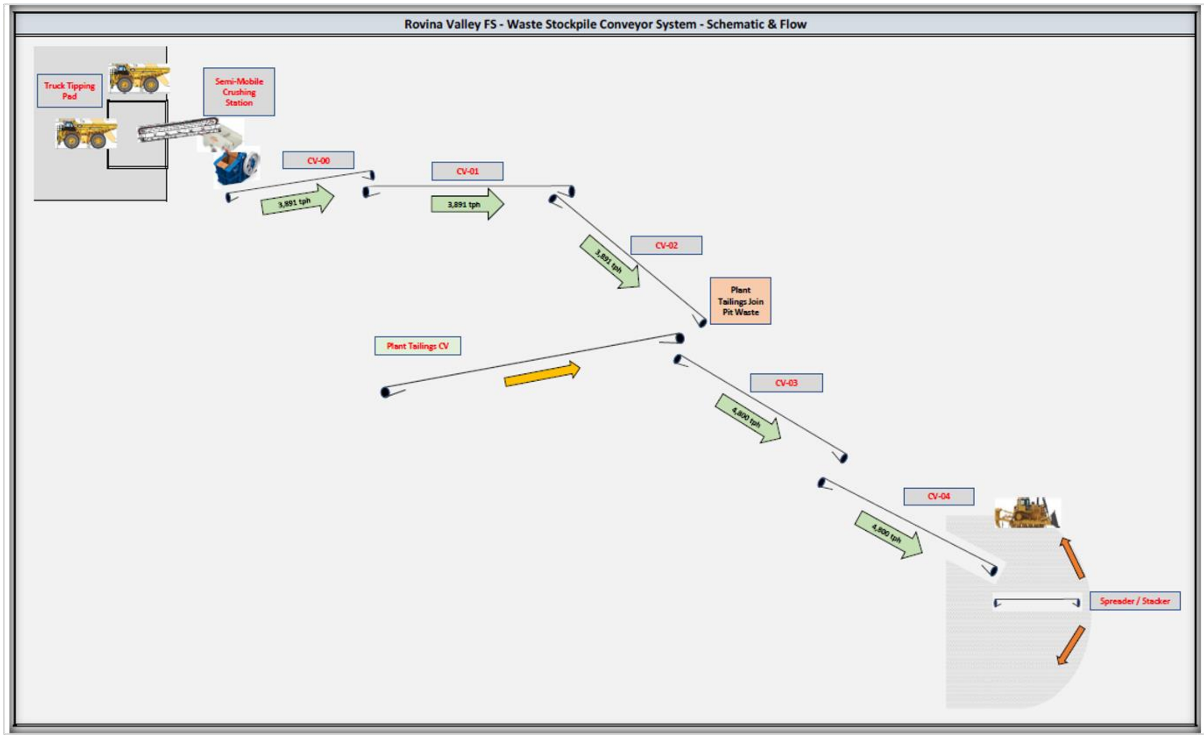


Figure 16.31: Waste Conveyor System Schematic

A comprehensive RFBP process was conducted to obtain pricing for the capital costing and scheduling aligned with the waste facility planning and design.

Some key items addressed during the design:

- Skid-mounted stringer sections that are 2.4 m long to ensure ease of movement and extension of conveyors
- Standardisation of conveyor drive units to minimise spares keeping
- Standardisation of the class of belting to minimise spare quantity
- Track-mounted telescopic spreader stacker to facilitate final waste storage facility construction with dozer assistance

The waste facility conveyor and semi-mobile crusher system will be utilised for the life of the Colnic pit mining. Once mining of the Rovina pit commences, these crusher and conveyor systems will be reclaimed and installed (reused) from the Rovina pit to be able to backfill the Colnic pit with Rovina waste and rougher tailing. This move will be done once the waste handling facility has reached the design limit. The timing of the move can, however, be optimised during the LOM planning process.

Once the crusher and conveyor systems have been moved to the Rovina pit position, the system will be utilised for both ore and waste in a batching regime. Ore will be conveyed to the existing plant ROM tip crushing system with a diversion chute arrangement. Waste will be conveyed via the same system to backfill the Colnic pit.

The system design in the Rovina pit configuration has been adapted to handle both waste and ore and will have a capacity of 4,800 t/h. This is possible as the conveyors in this configuration have lower inclinations and the system will also require less absorbed power.

Table 16.9 depicts the conveyor schedule summary for both the Colnic stage and the Rovina stage of mining.

Table 16.9: Colnic and Rovina Conveyor Schedule Summary

Conveyor Number	Conveyor Description	Belt Width mm	Length (Pulley Centre to Centre) (m)	Conveyor Vertical Lift (m)	Design Capacity (t/h)	Speed (m/s)	Belt Class	Covers (Top/ Bottom) (mm)	Installed Drive Unit (kW)	Explanatory Notes
CV01 Colnic	Leg 1 from Tip and Crushing	1,500	136	33	3,891	3.0	ST2500	10/6	2 × 250	Tandem Drive
CV02 Colnic	Leg 2 to Tailings Transfer	1,500	530	122	3,891	3.0	ST2500	10/6	4 × 370	Tandem Drive
CV03 Colnic	Leg 3 from Tailings Transfer	1,500	418	101	4,800	3.0	ST2500	10/6	4 × 370	Tandem Drive
CV04 Colnic	Leg 4 to Spreader/Stacker or CV05	1,500	1,210	20	4,800	3.0	ST2500	10/6	2 × 370	Tandem Drive
CV05 Colnic	Leg 5 Optional to Spreader/Stacker	1,500	697	20	4,800	3.0	ST2500	10/6	2 × 370	Tandem Drive
CV01 Rovina	Leg 1 from Tip and Crushing	1,500	388	20	4,800	3.0	ST2500	10/6	2 × 250	Tandem Drive
CV02 Rovina	Leg 2	1,500	880	25	4,800	3.0	ST2500	10/6	2 × 370	Tandem Drive
CV03 Rovina	Leg 3	1,500	347	20	4,800	3.0	ST2500	10/6	2 × 370	Tandem Drive
CV04 Rovina	Leg 4	1,500	160	40	4,800	3.0	ST2500	10/6	2 × 370	Tandem Drive
CV05 Rovina	Leg 5 to Colnic Pit	1,500	791	80	4,800	3.0	ST2500	10/6	4 × 370	Tandem Drive
Colnic Phase Summary	Total Length and Drives	1,500	2,991	2 × 250 kW Drive Units						
				12 × 370 kW Drive Units						
Rovina Phase Summary	Total Length and Drives (Reclaimed from Colnic Waste Management Facility)	1,500	2,566	2 × 250 kW Drive Units						When conveyors are moved to the Rovina pit, a lower class of belting can be utilised
				10 × 370 kW Drive Units						

16.8 PIT DEWATERING

The pit dewatering design is based on the KCB Prefeasibility Dewatering Report (M09759A02.730- October 2013).

The design capacity of the fixed dewatering system is 300 L/s delivered through a Ø300 mm pipe column. Dewatering will report to the collection pond north of the Colnic pit.

The design caters for a total static head of 180 m vertical for fixed dewatering pumps and a 90 m static head for the pit bottom mobile diesel pump.

A fixed pump station will be constructed at a depth of 180 m below natural ground level (NGL)(collection sump). This pump station will consist of two Weir Warman DWU 200 pumps (or equivalent) in series, equipped with 500 kW electric motors.

The two mobile diesel pumps for pit bottom dewatering, one Weir Warman 200DWU per unit, will initially pump to the collection pond. Once a fixed pump station is necessary, pump to the fixed pump station sump. The diesel pumps are rated at 150 L/s and operating one pump will be sufficient for normal dewatering. The second diesel pump can be used during high inflow situations. As per normal open-pit operations, an earth sump or collection area will be created close to the pit bottom to assist with managed collection of storm water.

The fixed pump station sumps will have a dirty water compartment (for settling purposes) and a clean water compartment supplying the fixed pumps through a Ø350 suction manifold. A standby pump train will also be installed. The fixed pump installations will operate 4.57 h/d under average inflow conditions.

The same pumps and design parameters will apply at the Rovina pit later in the LOM; however, due to the Rovina lower inflows, the pumps will be utilised less. Once more detailed information is available on the Rovina hydrology, this design could be optimised as part of the LOM stay-in-business planning.

Table 16.10 and Table 16.11 indicate the Colnic pit dewatering design parameters and design summary, respectively.

Table 16.10: Colnic Pit Dewatering Design Parameters

Description	Value	Unit
Groundwater Seepage (Average to High Flow)	18.8	L/s
Surface Water Inflow (Average)	38.3	L/s
Total (High Flow)	57.1	L/s
Volumes		
Average per Month	148,003	m^3/month
Average per Day	4,933	m^3/d
Design Peak (per Day)	23,760	m^3/d
Design Peak (per Hour)	1,080	m^3/h
Design Peak (Litres per Second)	300	L/s
Average Pumping Hours per Day	4.57	h/d

Table 16.11: Colnic Pit Dewatering Design Summary

Inputs					Outputs (for 300NB pipe)			Pumping Requirements		
Parameter	**Pump Static Head**	**Unit**	**Stage Length**	**Unit**	**Parameters**	**Total Head**	**Unit**	**Parameter**	**Value**	**Unit**
Pump Station 1 – Two-stage (to NGL to Collection Sump)	180.0	m	369	m	Pump Station 1 (to NGL - Collection Sump)	180.00	m	Continuous Pumping	**22**	h/d
Pump Station 2 – Two-stage (Pit Lower to ex-pit area)	180.0	m	294	m	Pump Station 2 (Pit Lower to ex- IPC area)	180.00	m	Volume per Day	**23,760**	m³/d
Diesel Pump – Single-stage (Pit Bottom)	90.0	m	300	m	Diesel Pump (Pit Bottom)	90.00	m	Volume per Hour	**1,080**	m³/h
								Volume per Second	**300**	L/s
	Total Pipe Length		**962.5**	m				Pipe Diameter (NB)	**300**	mm
Required Flow (L/s)	**300**				**Pump RPM (from pump curve) and Power Required**					
					DWU 200 RC – Weir Warman					
Assumptions					**Pump Station**	**RPM**	**kW (Absorbed)**		**Motor**	**Installed Motor (kW)**
Designed for 15 % redundancy					Pump Station 1	1,560	**492.47**	**Per pump (2×)**	525 V	**500**
Static (Vertical Head) kept constant at 180 m (2-stage pumping)					Pump Station 2	1,560	**480.06**	**Per pump (2×)**	4 Pole	**500**
1 Standby and 1 Operating set					Diesel Pump (150 L/s)	1,560	**200.84**		Fixed	
									1,500 RPM	

With the above dewatering capacities, the operating points (days to dewater) are indicated in KCB Prefeasibility dewatering report (M09759A02.730- October 2013)

Figure 16.32 to Figure 16.34 for each storm condition for the Colnic pit, as well as the Rovina pit as follows:

- Colnic Pit – Early-stage storm (100-year 72-hour) – 14 d to dewater
- Colnic Pit – Late-stage storm (100- year 72-hour) – 6 d to dewater
- Rovina Pit – 100-year 72-hour with 4 d to dewater

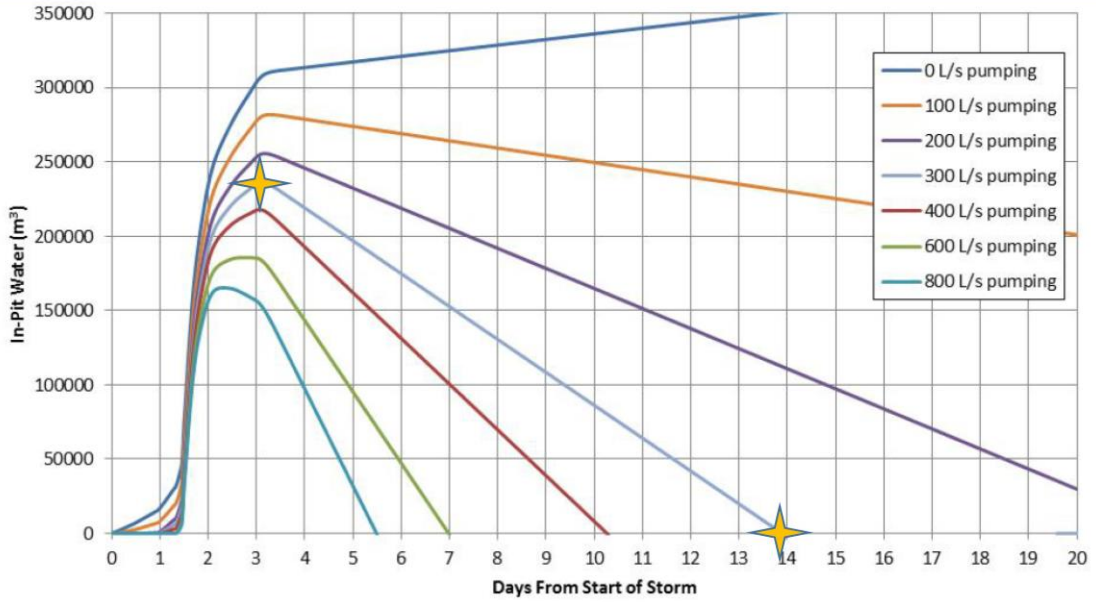


Figure 6.2 **Colnic Pit Early-Stage Storm Dewatering - Time to Dewater the 100 year 72 hour Design Storm at differing Pumping Rates.**

Source: KCB Prefeasibility dewatering report (M09759A02.730- October 2013)

Figure 16.32: Colnic Pit – Early-Stage Storm Conditions

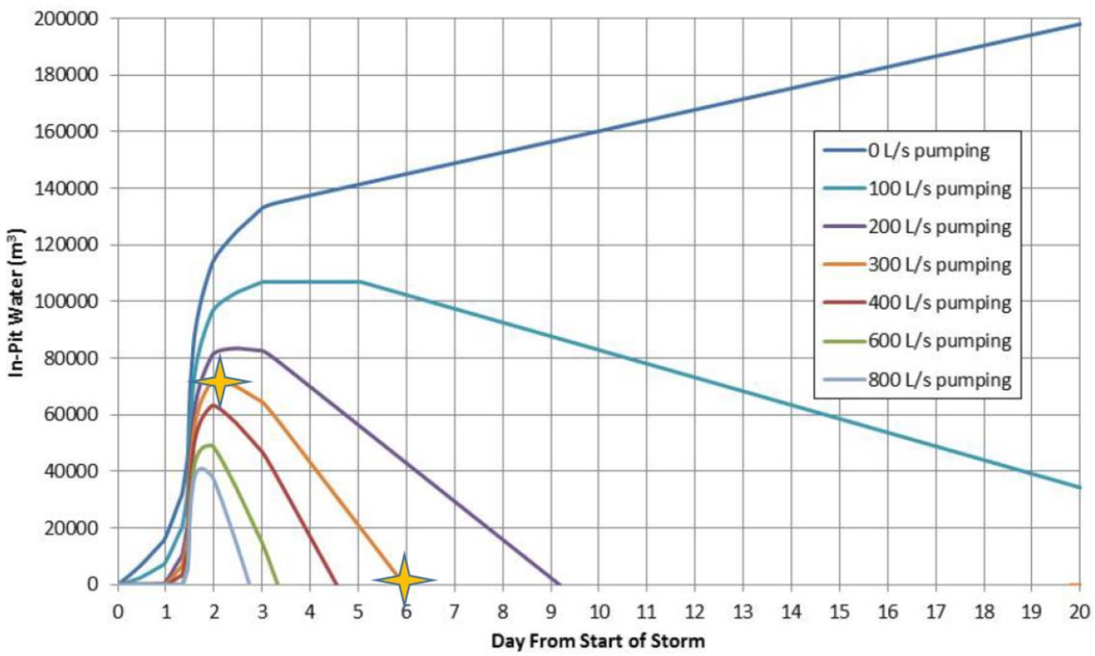


Figure 6.3 **Colnic Pit Late-Stage Storm Dewatering - Time to Dewater the 100 year 72 hour Design Storm at differing Pumping Rates.**

Source: KCB Prefeasibility dewatering report (M09759A02.730- October 2013)

Figure 16.33: Colnic Pit – Late-Stage Storm Conditions

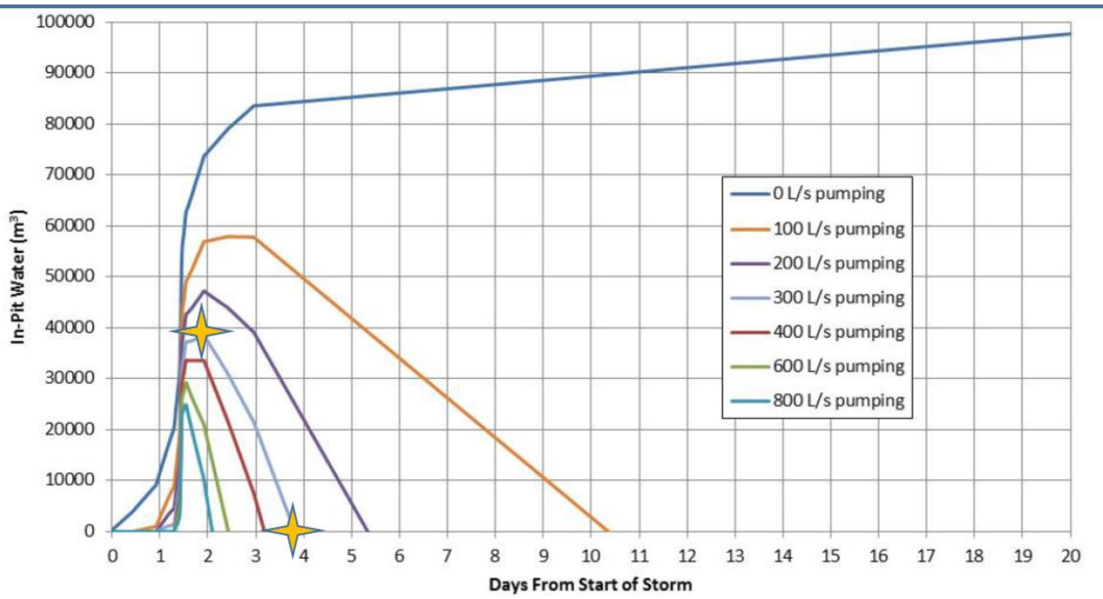


Figure 6.1 **Rovina Pit - Time to Dewater the 100 year 72 hour Design Storm at Differing Pumping Rates.**

Source: KCB Prefeasibility dewatering report (M09759A02.730- October 2013)

Figure 16.34: Rovina Pit – 100-Year 72-Hour Conditions

17 RECOVERY METHODS

17.1 PROCESS PLANT OVERVIEW

The Rovina Valley process plant design is based on well-known and established froth flotation to recover copper and gold concentrate from mined ore. The process plant will consist of a gyratory primary crushing circuit, a semi-autogenous grinding (SAG) mill in conjunction with a ball mill and pebble crushing (SABC) circuit, and conventional froth flotation as a mineral separation technique to beneficiate the sulphide copper ore.

The plant will treat 7.2 Mt/a of Colnic ore for the first seven years and Rovina ore for the next nine years of the LOM. ROM ore is crushed in a single-stage crushing circuit utilising a gyratory crusher. The crushed ore is conveyed to a stockpile and is then withdrawn using pan feeders onto the mill feed conveyor.

The milling circuit consists of a primary SAG mill and a secondary ball mill. A pebble crushing facility utilises a cone crusher to crush pebbles from the SAG mill discharge screen, and the crushed pebble material is recycled back onto the mill feed conveyor.

The discharge from both mills gravitates to the common sump and is pumped to a cyclone cluster. The cyclone underflow slurry gravitates to the secondary ball mill. The cyclone overflow slurry gravitates onto the trash removal screen, and the screen underflow gravitates to the flotation feed surge tank. The rougher scalper concentrate reports to the concentrate thickener, and the rougher-scavenger concentrate is transferred to the regrind mill for further liberation of gold and copper.

After regrinding, the concentrate is transferred to a multistage cleaner flotation circuit. The final concentrate from the last cleaner stage and rougher scalper is thickened, filtered, and bagged for sale. The flotation tails from the rougher, scavenger and cleaner flotation circuit are thickened, filtered, conveyed, and stacked at the WMF.

Facilities to mix, store and distribute reagents and consumables are allowed for in the design. The reagents and consumables include grinding media for the mills, flocculants for thickening, and flotation reagents (lime, frother, collector and promoter).

A dedicated low-pressure air system consisting of air blowers supplies blower air to the rougher-scavenger flotation cells. The low-pressure compressors supply the low-pressure air required for the cleaner flotation circuit. The compressed air for the pressure filters is supplied by separate dedicated air compressors. The instrument air requirements for the entire plant is supplied by separate dedicated high-pressure compressors in combination with the air dryers and air filters.

Raw, process, gland, potable and fire water storage facilities are provided, from which water is distributed throughout the process plant.

A simplified process flowsheet showing the major unit operations is shown in Figure 17.1, and the plant general arrangement is shown in Figure 17.2.

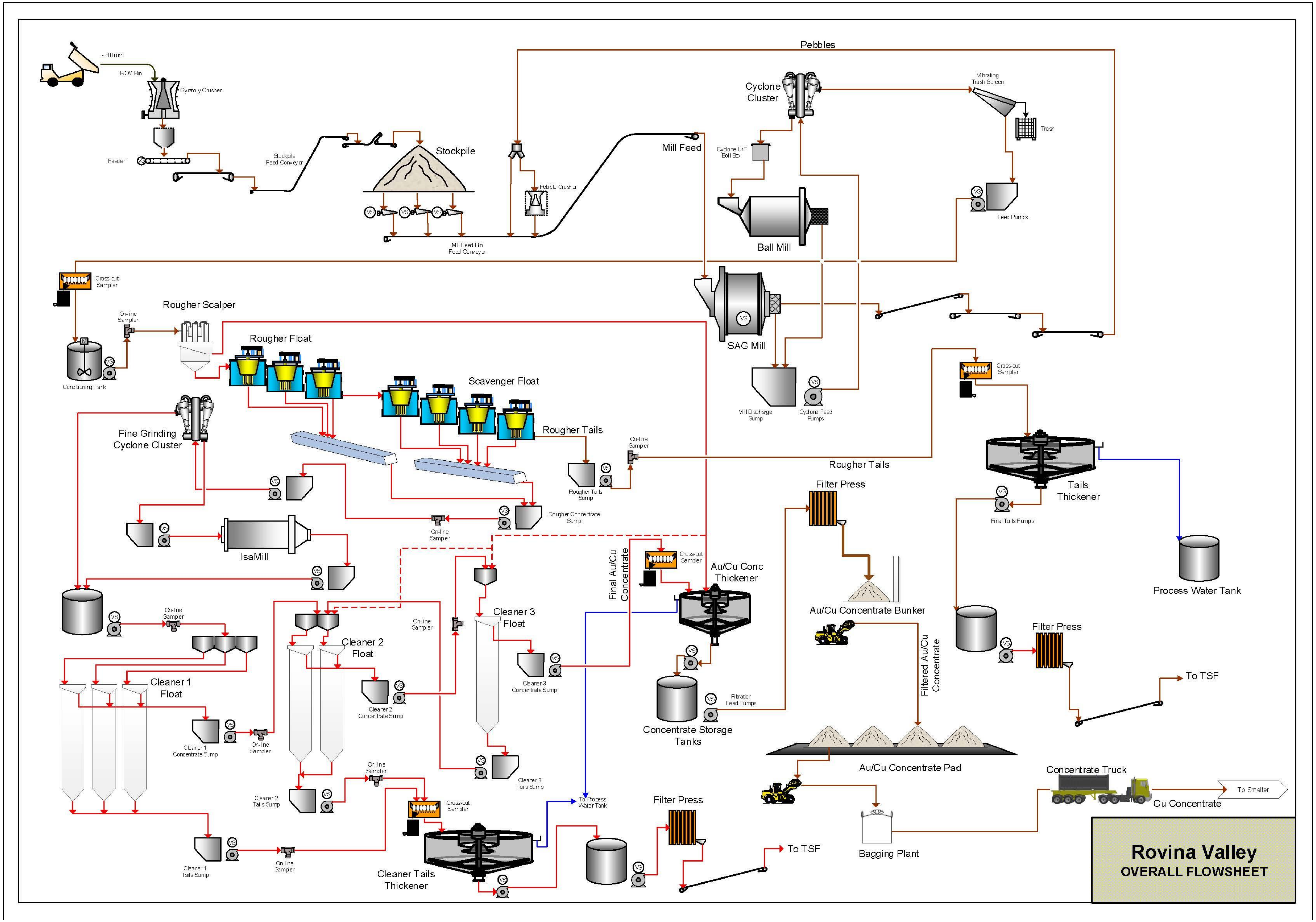


Figure 17.1: Process Flowsheet

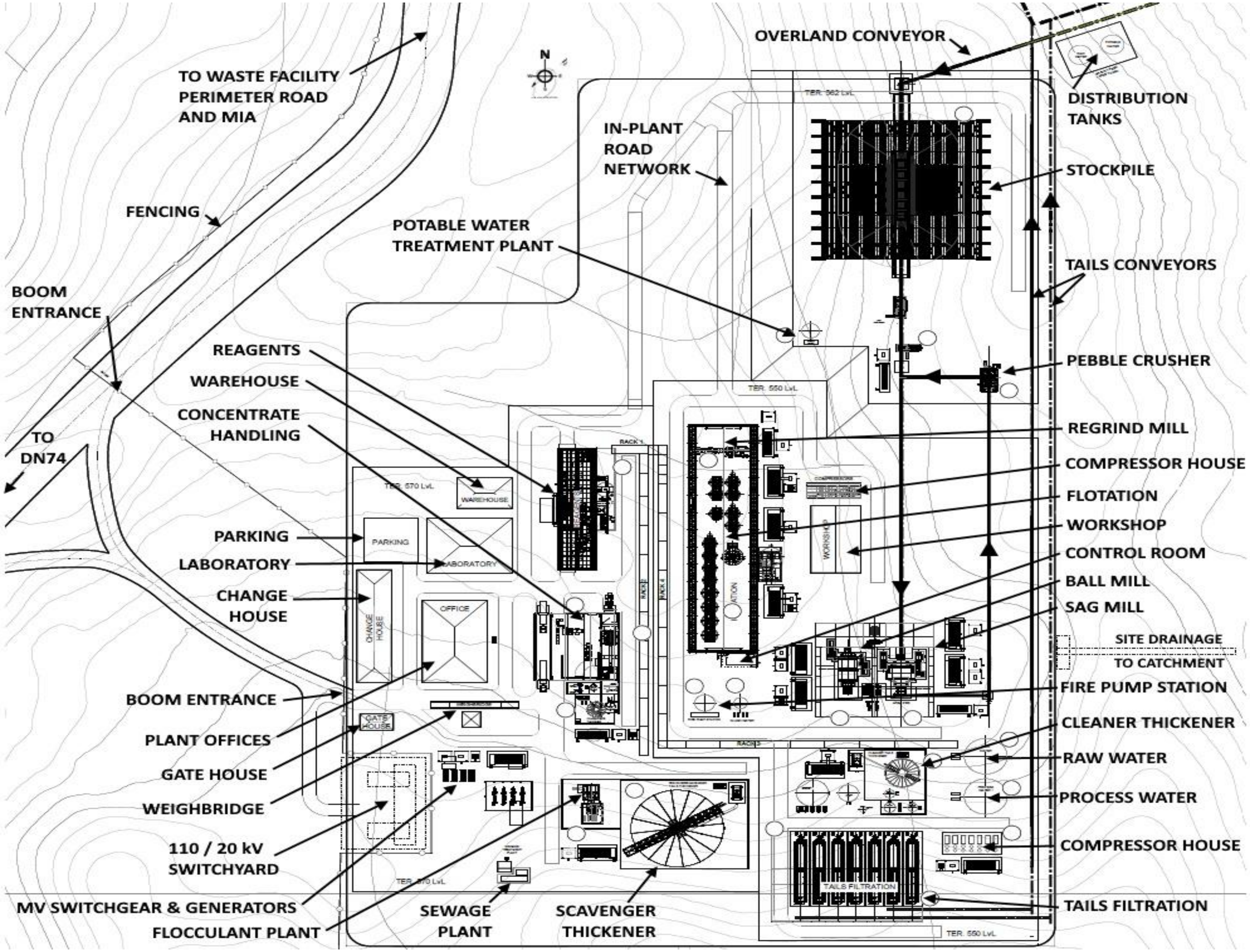


Figure 17.2: Process Plant General Arrangement

17.2 FLOWSHEET DEVELOPMENT

The Rovina Valley process plant flowsheet was developed from the interpretation of the results of various test work programmes conducted by SGS Lakefield, Eriez and Pocock laboratories on samples from the Colnic and Rovina porphyry orebodies. The flowsheet has been developed for a plant that will treat both orebodies separately and/or in a blend to optimise head grade.

In developing the process flowsheet, trade-off studies were conducted to optimise the process route selection. Specialised consultants, such as Orway Mineral Consultants (OMC) for comminution and Adroit Process Equipment (APE) for filtration, were engaged to assist with the flowsheet optimisation in the areas of their expertise. The trade-off studies and simulations conducted included the following:

- Materials handling (SENET)
- Comminution circuit (OMC)
- Flotation circuit (SENET)
- Concentrate regrind (SENET)
- Concentrate and tails filtration (SENET with APE input)

The conclusions from these trade-off studies and simulations led to the selection of the flowsheet, which includes the following processes, systems or equipment:

- Single-stage crushing facility utilising suitable primary crushing technology (such as a gyratory crusher) suitable for the high throughput Rovina Valley plant.
- A 1 km overland conventional conveying system
- SABC circuit suitable for competent Rovina Valley ores
- Rougher scalper flotation Jameson cell technology to recover finer and fast-floating particles, followed by conventional tank cell flotation to maximise both the roughing and scavenging recoveries
- Open-circuit IsaMill™ concentrate regrind technology to further liberate fine particles locked in gangue material to improve concentrate quality while minimising the generation of slimes
- Three-stage cleaning using column cell flotation technology to produce saleable quality concentrate
- Pressure filtration technology for both the final concentrate and final tails
- Concentrate filter cake bagging system
- Final tails filter cake stacking system
- Consumables and reagents
- Air and water services

Table 17.1 outlines the overall process design criteria developed for the Rovina Valley selected flowsheet/processing route.

Table 17.1: Summary of Key Process Design Criteria

Parameter	Unit	Value		Source
		Colnic Open Pit	Rovina Open Pit	
ORE CHARACTERISTICS				
Ore Source	–	Colnic Open Pit	Rovina Open Pit	Client
Ore Head Grades				
Cu Head Grade – LOM	%	0.10	0.24	DRA Mining
Au Head Grade – LOM	g/t	0.62	0.26	DRA Mining
Moisture Content	%	3.0	3.0	Client
Water Specific Gravity	–	1.00	1.00	SENET
Specific Gravity	–	2.67	2.63	Test Work
Unconfined Compressive Strength	MPa	146	–	Test Work
JK Drop Weight Index (DWI)	kWh/m^3	9.7	7.7	Test Work
Rod Mill Work Index	kWh/t	18.8	14.3	Test Work
Ball Mill Work Index	kWh/t	18.1	15.9	Test Work
Crusher Work Index	kWh/t	15.9	–	Test Work
Abrasion Index	–	0.323	0.285	Test Work
JK Tech Parameters:				
A × b	–	26	25	Test Work
t_a	–	0.25	0.24	Test Work
OPERATING SCHEDULE				
Plant				
Throughput	Mt/a	7.2	7.2	Client
Crushing				
Overall Utilisation/Availability	%	70	70	SENET/Industry
Annual Operating Hours	h	6,171	6,171	Calculated
Milling				
Overall Utilisation/Availability	%	91	91	SENET/OMC
Annual Operating Hours	h	8,000	8,000	Calculated
PRODUCTION SCHEDULE				
Copper Production				
Overall Head Grade – LOM	%	0.10	0.24	DRA Mining
Overall Recovery – LOM	%	88.5	92.5	SENET/ Test Work
Overall Product Grade – LOM	%	20.3	22.1	SENET/ Test Work
Gold Production				
Overall Head Grade – LOM	g/t	0.62	0.26	DRA Mining
Overall Recovery – LOM	%	80.0	78.8	SENET/ Test Work
Product Grade – LOM	g/t	65.87	22.92	SENET/ Test Work

Parameter	Unit	Value		Source
		Colnic Open Pit	Rovina Open Pit	
COMMINUTION				
Primary Crushing				
Primary Crushing Product Size (P_{80})	mm	150	150	OMC
Milling				
Mill Circuit Product Size (P_{80})	µm	75	75	Test Work
FLOTATION				
Rougher Flotation Residence Time – Laboratory	min	5.8	5.8	Test Work
Scavenger Flotation Residence Time – Laboratory	min	5.4	5.4	Test Work
Concentrate Regrind Product Size (P_{80})	µm	13.5	13.5	Test Work
Cleaner 1 Flotation Residence Time – Laboratory	min	10.6	10.6	Test Work
Cleaner 2 Flotation Residence Time – Laboratory	min	6.2	6.2	Test Work
Cleaner 3 Flotation Residence Time – Laboratory	min	6.2	6.2	Test Work
SCAVENGER TAILS THICKENING				
Type of Thickener	–	High Rate		SENET/Client
Specific Settling Rate	t/(m²/h)	0.578	0.578	Test Work
Underflow Solids (m/m) – Maximum	%	55.0	55.0	Test Work
SCAVENGER TAILS FILTRATION				
Type of Filter	–	Pressure Filter		SENET/Client
Cake Moisture Content	%	14.5	14.5	Test Work
Overall Filtration Rate	kg/(m²/h)	157	157	Test Work
CLEANER TAILS THICKENING				
Type of Thickener	–	High Rate		SENET/Client
Specific Settling Rate	t/(m²/h)	0.649	0.649	Test Work
Underflow Solids (m/m)– Maximum	%	55.0	55.0	Test Work
CLEANER TAILS FILTRATION				
Type of Filter	–	Pressure Filter		SENET/Client
Cake Moisture Content	%	11.7	11.7	Test Work
Overall Filtration Rate	kg/(m²/h)	157	157	Test Work
CONCENTRATE THICKENING				
Type of Thickener	–	High Rate		SENET/Client
Specific Settling Rate	t/(m²/h)	0.649	0.649	Test Work
Underflow Solids (m/m) – Maximum	%	55.0	55.0	Test Work

Parameter	Unit	Value		Source
		Colnic Open Pit	Rovina Open Pit	
CONCENTRATE FILTRATION				
Filter Type	–	Pressure Filter		SENET/Client
Cake Moisture Content	%	8.5	8.5	Test Work
Overall Filtration Rate	kg/(m^2/h)	157	157	Test Work
REAGENTS				
Collector (PAX)	g/t ore	13	13	Test Work
Promoter (Aerophine 3418A)	g/t ore	18.1	18.1	Test Work
Frother (Aerofroth 65)	g/t ore	20.5	20.5	Test Work
Lime	g/t ore	580	580	Test Work
Available CaO	%	90	90	Industry
Flocculant	g/t ore	37	37	Test Work

17.3 PROCESS DESCRIPTION

17.3.1 Crushing and Stockpiling

Ore is delivered via haul trucks into the ROM receiving bin, which feeds directly to the primary crusher. The crusher product reports to the product bin. The primary crusher discharge apron feeder transports the crushed ore to the crusher sacrificial conveyor, which transports the crushed ore to the overland conveyor.

A belt weightometer installed on the sacrificial conveyor records the feed rate from the gyratory crusher. While on the sacrificial conveyor, the ore travels under a tramp metal magnet so that any metal present can be removed. A metal detector is situated further along to detect any metal that was not successfully removed from the crushed ore. If metal is detected, the sacrificial conveyor will stop.

The overland conveyor conveys the crushed ore onto the tripper conveyor, which discharges to the crushed ore stockpile. The crushed ore stockpile provides a surge capacity and ensures a constant feed to the milling circuit when maintenance work is being carried out under scheduled shutdown or upset conditions in the crushing section.

Dust control is very important in the crushing and mill feed storage sections. Dust control is by way of both containment and suppression. The dust suppression system uses fine water sprays at the main dust-generating points in the crushing, stockpile and mill feed sections.

Electric hoists are included in the crushing section to facilitate maintenance.

17.3.2 Stockpile Reclaim and Milling

The mill feed stockpile is fitted with six vibrating feeders. The feeders are equipped with variable-speed drives for better mill feed control. The feeders regulate the rate of ore withdrawal from the stockpile and, consequently, the SAG mill feed rate. The feeders withdraw material from the stockpile onto the mill feed conveyor, which conveys the material to the SAG mill.

The milling circuit is an SABC configuration, consisting of a SAG mill, ball mill and pebble crushing circuit. The SAG mill is equipped with a variable-speed drive, which allows the mill to treat ores with varying work indices by changing the mill energy input and protects the mill shell liners in the event of low-load conditions. The SAG mill slurry discharges through a trommel screen, and the screen oversize is conveyed to the pebble crushing circuit for further size reduction, from where it is directed onto the mill feed conveyor.

Overband magnets on the pebble crusher feed conveyors pick up any steel balls discharged together with the pebbles. A metal detector on the pebble conveyor detects any metal not successfully removed by the tramp metal magnet. A bypass system is incorporated in the design before the pebble crusher feed bin to bypass the pebble crusher and divert the material to the SAG mill conveyor when the pebble crusher is not available due to maintenance.

The SAG mill discharge trommel screen undersize is combined with the ball mill discharge in a common sump and then pumped to a hydrocyclone cluster for classification. The hydrocyclone overflow advances to the flotation circuit via the trash screen, and the hydrocyclone underflow gravitates to feed the ball mill. A trommel screen is installed at the discharge end of the ball mill for the removal of scats. The scats are collected in the scats bunker periodically using a front-end loader (FEL).

Process water is fed at a ratio to the SAG mill feed tonnage to obtain the required mill discharge density and is also added at a controlled rate to the mill discharge sump to adjust the set cyclone feed density. The balance of the process water to the circuit is used for ball mill dilution and high-pressure spray water on the mill discharge screens to ensure efficient wet screening.

Milling and cyclone spillage is contained by a concrete bund area with a sloped-end floor to direct spillage to the mill feed and mill discharge end spillage sumps. Each sump has a vertical spindle pump that pumps the spillage to the mill discharge sump.

17.3.3 Flotation Feed Preparation

The cyclone overflow reports to a trash removal vibrating screen. The trash screen removes any woodchips, plastics, oversize material, and other foreign material from the flotation feed slurry. The trash screen oversize reports to a concrete trash bunker. The trash screen undersize gravitates to the flotation section and is sampled in an automated sampler, from which it flows to a rougher flotation conditioning tank. The automatic sampling system consists of a feed box, launder, and a cross-cut and vezin sampler, and takes the flotation feed sample for metallurgical accounting purposes at regular intervals.

The slurry in the conditioning tank is pre-treated with the flotation reagents (collector and promoter). Hydrated lime slurry is also added to the conditioning tank to adjust the pH for pyrite depression.

17.3.4 Flotation Circuit

The flotation circuit consists of a rougher scalper flotation Jameson cell, rougher and scavenger flotation tank cells, concentrate regrind, three-stage cleaner flotation column cells, and an on-line stream analyser. The entire flotation and regrind section of the plant is housed under roof and also under crane to allow for ease of maintenance of the flotation equipment.

17.3.5 Rougher Scalper Flotation

The rougher scalper flotation Jameson cell receives feed at a controlled flow and density from the conditioning tank. Frother is added to the cell together with air entrained from the atmosphere. Concentrate from the rougher scalper gravitates to the sump, from where the concentrate is pumped to the final concentrate thickener via a final concentrate sampling system. Tails from the rougher scalper gravitate to the sump, from where the tails are pumped to the rougher-scavenger flotation cells.

17.3.6 Rougher-Scavenger Flotation

The rougher-scavenger flotation cells receive feed at a controlled flow and density from the rougher scalper tails or from the conditioner tank (when bypassing the scalper cell). Collector and frother reagents are added to the flotation cells together with the blower air. Concentrate from the rougher-scavenger cells gravitates to the sump, from where the concentrate is pumped to the regrind mill circuit. Tails from the rougher-scavenger flotation cells gravitate to the sump, from where the tails are pumped to the tails thickener via the automatic flotation tails sampling system.

17.3.7 Concentrate Regrind Mill Circuit

The rougher-scavenger flotation concentrate reports to the regrind mill cyclone feed sump. The concentrate slurry from the sump is pumped to the regrind cyclone for classification. The cyclone underflow gravitates to the regrind mill feed sump, and the cyclone overflow slurry is directed to the cleaner flotation feed conditioning tank.

The regrind mill operates in an open circuit, and the mill product gravitates to the regrind mill discharge sump, from where it is pumped to the cleaner flotation feed conditioning tank.

17.3.8 Cleaner Flotation Feed

The regrind mill product, together with the cyclone overflow, reports into the agitated cleaner flotation feed conditioning tank.

The slurry in the conditioning tank is pre-treated with the flotation reagents (collector and promoter). Hydrated lime slurry is also added to the conditioning tank to adjust the pH for pyrite depression.

17.3.9 Cleaner 1 Flotation

The Cleaner 1 flotation column cells receive feed at a controlled flow and density from the conditioning tank. Frother is added to the cells together with sparger air from the low-pressure compressors. Concentrate from the Cleaner 1 flotation cells gravitates to the sump, from where the concentrate is pumped to the Cleaner 2 flotation feed. Tails from the Cleaner 1 flotation cells gravitate to the sump, from where the tails are pumped to the cleaner tails thickener via the automated sampling system.

17.3.10 Cleaner 2 Flotation

The Cleaner 2 flotation column cells receive feed at a controlled flow from the Cleaner 1 concentrate tank. Frother is added to the cells together with sparger air from the low-pressure compressors. Concentrate from the Cleaner 2 flotation cells gravitates to the sump, from where the concentrate is pumped to the Cleaner 3 flotation feed. Tails from the Cleaner 2 flotation cells gravitate to the sump, from where the tails are pumped to the Cleaner 1 flotation feed.

17.3.11 Cleaner 3 Flotation

The Cleaner 3 flotation column cell receives feed at a controlled flow and density from the Cleaner 2 concentrate tank. Frother is added to the cells together with sparger air from the low-pressure compressors. Concentrate from the Cleaner 3 flotation cells gravitates to the sump, from where the concentrate is pumped to the final concentrate thickener via the final concentrate sampling system. Tails from the Cleaner 3 flotation cells gravitate to the sump, from where the tails are pumped to the Cleaner 2 flotation feed.

17.3.12 On-Line Analyser

An on-line analyser is used to analyse critical streams for process control. The analyser receives sample streams from the following points:

- Scalper rougher flotation feed
- Scalper rougher concentrate
- Rougher-scavenger flotation feed
- Rougher-scavenger flotation tails
- Cleaner 1 concentrate
- Cleaner 1 tails
- Cleaner 2 concentrate
- Cleaner 2 tails
- Cleaner 3 concentrate
- Cleaner 3 tails

The analyser analyses each stream for the following:

- g/t Au
- g/t Ag
- % Cu
- % Zn
- % Fe
- % S
- % solids

From this information, the grades and recoveries are derived. All the information from the analyser is displayed on the operator interface screen.

17.3.13 Concentrate Thickening

The final flotation concentrate from the scalper rougher flotation cell and Cleaner 3 flotation cells is pumped to the thickener for dewatering prior to filtration. The thickener feed flows through an automatic sampling system consisting of a stilling box, launder, and cross-cut and vezin samplers, where the final flotation concentrate metallurgical accounting sample is taken at regular intervals. Rejects from the secondary sampler are directed to the concentrate thickener area sump. The rest of the stream slurry goes through the primary sampler launder and gravitates to the concentrate thickener feed box. Diluted flocculant is added to the thickener feedwell to aid solid-liquid separation in the thickener. Thickener overflow is pumped to the process water tanks, and the underflow is pumped to the concentrate filter feed tank.

17.3.14 Concentrate Filtration and Bagging

The slurry from the concentrate thickener underflow is pumped to the filter feed tank for surge storage capacity. The concentrate slurry is pumped into the concentrate filter, and the filtrate is pumped back to the concentrate thickener. The concentrate is collected between the cloth surfaces of the filter and forms the filter cake. After a complete filter cycle, the concentrate filter cake is discharged onto the bund floor where it is picked up using an FEL and transported to the load-out system for bagging into bulk bags.

17.3.15 Scavenger Tails Thickening

The flotation tails from the rougher-scavenger flotation cells are pumped to the thickener for dewatering prior to filtration. The thickener feed flows through an automatic sampling system consisting of a stilling box, launder, and cross-cut and vezin samplers, where the metallurgical accounting sample is taken at regular intervals. Rejects from the secondary sampler are directed to the thickener area sump. The rest of the stream slurry goes through the primary sampler launder and gravitates to the thickener feed box. Diluted flocculant is added to the thickener feedwell to aid solid-liquid separation in the thickener. Thickener overflow is pumped to the process water tanks, and the underflow is pumped to the tails filter feed tank.

17.3.16 Scavenger Tails Filtration

The slurry from the scavenger tails thickener underflow is pumped to the scavenger tails filter feed tanks for surge storage capacity. The tails slurry is pumped into the tails filters, and the filtrate is pumped back to the scavenger tails thickener. The tails are collected between the cloth surfaces of the filter and form the filter cake. After a complete filter cycle, the tails filter cake is discharged onto intermediate conveyors, from where the filter cake is transported via tails disposal conveyors to the tailings management facility.

17.3.17 Cleaner Tails Thickening

The flotation tails from the Cleaner1 flotation are pumped to the cleaner tails thickener for dewatering prior to filtration. The thickener feed flows through an automatic sampling system consisting of a stilling box, launder, and cross-cut and vezin samplers, where the metallurgical accounting sample is taken at regular intervals. Rejects from the secondary sampler are directed to the thickener area sump. The rest of the stream slurry goes through the primary sampler launder and gravitates to the thickener feed box. Diluted flocculant is added to the thickener feedwell to aid solid-liquid separation in the thickener. Thickener overflow is pumped to the process water tanks, and the underflow is pumped to the cleaner tails filter feed tank.

17.3.18 Cleaner Tails Filtration

The slurry from the cleaner tails thickener underflow is pumped to the cleaner tails filter feed tank for surge storage capacity. The tails slurry is pumped into the tails filter, and the filtrate is pumped back to the cleaner tails thickener. The tails are collected between the cloth surfaces of the filter and form the filter cake. After a complete filter cycle, the tails filter cake is discharged onto intermediate conveyors, from where the filter cake is transported via tails disposal conveyors to the tailings management facility.

Cleaner tails are stored separately from the scavenger tails at the waste management facility (WM) due to the high pyrite content with acid-producing characteristics.

17.3.19 Reagents

Facilities to mix, store and distribute reagents and consumables are allowed for in the design. These reagents and consumables include lime, flocculant, collector (potassium amyl xanthate – PAX), promoter, frother, and grinding media. In addition, potassium permanganate and sodium hypochlorite will also be used in the potable water treatment facility.

The reagent consumptions obtained during bench scale and pilot laboratory tests were used to estimate the size of the equipment associated with mixing, storage and distribution of most of the reagents. The reagent consumptions used for the water treatment plant were obtained from the selected water treatment plant vendor.

As reagents are generally classified as a safety risk, safety showers will be incorporated into the design.

17.3.19.1 Hydrated Lime

Quicklime is supplied in a bulk road tanker and slaked at the plant to hydrated lime. The lime section consists of a dry receiving and storage area, make-up area, wet storage area and dosing area. The slaking system and dosing facility are designed taking into account the total hydrated lime usage in the flotation circuits. Hydrated lime is added to the roughers and cleaners to maintain the slurry in the circuit at pH 9.5 to 11. The design allows for two make-ups per day and a 48 h storage capacity for the silo and a 24 h storage capacity for the dosing tank.

17.3.19.2 Flocculant – SNF AN905 SH

Flocculant is delivered to site in bulk bags. The flocculant section consists of a dry receiving and storage area, make-up area, wet storage area and dosing area. The flocculant make-up system and dosing facility are designed taking into account the total usage in the thickening circuits. Flocculant is added to the concentrate, scavenger tails and cleaner tails thickeners to aid solid-liquid separation. The design allows for two make-ups per day and a 24 h storage capacity for the dosing tank.

17.3.19.3 Collector - PAX

The collector (PAX) is delivered to site in bulk bags. The collector section consists of a dry receiving and storage area, make-up area, wet storage area and dosing area. The collector is added to the rougher scalper flotation cells, rougher-scavenger flotation cells and to the multistage cleaner flotation circuit. The collector make-up and dosing facility is designed for one make-up per day and a 24 h storage capacity for the dosing tank.

17.3.19.4 Frother – Aerofroth 65 (A-65)

Frother is delivered to site in intermediate bulk containers (IBCs). The frother section consists of an IBC receiving and storage area, and a frother decanting and dosing area. The frother decanting and dosing area is designed for one IBC top-up per day and a 24 h storage capacity for the dosing tank.

17.3.19.5 Promoter – Aerophine 3418A

Promoter is delivered to site in IBCs. The promoter section consists of an IBC receiving and storage area, and a promoter decanting and dosing area. The promoter decanting and dosing area is designed for one IBC top-up per day and a 24 h storage capacity for the dosing tank.

17.3.20 Air Services

The plant air high-pressure compressors (one duty and one standby) supply the plant air required for the plant. The air from the compressors for plant distribution is filtered through a pair of air filters before it is stored in the filtered air receiver.

Instrument air is passed through a pair of air filters and the instrument air dryer. Dried instrument air is filtered again through a pair of air filters before it is stored in the instrument air receiver. The instrument air receiver distributes the instrument air to all the air-operated instruments throughout the plant.

The low-pressure compressors (one duty and one standby) supply the low-pressure air required for the multistage cleaner flotation circuit.

The low-pressure blowers (one duty and one standby) supply the low-pressure blower air required for the rougher and scavenger flotation circuit.

17.3.21 Water Services

17.3.21.1 Process Water Distribution

The sources of process water are the concentrate thickener overflow, scavenger tailings thickener overflow, cleaner tailings thickener overflow, filtrate from the tailings filters, and the contact water catchment system. Process water is used in milling for dilution, flushing, hosing, and screen washing applications.

17.3.21.2 Raw Water Distribution

Raw water is obtained from the contact catchment system. Water from contact water catchment system passes through a set of filters before it is stored in the raw water storage tank. Raw water is used for mill cooling water, reagent make-up, flotation froth spray water, fire water, process water top-up, potable water treatment and gland seal water supply.

The fire water storage tank is included in the design to ensure that firefighting water is available to be distributed throughout the plant. An electric pump serves as the primary fire water pump with a diesel-driven pump as a standby pump to supply water to the fire water system in the event of a power outage. A jockey pump is used to maintain the fire water system pressure.

17.3.21.3 Potable Water Distribution

Raw water is supplied to the water treatment plant and treated for potable water distribution. The potable water is stored in the potable water storage tank and delivered to the potable water hydrosphere and the safety shower water hydrosphere. The hydrospheres are used to maintain the required pressure in the potable water distribution and safety shower headers.

A potable water line is installed to supply water to a header tank that is located at an elevated level, from which the potable water is distributed to the nearby community using gravity.

17.3.21.4 Gland Seal Water Distribution

Gland water is obtained from the raw water system and it is stored in a dedicated gland water distribution tank. Gland water is distributed using dedicated pumps for each system: the milling circuit and the flotation circuit and filtration circuits.

18 PROJECT INFRASTRUCTURE

18.1 PROJECT ON-SITE INFRASTRUCTURE

The RVP is a greenfield project, and as such, infrastructure has yet to be established on the project site. The on-site infrastructure is primarily related to the processing plant and its immediate supporting facilities as follows:

- In-plant access roads
- Process plant infrastructure buildings
- Process plant reagents building
- Process plant site drainage
- Process plant sewage collection and disposal
- Process plant security
- Process plant water supply and distribution
- Process plant communications and control systems
- Process plant power supply
- Filtered tailings and mine waste facility

The overall process plant general arrangement is shown in Figure 18.1.

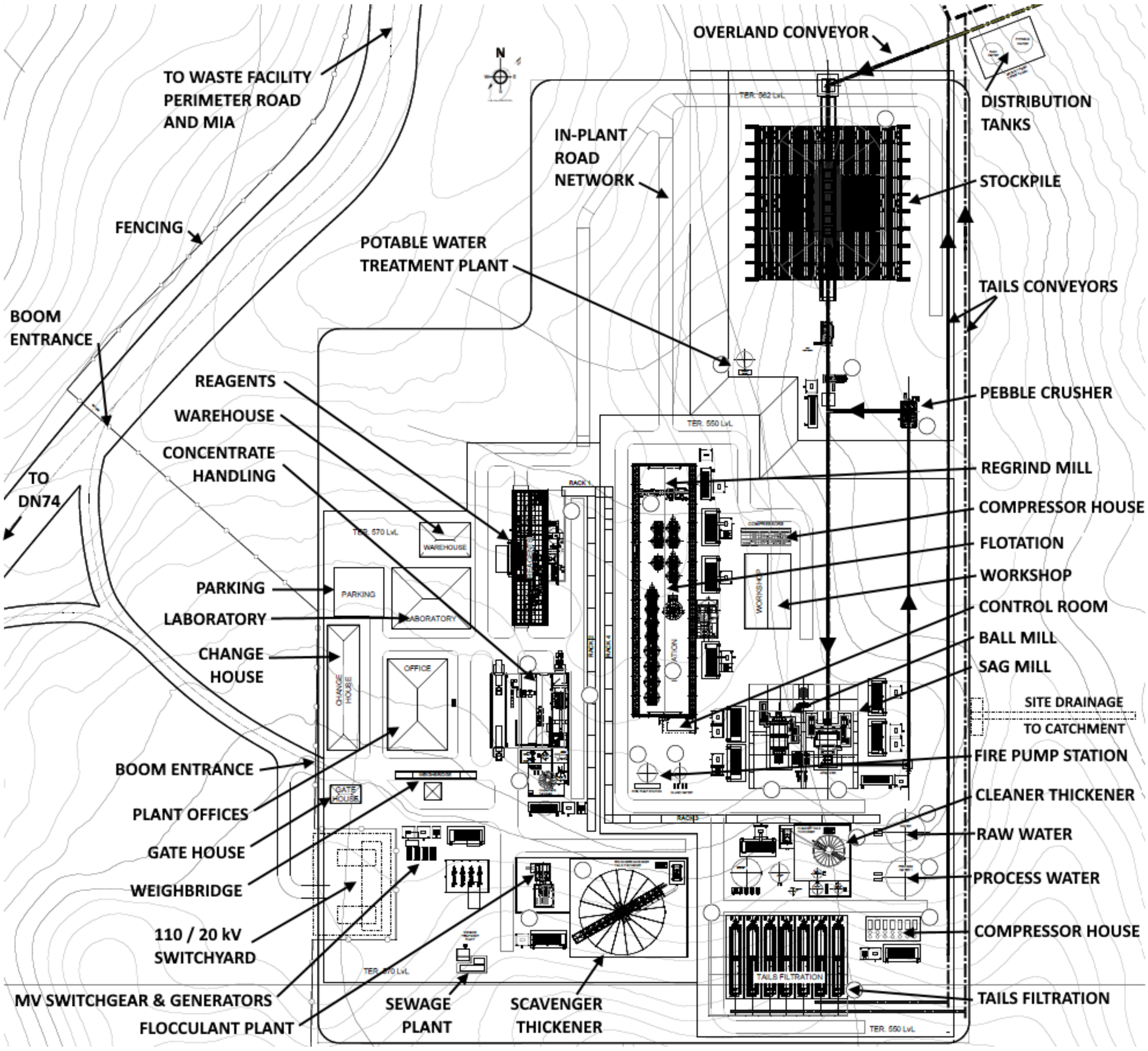


Figure 18.1: Process Plant General Arrangement

18.1.1 In-Plant Access Roads

In-plant roads will be constructed to provide unrestricted access to the following key areas within the process plant fenced perimeter:

- Gatehouse main access to process plant area
- Change house and office complex
- Reagents stores and lime storage silo
- Workshops and warehouse
- Plant metallurgical/assay laboratory
- Stockpile and pebble crushing circuit
- Ball and SAG mills building
- Flotation and compressor buildings
- Rougher-scavenger tails thickener and flocculant plant
- Cleaner tails thickener and tails filtration plant

The in-plant access road network construction comprises ripping and recompacting to a depth of 200 mm; thereafter, backfilling with suitable borrow material and finally compacting to 95 % MOD AASHTO, which means that in-situ soil will be compacted to 95 % of the maximum dry density by means of different kinds of rollers (depending on the soil characteristics). The Proctor compaction test will be performed to find out the maximum dry density and optimum moisture content of a soil.

The process plant main access road features a diversion road to the south, directly before entering the process plant area. This road leads to a dedicated and fenced-off municipal electrical mains power incoming supply switchyard, ensuring restricted access only to the power utility company personnel to perform regulatory operation and maintenance activities.

The in-plant road network is shown in Figure 18.1.

18.1.2 Process Plant Infrastructure Buildings

The process plant infrastructure buildings will consist of the following:

- Gatehouse
- Change house
- Control room
- Workshop
- Office building
- Metallurgical/assay laboratory
- Warehouse
- Weighbridge control room

The position of the process plant infrastructure buildings is shown in Figure 18.1.

All prefabricated infrastructure buildings will be completely furnished and, where applicable, feature adequate fit-for-purpose lighting, power socket outlets, data and communication trunking, hot/cold water plumbing systems, heating, ventilation, extraction, and air-conditioning systems.

All prefabricated structures will be founded on mesh reinforced concrete slabs, complete with surrounding concrete aprons leading to culverts to facilitate proper storm water management.

18.1.2.1 Process Plant Gatehouse

A prefabricated, insulated panel gatehouse of area 122 m^2 (14.2 m × 8.5 m) will be constructed at the main access gate to the process plant and includes a security supervisors' office, complete with observation windows and a fitted desk arrangement. The building also includes a small kitchenette, male/female ablution facilities, security office, and a waiting room, complete with turnstile-controlled access.

All workers and visitors will enter the plant through the reception area. Authorised personnel can then proceed to the change house via the access-controlled turnstiles. Personnel and visitors' access will be monitored from the fully furnished security office, which contains an access control and closed-circuit television (CCTV) system.

The process plant gatehouse will be the central hub for control and monitoring of all security, fire, and emergency systems.

18.1.2.2 Process Plant Change House

A prefabricated, insulated panel change house building of area 400 m^2 (40 m × 10 m) will be constructed near the main access to the plant and will include a change house, laundry, first-aid room, and dining/conference room. The prefabricated structure will be founded on a mesh reinforced concrete slab. A 2 m concrete apron slab will form a walkway around the building.

The change house will contain both male and female change areas consisting of lockers and benches for the process plant staff to store their clothes when they arrive for their shift. At the end of the shift, the staff will return to the change area, remove their dirty work clothes (which will be washed in the adjacent laundry), shower, and then proceed through the change area, where they will retrieve their own clothes.

The process plant change house has been sized to cater for a maximum of 134 workers taking office staff, hourly shift staff, R&R staff, plant shutdowns, third-party workers, and visitors into account.

The male change house area has been sized as follows:

- 104 double lockers in the change area (1 locker per person)
- 20 showers (1 shower per 5 people maximum)
- 10 toilets (1 toilet per 10 people maximum)
- 10 wash-hand basins (1 wash-hand basin per 10 people maximum)
- 10 urinals (1 urinal per 10 people maximum)

The female change area has been sized as follows:

- 30 double lockers in the change area (1 locker per person)
- 6 showers (1 shower per 5 people maximum)
- 3 toilets (1 toilet per 10 people maximum)
- 3 wash-hand basins (1 wash-hand basin per 10 people maximum)

A fully equipped laundry room (5 m × 5 m), where all the overalls from the shifts will be washed, will be provided adjacent to the change house.

A fully furnished first-aid room (5 m × 5 m) will be provided for the treatment of minor injuries sustained within the process plant or as a holding/waiting room before transportation to the nearest hospital from the process plant site.

A fully furnished canteen/training room (10 m × 7 m), designed to accommodate 70 personnel, will be provided. A section of the room can be partitioned off to form a conference/training room as required. The canteen will be utilised as an area where staff can partake of their self-provided meals and refreshments.

18.1.2.3 Process Plant Control Room

A dedicated prefabricated plant control room building will be located alongside the flotation building and directly adjacent to the ball and SAG mills building. The control room building will be insulated and air conditioned and will house the supervisory control and data acquisition (SCADA) system, complete with an uninterruptible power supply (UPS) system.

The control room building total floor area will be 102 m² and comprise two floors.

The ground floor (51 m²) will feature a small kitchenette, single male and female ablutions, and an engineering room fitted with desks/workspace arrangements.

The first floor (51 m²) will house the actual control room, complete with observation windows and fitted desk arrangements including housing the UPS units.

18.1.2.4 Process Plant Workshop

A workshop will be established to enable maintenance and repair of the plant equipment and is located on the east side of the process plant, adjacent to the flotation building.

The workshop will consist of a steel pre-engineered portal frame building enclosed by Chromadek roof and side-wall sheeting. The steel columns will be founded on reinforced concrete plinths and spread footings, and the workshop floor will consist of a mesh reinforced concrete slab.

The workshop floor area will be 450 m² (30 m × 15 m), which will be split into a mechanical repair shop of 300 m² and an electrical repair shop of 150 m², both of which will be serviced by a single 5 t overhead gantry crane.

A two-storey brickwork annex will be provided adjacent to the workshop floor area within the steel portal frame structure. An instrument workshop, tool store, male and female ablutions, and crib room will be provided on the ground floor, and four fully furnished offices for plant maintenance staff and a 12 m × 5 m store will be provided on the first floor.

A 30 m × 6 m storage area will be available adjacent to the two-storey brickwork annex under the workshop roof and will be enclosed by security fencing.

The appropriate plant workshop tools will be provided for the workshop.

18.1.2.5 Process Plant Office Building

A fully furnished, prefabricated, Chromadek insulated panel construction plant office building will be provided for the process plant management and accounts personnel.

The plant office building will have a total floor area of 634 m² and will be located on the west side of the process plant. The plant office building will comprise the following:

- 110 m² accounts open-plan office
- 125 m² procurement open-plan office
- 18 m² safety manager's office
- 12 m² safety supervisors' office
- 24 m² general manager's office

- 12.8 m² process plant manager's office
- 20 m² human resources manager's office
- 12.5 m² environmental manager's office
- 12.5 m² metallurgical manager's office
- 12.5 m² security manager's office
- 2 × 9 m² visitors' offices
- 9 m² logistics clerk's office
- 23 m² meeting/boardroom
- 10 m² reception office
- 9 m² strongroom
- 5 m² printing room
- 10 m² kitchenette
- Male and female ablutions

18.1.2.6 Plant and Mining Metallurgical/Assay Laboratory

A plant and mining metallurgical/assay laboratory will be provided. A prefabricated, Chromadek insulated panel laboratory building of area 667 m² (31.4 m × 21 m) will be constructed near the main access to the plant and will comprise the following:

- 29 m² laboratory manager's office
- 47 m² wet laboratory area
- 26 m² environmental laboratory area
- 67 m² instrument room no. 1
- 61 m² instrument room no. 2
- 27 m² metallurgical laboratory area
- 6 m² server room
- 13 m² weigh room
- 5 m² high-grade (HG) preparation room
- 5 m² low-grade (LG) preparation room
- 143 m² sample preparation room
- 79 m² fire assay area
- Male and female ablutions

The prefabricated structure will be founded on a mesh reinforced concrete slab. A 2 m concrete apron slab will form a walkway around the building.

A steel structure of floor area 105 m² with roof sheeting will also be erected on a mesh reinforced concrete slab directly alongside the laboratory prefabricated building and will house the laboratory peripheral equipment, i.e. dust extraction and collector units, complete with exhaust fan, inline dampers and silencers, compressors, dryer, vacuum pumps and filters, ovens, lump breakers, acetylene and oxygen bottles, geysers and air conditioner units.

The laboratory will house the necessary metallurgical and assay test equipment, complete with shelving, and will be furnished to suit.

18.1.2.7 Process Plant Warehouse

A 330 m^2 enclosed steel and cladded structure warehouse will be provided west of the process plant for the purpose of storing spares and equipment for the process plant. Suitable shelving and racking will be incorporated in this building to store and manage the stored items.

An unfenced laydown area is located adjacent to the plant warehouse.

The warehouse includes a brickwork stores dispatch room, with a service counter and waiting area (5.3 m × 4 m), and a warehouse management office (4.3 m × 2.4 m).

18.1.2.8 Process Plant Weighbridge Control Room

A prefabricated, insulated panel weighbridge control room of area 13 m^2 (3.6 m × 3.5 m) will be provided directly opposite the weighbridge at the main access gate to the process plant and includes a furnished office, complete with observation windows and a fitted desk arrangement.

The prefabricated structure will be founded on a mesh reinforced concrete slab.

18.1.3 Process Plant Reagents Building

The process plant reagents building location is shown in Figure 18.1.

A reagents building of area 945 m^2 (63 m × 15 m) that comprises a steel-clad structure and concrete floor slab, complete with drainage and spillage handling facilities, will be provided.

The reagents building will provide storage and mixing facilities for the reagents as follows:

- PAX Collector
  - PAX collector storage capacity = 50 × 1 t bulk bags
  - PAX collector mixing tank
  - PAX collector dosing tank
- Promoter
  - Promoter storage capacity = 30 × 1 m^3 IBCs
  - Promoter mixing tank
  - Promoter dosing tank
- Frother (Demarcated Area – Flammable)
  - Frother storage capacity = 30 × 1 m^3 IBCs
  - Frother mixing tank
  - Frother dosing tank
- Lime
  - Lime storage capacity = 80 m^3 silo (located alongside the reagents building)
  - Lime mixing tank
  - Lime dosing tank

- Flocculant
  - Flocculant storage capacity = 60 × 1 t bulk bags
  - Flocculant make-up plant located alongside the rougher-scavenger thickener

A concrete bunded area will also be provided directly alongside the reagents building where the 80 m^3 lime silo arrangement will be constructed to facilitate the offloading of lime by means of bulk carriers.

A suitable concrete drive-through roadway will be provided within the reagents building to facilitate the offloading of reagents by means of a 5 t overhead crane and placement into the respective reagents' store areas, and movement from the store area(s) to the reagents make-up area(s) .

The reagents building will be equipped with the following fire protection systems:

- Sprinkler system
- Foam monitors with foam induction nozzles

18.1.4 Process Plant Site Drainage

The process plant terraced areas will be constructed with berms and side drainage as required to ensure that any storm or dirty water runoff not contained in the bunded areas and returned to the process storage tank is diverted via a series of dedicated drainage channels and then routed via an overland pipe network and finally discharged into the main contact (dirty) water catchment pond located north of the Colnic pit.

18.1.4.1 Storm Water Berms

Storm water cut-off berms will be constructed to prevent storm water from entering lower lying areas from areas with a higher elevation. The berms will be constructed using the material from the bulk excavations when the bulk earthworks are carried out.

18.1.4.2 Surface Drainage Side Drains

Surface drainage of the plant area will be achieved by using side drains. The surface of the plant will be sloped to allow the water to flow freely away from the plant. The plant roads will be built to have a single cross fall of between 2 % and 4 % in the direction of the side drain.

18.1.5 Process Plant Infrastructure Sewage Collection and Disposal

A 21 m^3/d containerised biological sewage treatment plant will be provided southwest of the main process plant terrace for the treatment and disposal of domestic raw sewage and change house wastewater that is generated by and collected from the various infrastructure buildings. Fats, oils, and greases will be treated with fat traps at the source, as required, and will not exceed 10 mg/L before entering the sewage plant.

Sewage reticulation piping and manholes will be provided to facilitate the flow of sewage under gravity to a collection manhole located adjacent to the sewage treatment plant. The sewage will be pumped via a submersible pump into the containerised treatment plant.

The technology selected is compact, simple, and robust, and is based on a standard activated sludge system, whereby the septic tank is integrated into the container, enabling settable suspended solids to be settled and the clarifying zone biological treatment chamber microorganisms to perform degradation of the organic load, complete with submerged aerated filters. The biochemical oxygen demand is broken down using air and bacteria that grow in this medium.

This system provides optimised nitrification and an effluent quality to a standard that complies with the requirements and general limits as stipulated in the Romanian waste management law; Law No. 211/2011.

The treated wastewater will also be disinfected in the integrated ultra-violet-system, enabling the safe release into the environment and reuse for irrigation.

18.1.6 Process Plant Security

The plant site will be fully enclosed by a 2.4 m high mesh fence to keep out range animals and unauthorised people. Access to the plant site will be restricted to one access point at the main gate, which will be equipped with a gatehouse that is manned 24 h/d. Other emergency and maintenance access gates will be provided but will be kept locked at all times.

Booms to control vehicle access will be provided at the entry gate to the process plant and at the entrance to the waste facility perimeter road leading to the MIA.

Furthermore, the plant will be fitted with CCTV cameras installed at strategic locations. The cameras will be integrated with the plant's overall network, and dial-up into these cameras via the Internet will be enabled. Views from the cameras will be fed to the central security control room situated in the gatehouse.

18.1.7 Process Plant Water Supply and Distribution

To ensure an uninterrupted supply of water to the process plant and peripherals, the process and raw water will be distributed within the process plant area and to the off-site infrastructure via the two 5,000 m^3 on-site process and raw water storage tanks, respectively. Refer to Figure 18.2.

A water balance was completed for the plant. It incorporated the interaction of the various flows required for the process plant and the water availability from the main source, the contact water catchment pond. This water balance was used as the basis for sizing the water storage vessels at the process plant. Water stored in the contact water catchment dam will be pumped to the process plant for make-up operations and filtered for raw water distribution.

The on-site process and raw water storage tanks will be fed from the following sources during process plant operation:

- The recovered inner-plant tailings filtration and thickener water return systems (1,912 m^3/h design flow rate), pumped directly to the 5,000 m^3 on-site process water storage tank only
- The contact water catchment dam, which is fed from the surrounding hills storm water and the waste facility underdrainage, including feed from the respective pit

dewatering facilities, pumped to the two 5,000 m^3 on-site process and raw water storage tanks respectively for further distribution and/or treatment with respect to the potable water requirement. Excess water from the contact water catchment dam will be treated at the wastewater treatment plant (WWTP) and then discharged to the environment.

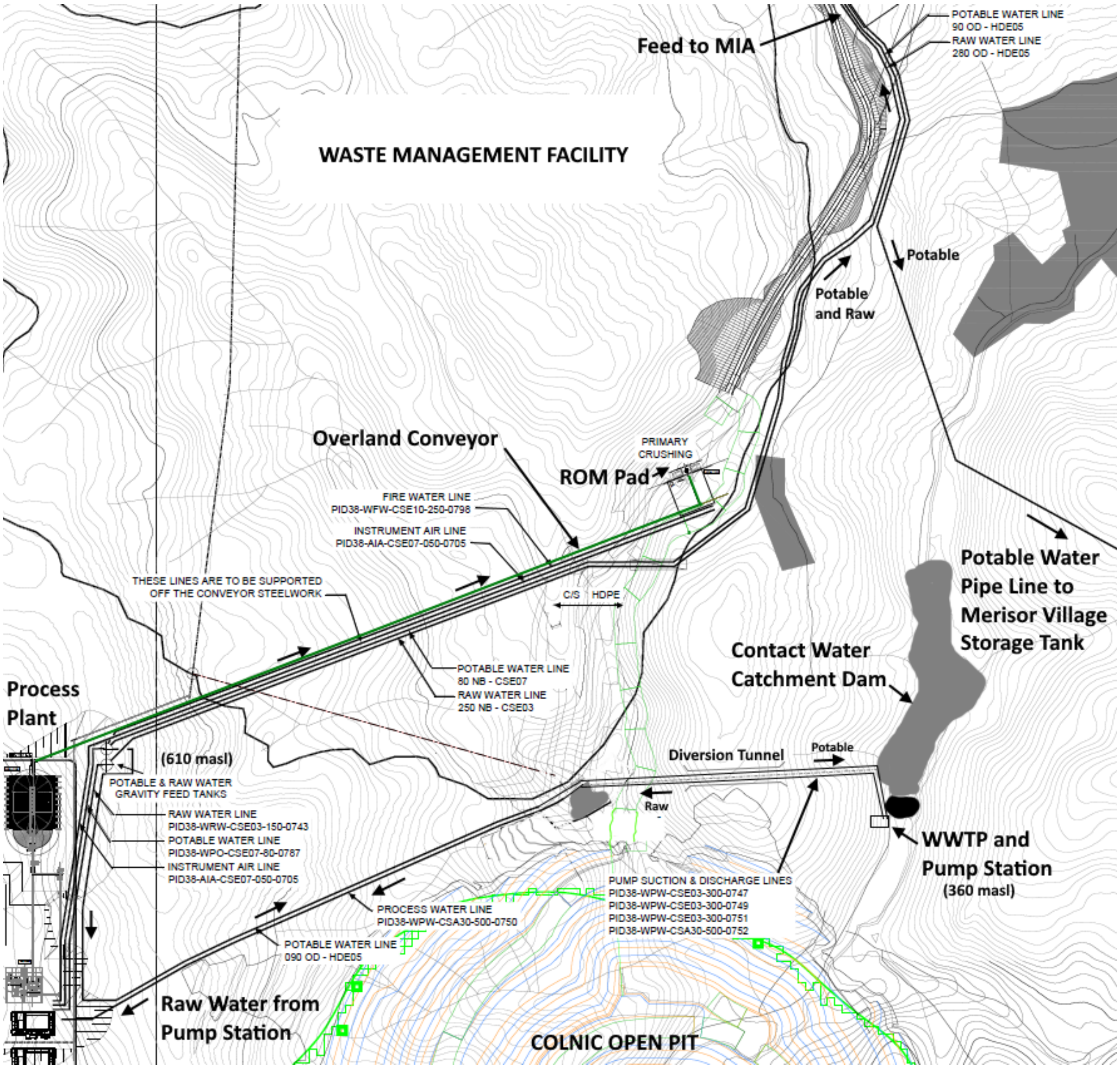


Figure 18.2: Overland Piping Arrangement

A pumping system, which houses six 200 kW pumps (i.e. two trains of three pumps operating in series), will be installed at the contact water catchment dam. The overland pipeline will be 450NB carbon steel pipe and approximately 1.2 km long. It will run adjacent to the WWTP located at the contact water catchment dam, through the contact water sump diversion tunnel and up to the process plant. The pumping system will abstract water from the dam to be fed to the respective water storage tanks located at the process plant. Water to the plant will not be treated via the WWTP, only filtered for raw water distribution. Process water will be used as is, and only excess water to be discharged to the environment will be treated.

Raw water will also be supplied to the MIA via a dedicated raw water tank (150 m^3), which is located on a hill adjacent to the process plant alongside the ROM crushed ore stockpile. Raw water from this storage tank will be gravity fed by means of an overland pipeline to a raw water storage tank and a dedicated fire water storage tank, both located at the MIA, which will be used for raw water and fire water distribution, respectively, at the MIA.

18.1.7.1 Process Plant Potable Water and Distribution

Raw water will be supplied from the 5,000 m^3 on-site raw water storage tank to the potable water treatment plant located in the process plant area. The treated water will be reticulated from the 350 m^3 potable water storage tank to all the relevant infrastructure and safety showers within the process plant boundary via buried overland piping or piping running above ground in the plant area.

The potable water plant will be a containerised unit capable of producing 15 m^3/h of potable water. The water will be pH adjusted and chemically oxidised. After oxidation, the water is then fed to the reverse-osmosis (RO) system. Product from the RO system will be chlorinated and stored for distribution.

Potable water from the process plant water treatment facility will also be supplied to the following off-site infrastructure:

- The MIA and surrounding villages, via a dedicated potable water storage tank (150 m^3), which is located on a hill adjacent to the process plant alongside the ROM crushed ore stockpile. Potable water from this storage tank will be gravity fed by means of an overland pipeline to a potable water storage tank located at the MIA, where it will be redistributed throughout the MIA by means of two 37 kW centrifugal pumps (one duty, one standby). These pumps will also pump potable water to two dedicated potable water storage tanks (100 m^3 each) located in close proximity to the Rovina and the Merișor villages, respectively. These tanks, in turn, will be used to gravity feed potable water to the respective villages.
- The ROM pad/tip ablution facilities and safety showers.
- The WWTP ablution facilities and safety showers.

18.1.7.2 Process Plant Fire Water Protection System

The process plant and MIA will each be fitted with a containerised electric and diesel-powered fire water pumping system directly connected to the dedicated 500 m^3 fire water storage tanks. The electric-powered pump will be used in the event of a fire, and the diesel pump will be a backup in case of power failure due to the unavailability of motor control centres (MCCs). An electrically operated jockey pump will be provided to maintain the pressure in the fire water header during normal plant runs.

The fire water protection system is fully automated and integrated with the process plant's central control system whereby alarms will be sounded at the plant for low system pressure etc.

The fire water pump arrangements will be housed in a containerised unit, complete with its own automated fire control system and dedicated internal sprinkler system.

The fire water system reticulation at the process plant, MIA and ROM pad will consist of buried fire water piping loops and strategically placed hydrants, complete with hose cabinets, and supplemented with portable fire extinguishers placed within the respective infrastructure facilities.

The administration building, mine change house and canteen will have sprinkler systems, hose reels and portable fire extinguishers.

The hydraulic packs at the respective crusher stations, i.e. primary crushing at the ROM pad and pebble crushing at the process plant, including the drive stations of the various conveyors, will each be equipped with an automated sprinkler system. Hose reels and hydrants will be installed along the conveyors to cover their complete length, including the transfer towers.

Foam monitors with foam nozzles will be installed at the fuel dispensing area at the MIA.

A complete self-contained fire alarm system will be installed in all the buildings in order to comply with the local codes and insurance underwriter's regulations for fire protection.

18.1.8 Process Plant Communications and Control Systems

There are communication requirements for accounting, purchasing, maintenance, and general office business, as well as specialised requirements for control systems.

An integrated information system will be provided, taking full advantage of the latest operating software, hardware, and information technology, and enabling effective telephonic and digital communications.

A fibre optic network backbone will be provided, interfacing with all the infrastructure and systems and enabling data acquisition and communications. Infrastructure buildings will be equipped with data trunking and data outlets to facilitate data communications, including utilising voice-over-internet-protocol for telephonic communications.

The office ethernet network will support accounting, payroll, maintenance, and other servers, as well as individual user computers. High-bandwidth routers and switches will be used to logically segment the system and to provide the ability to monitor and control traffic over the network.

The process plant SCADA system ethernet network will support the screen, historian, and alarm servers and connect to the process plant control room computers, as well as the programmable logic controllers (PLCs) and other control systems provided with ethernet communication capabilities. This system will incorporate redundancy and will be designed to minimise traffic and latencies. No phone or user computer will be connected to this system.

Refer to the Control System Architecture Diagram Doc. No. SP0829-9300-I-BLOC-20001

A security system will also be incorporated into the plant network. Using a dedicated video server and monitors, internet protocol cameras, utilising power over ethernet connections, will be plugged into dedicated switches. Security cameras are typically located in storerooms, parking lots, visitor lobbies, warehouses, and areas where sensitive materials are kept.

Mobile radios will also be used by the mine and plant operation personnel for daily control and communications while outside the offices.

The plant management information and communication systems will be specified by ESM.

18.1.9 Process Plant Power Supply

The project's primary energy source will be electricity from the Romanian power grid. An agreement between Mintia-Deva Power Station, Transelectrica, and ESM will be realised to provide the electrical power supply, with an estimated grid connection capacity of 100 MVA, for the project.

An underground transmission line will supply power to the switchyard located at the process plant. The line will tap into the existing 110 kV switchyard located at the power station, which is situated in the Hunedoara county (south-eastern Transylvania), on the banks of the Mureş River, 7 km from the city of Deva. The 36 km underground transmission line will follow an existing underground power line for a section of the route and will be terminated at the process plant 110/20 kV switchyard substation as shown in Figure 18.3. The power line and switchyard will be owned and operated by Transelectrica.

The switchyard at the process plant will consist of two 110/20 kV 30 MVA duty power transformers. The design, specification, supply, and installation of the switchyard, including the 110/20 kV transformers, will be by a local contractor and will be completed in accordance with the operator's requirements. The switchyard scope includes independent metering and protection panels associated with the switchyard and transformers.

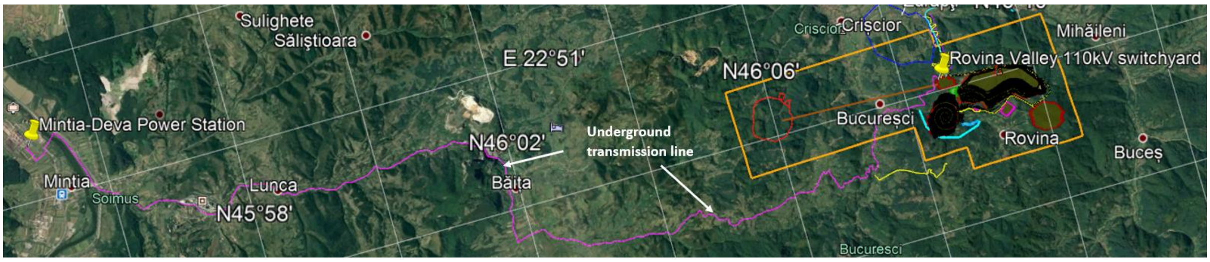


Figure 18.3: 110 kV Grid Connection Underground Transmission Line Route

18.1.9.1 Power Demand

The steady-state continuous power demand is estimated at **50,204 kW**, with the SAG and ball mills being the only loads of critical relevance to the maximum start-up energy demand. The SAG and ball mill drives will be driven by dual 8,000 kW squirrel cage motors, and the start-up current will be limited with a VSD. It is anticipated that the VSD will reduce the starting power demand of the ball mill to a maximum of 1.6 times the rated capacity of the motor.

The start-up sequence will have the SAG mill start up first, followed by the ball mill, which will have a start-up sequence lasting for approximately 20 s to 30 s, and it will increase the plant's maximum power demand to 62,555 kW. The power transformers are suitably designed to deliver the required power for the start-up duration without interruptions to the other loads and will limit the voltage regulation to within 10 % of the rated system voltage.

The steady-state continuous power demand will increase to **50,994 kW** when the mining activities are transferred to the Rovina pit, with the plant's maximum power demand increasing to 63,345 kW.

An outline of the electrical power demand is shown in Table 18.1, based on the mechanical equipment list and plant infrastructure.

Table 18.1: Electrical Power Demand

Project Load	Colnic Pit		Rovina Pit	
	Continuous Power Demand (kW)	Maximum Start-Up Demand (kW)	Continuous Power Demand (kW)	Maximum Start-Up Demand (kW)
Process Plant	17,259	17,259	17,259	17,259
Off-Site Infrastructure	1,152	1,152	1,152	1,152
Mining and Dewatering	790	790	1,580	1,580
Waste Facility Conveyors	5,347	5,347	5,347	5,347
SAG Mill	12,407 (16,000[a])	12,407	12,407 (16,000[a])	12,407
Ball Mill	13,249 (16,000[a])	25,600	13,249 (16,000[a])	25,600
Total	**50,204**	**62,555**	**50,994**	**63,345**
[a] Rated				

18.1.9.2 Electrical On-Site Infrastructure

The power transformers from the 110/20 kV switchyard will tie into the process plant main intake substation located next to the switchyard. The on-site infrastructure will be reticulated from the process plant main intake substation via a buried medium-voltage (MV) backbone network.

Power reticulation to the electrical systems will be done at the following nominal voltages, which will comply with IEC 60038, together with the permitted variation at the point of use:

- 20 kV ± 5 %, three-phase, three-wire, solidly earthed
- 400/230 V ± 5%, three-phase, four-wire, solidly earthed i.e. separate protective earth (PE) and neutral (N) conductors from transformer to consuming device, which are not connected at any point after the building distribution point (TN-S system)

The system's three-phase fault levels will not exceed the following values:

- 20 kV system: 21 kA
- 400 V system 65 kA

The electrical on-site infrastructure caters for the following process plant infrastructure:

- Gatehouse

- Change house
- Plant control room
- Plant workshop
- Plant office building
- Plant metallurgical/assay laboratory
- Plant warehouse
- Weighbridge control room

18.1.9.3 Medium-Voltage Switchgear

A common 20 kV prefabricated main intake MV substation has been allocated for the on-site and off-site infrastructure. The substation will be supplied with a metering panel to check the independent metering of the 110 kV switchyard.

The switchgear will be rated using the following details:

- Medium voltage: 20 kV
- Frequency: 50 Hz
- Main busbar rating: 2,000 A
- Basic insulation level: 125 kV
- Short-circuit rating: 31.5 kA – 3 s

Switchboards will be of the indoor type, of self-supporting extensible metal clad construction, with a minimum degree of protection of IP42 (Ingress Protection or International Protection ratings as defined in IEC 60529), and a single busbar system with circuit breakers and electrically held contactors. Circuit breakers will be used to control transformer and other feeders, as well as large motors, while contactors may be used for smaller motors. Circuit breakers will have trip-free operating mechanisms and use a vacuum as the insulating and switching medium. Contactors will be electrically held and also use a vacuum as the switching medium. All circuit breakers and contactors will be horizontally withdrawable and be provided with circuit earthing facilities.

The switchboards will be arranged in line as one integral unit, with provision for a spare feeder for future extensions.

The circuit breaker current ratings will be 630 A, 1,250 A, and 2,500 A, with a motor charged, electrically released spring closing. Each circuit breaker will have local electrical tripping and closing, except for the VSD feeders which will be controlled entirely by the drive.

For switches used for motor starting, the local closing facility will be operative only when the circuit breaker or contactor is in the test position.

All incoming and outgoing circuit breakers will be fitted with power-monitoring relays and/or included within the protection relays. The relays will include a Modbus communications port for serial communications with the plant's SCADA system.

The following typical standard protection functions are used within the project:

- Overcurrent Protection (Breaker types 50 and 51)
  The three-phase overcurrent protection includes two stages: the instantaneous or high-set overcurrent (50) and the time-delayed or low-set overcurrent (51). When at least a one-phase current exceeds the pre-set value, the relay will deliver a trip command or activate a timer, depending on the location of the protection in the distribution levels. When the set operating time has elapsed and the current is still above the threshold value, the trip command is issued. The high-set stage is used as short-circuit protection. The set value must not exceed the calculated minimum short-circuit current to ensure proper detection of faults. The low-set stage is used as overcurrent protection. The set value is related to the nominal current of the load connected.

- Earth Fault Protection (Breaker type 51N)
  In a solidly earthed network, the earth fault is detected by the overcurrent protection system. The earth fault current is detected by neutral current measurement (51N). If the earth fault current exceeds the pre-set value, the relay will either deliver a trip command or activate a timer. When the set operating time has elapsed and the earth fault current is still above the threshold value, the trip command is issued. The set value must not exceed the minimum earth fault current to ensure proper detection of faults.

- Overvoltage (Breaker type 59) and Undervoltage Protection (Breaker type 27)
  The overvoltage and undervoltage protection functions measure the voltage between phases in a three-phase system. Time and voltage settings are used to achieve discrimination between different protection levels. The overvoltage stage operates if the measured voltage exceeds the set value for more than the adjusted time, e.g. to protect an on-load tap changer or a motor. The undervoltage stage operates on voltage drop or complete loss of primary network voltage and is detected at the cable side of the incomer and at the busbar. In the case of undervoltage, the incomer will be tripped.

- Motor Start-Up Supervision (Breaker types 66, 48, 51LR)
  The motor start-up supervision is set to protect the motor from undesirable thermal stress during the motor starting process and to avoid/prevent multiple starts higher than specified by the manufacturer within the determined time. This supervision function block includes a start-up supervisor, thermal stress calculator, and cumulative start-up protection. Additionally, stall protection can be activated to indicate whether a motor is accelerating during start-up or not.

The allocations of the protection functions given in Table 18.2 are proposed general requirements. Once the implementation phase of the project is reached, then the protection schemes will be finalised, with the emphasis on stability and reliability.

Table 18.2: Power Plant Incomers

Breaker Type	50	51	51N	27	59	81	51LR	48	66
Incomer	X	X	X	X	X	X			
Bus coupler			X						
Feeder	X	X	X						
VSD Feeder	X	X	X						
Motor Feeder	X	X	X	X			X	X	X

18.1.9.4 Harmonic and Power Factor Correction

A harmonic and power factor correction system will be installed at the main intake substation. The system will be designed to comply with the requirements of IEC 61000-3-6 and the Romanian Distribution Interconnection Permit. This will be calculated at the point of common coupling. The total harmonic voltage distortion at the 20 kV level, evaluated for 2nd to 40th order components, will generally not exceed 8 % continuously or 12 % for 15 s in any 150 s period as a result of the combined effects of the plant and the utility supply.

The harmonic filter bank shall correct the system's power factor to 0.96 lag and shall maintain the system's harmonic voltage and current distortion within the IEEE 519 limits and as given in IEC 61000-3-6.

18.1.9.5 Standby Power Supply

The standby power requirement for the project will be provided by four 1,250 kVA diesel generator set(s) installed onto the main 20 kV switchboard via step-up transformers and supplying power to plant-wide substations containing critical loads.

In the event of restricted availability of power or the loss of the main supply, the standby power generators will operate. A load management system will ensure that only critical loads are energised within the available capacity.

The generator(s) will be capable of paralleling with the grid supply in the event of main power return to ensure a seamless transition back to grid supply.

Each of the generator set(s) will have its own diesel tank with a capacity of 1,500 L, sufficient to keep each unit running for a period of 8 h at 75 % loading. Diesel for these generator sets will be obtained from the diesel storage facility located at the MIA and decanted into the generator diesel day tanks.

18.1.9.6 Transformers

Transformers will be double-wound (copper), oil-immersed, three-phase, three-wire, 50 Hz systems having standard phase rotation and will be manufactured in accordance with IEC 60076.

Transformer impedance will be in accordance with IEC 60076 unless other considerations (e.g. fault level) dictate otherwise. The vector group for high-voltage to low-voltage (LV) transformers will be Dyn11. In general, tap changers will be of the offload type with a tapping range of +5 %, +2.5 %, 0, −2.5 %, −5 %, of primary voltage.

18.1.9.7 Low-Voltage Distribution

The maximum transformer rating for LV supplies will be 2,500 kVA. Each transformer will feed a 400 V MCC that supplies power to multiple plant areas. Feeds to the MCCs will be single feeds only.

The MCCs will feed the lighting distribution boards that will supply lighting and small power distribution (normal) at 400 V/230 V (single-phase or three-phase).

18.1.9.8 Control Power

The control voltage for motors and heaters using contactor-type controllers will be 110 VAC. The control power for contactors will be derived from integral control single-phase, dry-type, double-wound power transformers located in each MCC, provided with an earthed screen between the windings and with one leg of the 110 V winding connected to earth.

All field controls will be at 110 VAC.

The control voltage for MV circuit breaker closing and tripping circuits, circuit breaker spring charging, protection relay power supplies and other auxiliary functions will be 110 VDC, supplied from a high-integrity battery and charging system. Ring main units will be operated by a 24 VDC control voltage.

The control voltage for LV circuit breaker closing and tripping circuits, circuit breaker spring charging, protection relay power supplies and other auxiliary functions will be 110 VAC, supplied from suitable-sized and rated fuses on the supply (live) side of the circuit breaker.

Where the plant control system is required to control 110 V powered devices, suitable 24 VDC interposing relays will be incorporated within the device enclosures. Signals to the plant control system will be by means of potential-free contacts.

18.1.9.9 Prefabricated Substations

Substations will employ prefabrication techniques to minimise on-site construction activity. They will be prefabricated, steel-framed, clad and roofed structures, insulated and lined with non-combustible materials. Materials of construction and protective coatings will be appropriate for the corrosive conditions at site.

The substations will be installed on suitable footings to provide a minimum floor level of 1.2 m above local ground level or be incorporated within the structure of the plant buildings. A minimum of two entrances will be provided to permit safe egress in the event of a fire at any location in the substation. An accessway of at least 600 mm will always be maintained with any number of doors opened or switchgear in the racked-out position.

Substations will be heated/cooled and pressurised with filtered air. The system will be designed and manufactured to suit the specified site conditions, and the room temperature will be maintained within a range of 5 °C to 25 °C, including when any one component of the system is out of service.

Handheld fire extinguishers suitable for electrical fires and of adequate capacity for the size of the substation will be provided next to the doors of the substations. Each substation will have a fire detection system with a fire indication panel alarm signal linked back to the SCADA system.

18.1.9.10 Motor Control Centres

The MCCs will be of the compartmentalised, non-withdrawable type with moulded-case circuit breakers, magnetic contactors, and earth bus, and they shall comply with IEC 61439-2 and shall bear the European Commission CE Mark to signify conformance with the European Norms (EN) and regulations.

The MCCs will have the following features:

- Enclosures will be of the general-purpose type for indoor service (IP 54, see IEC 60529) or weatherproof type for outdoor service (IP 65, see IEC 60529), as required.
- Main buses will have a minimum capacity of 400 A.
- Each MCC will have a single 230/110 V AC control transformer rated for each MCC's control circuit requirements.
- The control circuit will have UPS backup rated at 10 kVA in the event of a power failure.
- Outgoing power and control wiring will be brought out to terminals in the wireways.
- Siemens equipment will be used with the SIMOCODE Pro V relay for efficient and reliable motor protection.

18.1.9.11 Electrical Motor Control Stations

Every motor and every item of process equipment will have a local control station with Start-Stop and mushroom-head type, latch-off Emergency-Stop push buttons. Latch Emergency-Stop push buttons will stay in the open position until manually reset. Additional controls for use in manual mode, such as forward and reverse, may also be provided. The local control station will be within sight of the associated drive motor or equipment.

18.1.9.12 Earthing and Lightning Protection

A common earthing system will be provided whereby the MV and LV earthing systems will be interconnected. Each substation will be provided with a main earthing bar to which all earth cables will be connected. Each substation location will be provided with one earth system only.

Each major item of electrically powered equipment will be earthed by connecting it to the local main equipotential bonding conductor, which will, in turn, be connected to the earth bar of its supply switchboard. All metallic structures and equipment will be interconnected by equipotential bonding conductors to ensure that step and touch potentials are limited to permissible levels. The switchboard earth bars will be connected to the substation earth bar.

A plant-wide lightning risk assessment of all the buildings and structures will be conducted during the detailed design phase of the project to determine the necessity to provide structural lightning protection. Where specific measures are required, they shall be in accordance with IEC 61312.

18.1.9.13 Electrical Cables

The following shall apply to electrical cables:

- Cables shall be designed, manufactured, and tested in accordance with IEC 60502.
- All cables shall be approved, and CE labelled.
- Cable cores shall be stranded copper, Class 2, in accordance with IEC 60502.
- Cables shall be non-flame propagating in accordance with IEC 60332-3-24:2018.
- All outdoor cables shall either be buried in the ground or placed on cable racking.
- Cables shall cross underneath roads in dedicated sleeve polyvinyl chloride pipes.
- Jackets shall have the manufacturer's name, voltage, conductor size and number of conductors, construction number, and flame test classification imprinted at regular intervals. Cables shall be metre-marked.
- Armoured cables shall be utilised throughout the installation.

18.1.9.14 Cable Racking

Cable racking will be used where cables are running on structures or indoors, or where cable support is required. Heavy-duty cable ladders will be provided for edge-mounted applications. Materials of construction of ladders will be determined on the basis of the expected environmental conditions at the point of installation.

Appropriate materials will include hot-dipped galvanised steel for non-aggressive environments, and 316L stainless steel and fibre-reinforced polyester for highly corrosive areas.

Cable ladders will be sized to provide 20 % spare capacity remaining at the end of the design phase of the project.

18.1.9.15 Lighting

Provision has been made for LED (light-emitting diode) lighting, which will be structure-mounted to ensure safe working conditions around the process plant area. Lighting will also be installed to ensure that visual security monitoring can always be conducted in and around the process plant and associated infrastructure to maintain a safe work environment. The final design and layout will be confirmed during the implementation phase. Lighting will be designed to achieve levels of luminance generally in accordance with, and no lower than, the values given in the International Commission on Illumination (CIE) Document: CIE 129.

18.1.9.16 Fire Detection System

Provision has been made for fire detection systems for all MV switches, MCCs, servers and control rooms. These rooms will also be equipped with handheld firefighting equipment. The fire detection systems will be integrated with the plant's central control system to alert the plant operators of any fire incidents.

18.2 PROJECT OFF-SITE INFRASTRUCTURE

The proposed off-site infrastructure will support the mining, plant, and waste facility operations.

Figure 18.4 shows the off-site infrastructure.

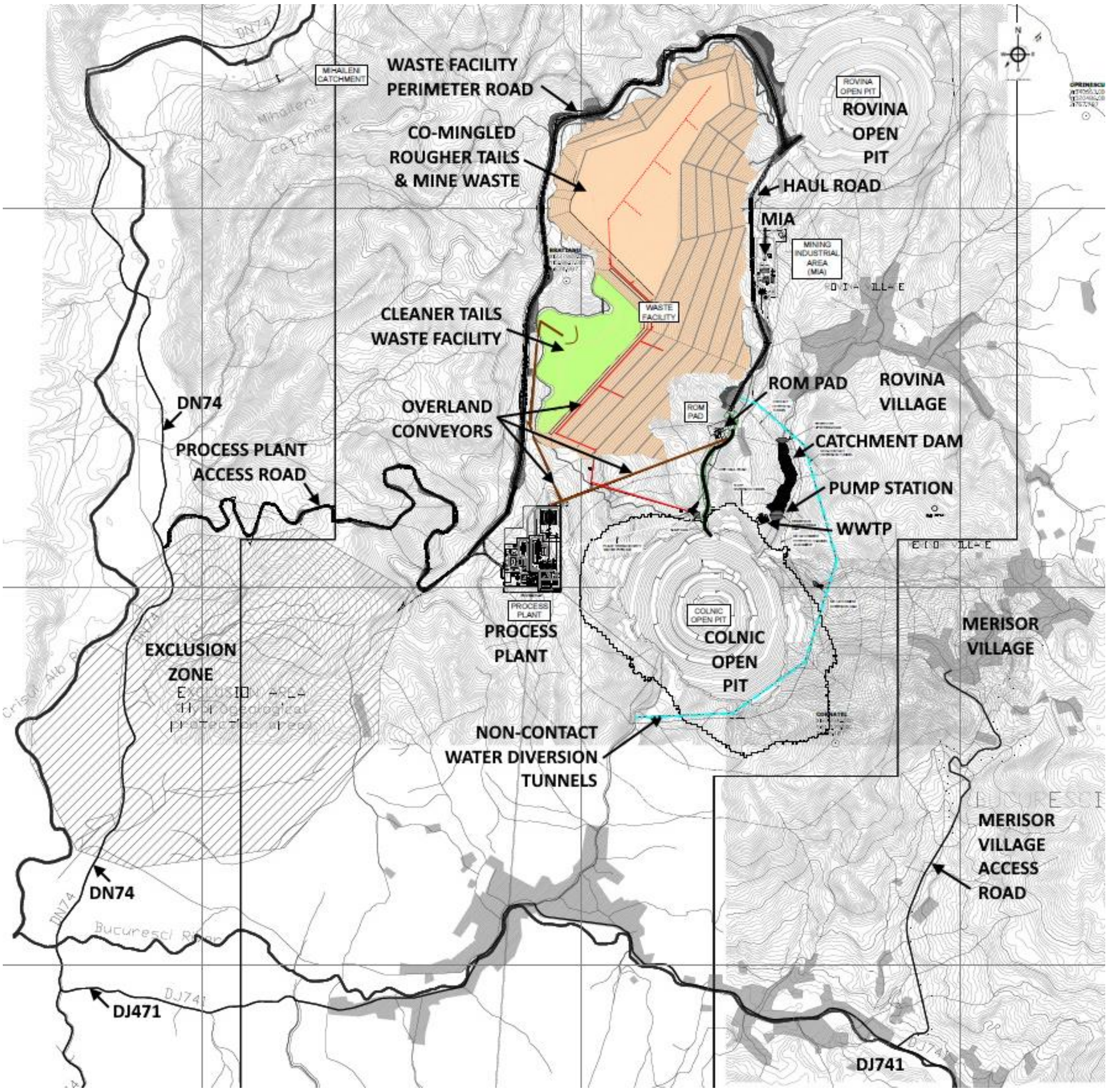


Figure 18.4: RVP Off-Site Infrastructure

The off-site infrastructure required for the development of the project will be the following:

- Roads
- MIA:
  - MIA infrastructure buildings
  - MIA site drainage and pollution control
  - MIA sewage collection and disposal
  - MIA security

- MIA raw water supply
- MIA potable water supply
- MIA fire protection system
- MIA fuel depot
- Explosives
- MIA communications

- Off-site infrastructure power supply and distribution

18.2.1 Roads

The Rovina Valley project road network comprises the following roads (see Figure 18.4):

- Merișor village access road
- Process plant access road
- Waste facility perimeter road
- Mine haul roads

Design, engineering, and construction of these fit-for-purpose roads include erosion and sediment control, and the design and erection of suitable bridge structures, complete with open channels, floodways and, in particular, the culvert drainage systems.

Road furniture, road signage, and safety aspects are catered for in the project.

18.2.1.1 Merișor Village Access Road

The Merișor village access road will be constructed from the Bucureşci Road (DJ741) and comprise the following:

- Classification: main public road
- Surface: asphalt
- Length: 3.0 km
- Width: 7 m wide, i.e. two 3 m wide lanes with a 0.5 m shoulder on either side
- Maximum speed: 60 km/h
- Road starts: at the DJ 741 road, 335.000 masl
- Road ends: at entrance to Merișor village, 565.053 masl

18.2.1.2 Process Plant Access Road

The process plant access road will be constructed from the Crişcior–Zdrapti Road (DN74), lead directly to the process plant site main entrance, and comprise the following:

- Classification: private – secondary
- Surface: base/crushed stone
- Length: 3.8 km
- Width: 7 m wide, i.e. two 3 m wide lanes with a 0.5 m shoulder on either side
- Maximum speed: 60 km/h
- Road starts: at the DN 74 road, 304.929 masl

- Road ends: at the process plant terrace, 598.275 masl

18.2.1.3 Waste Facility Perimeter Road

The waste facility perimeter road will be constructed from the process plant terrace and routed along the waste facility western and northern perimeters leading to the MIA terrace and comprise the following:

- Classification: private – secondary
- Surface: dirt
- Length: 5.8 km
- Width: 7 m wide, i.e. two 3 m wide lanes with a 0.5 m shoulder on either side
- Maximum speed: 40 km/h
- Road starts: at the process plant terrace road, 598.146 masl
- Road passes: haul road intersection at 437.129 masl
- Road ends: at MIA terrace entrance

18.2.1.4 Mine Haul Roads

A system of mining haul roads will be constructed across the site to connect the pits with the ROM pad/tip, the waste rock tip, and the waste storage facilities.

The network of haul roads comprises a main haul road between the Colnic and Rovina pits of typical cross section as shown in Figure 18.5.

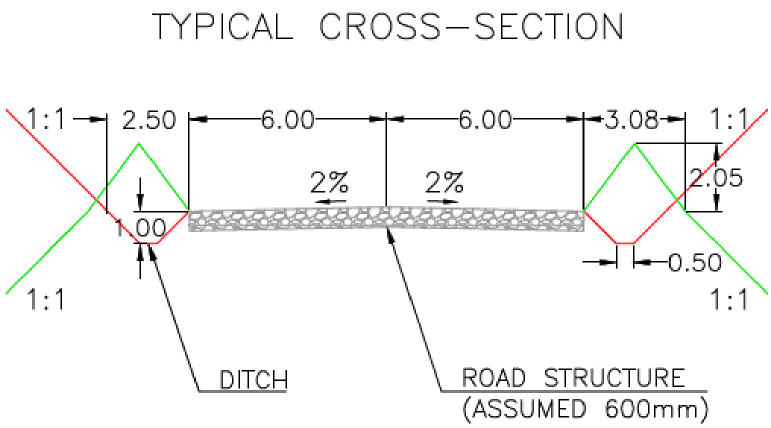


Figure 18.5: Typical Cross Section of Haul Roads

18.2.1.5 Road Culverts

The road culverts will generally comprise the following:

- Excavating for pipe trenches
- Laying of bedding material
- Laying of Class 100D precast pipe culverts, 600 mm diameter
- Construction of reinforced concrete culvert bases, headwalls and wingwalls

18.2.2 MIA Infrastructure Buildings

The MIA infrastructure buildings will consist of the following:

- Gatehouse
- Change house
- Mining main office and administration building
- Mining supervisor/survey mine operations building
- Mining fuel dispensing office
- Heavy-vehicle workshop
- Tyre workshop
- Welding workshop
- Warehouse
- Wash bay

All prefabricated infrastructure buildings will be completely furnished and, where applicable, feature adequate fit-for-purpose lighting, power socket outlets, data and communication trunking, hot/cold water plumbing systems, heating, ventilation, extraction, and air-conditioning systems.

All prefabricated structures will be founded on mesh reinforced concrete slabs, complete with surrounding concrete aprons leading to culverts to facilitate proper storm water management.

The position of the MIA infrastructure buildings is shown in Figure 18.6.

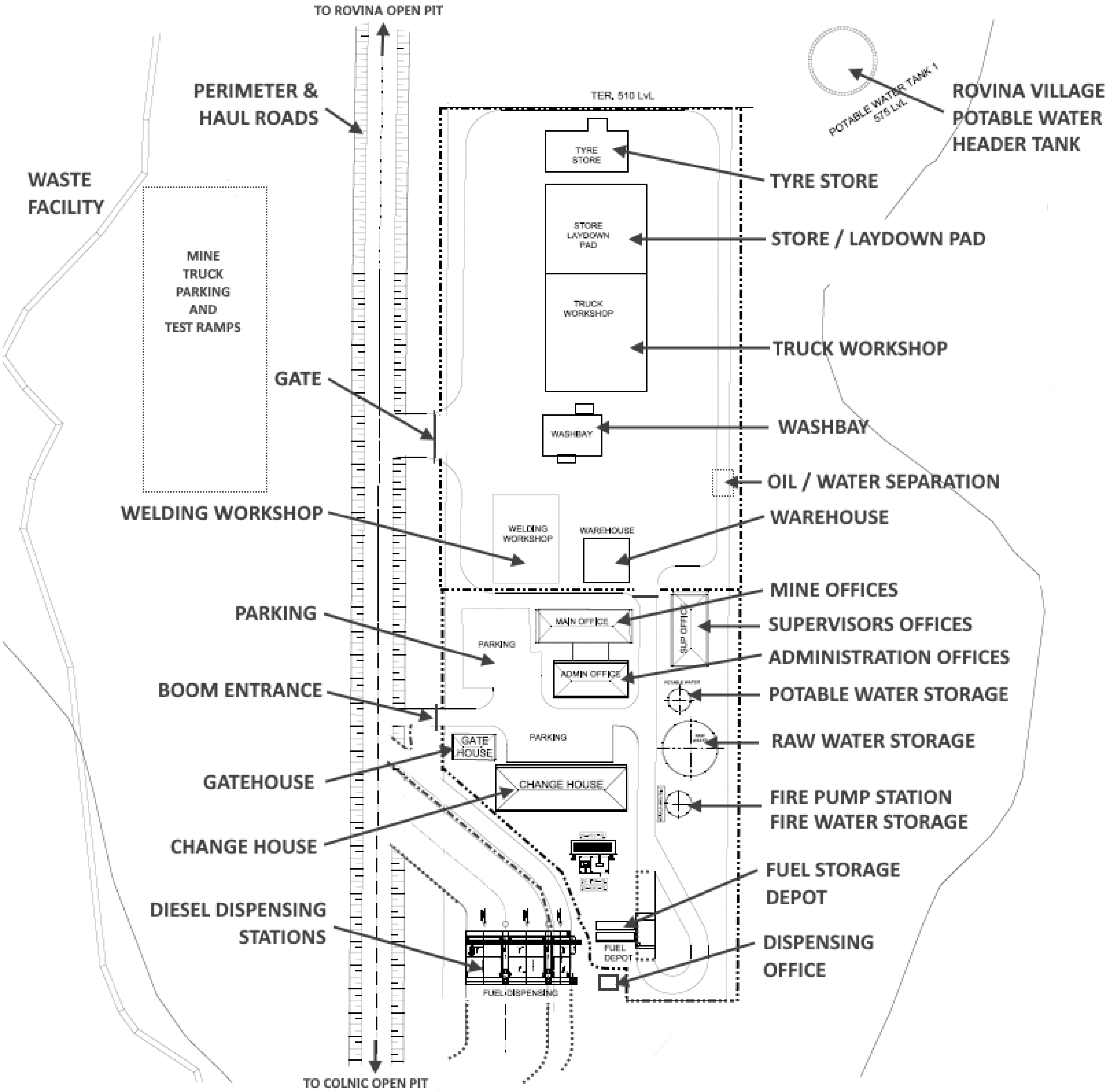


Figure 18.6: MIA Infrastructure Buildings

18.2.2.1 MIA Gatehouse

A prefabricated, insulated panel gatehouse of 122 m^2 (14.2 m × 8.5 m) will be constructed at the main access gate to the MIA and includes a security supervisor's office, complete with observation windows and a fitted desk arrangement. The building also includes a small kitchenette, male/female ablution facilities, security office, and a waiting room, complete with turnstile-controlled access.

All workers and visitors will enter the MIA through the reception area. Authorised personnel can then proceed to the change house via the access-controlled turnstiles. Personnel and visitors' access will be monitored from the fully furnished security office, which contains an access control and closed-circuit television (CCTV) system.

The MIA gatehouse will be the central hub for control and monitoring of all security, fire, and emergency systems within the mining area.

18.2.2.2 MIA Change House

A prefabricated, insulated panel change house building (40 m × 10 m) will be constructed near the main access to the MIA and will include a change house, laundry, first-aid room, and dining/conference room. The prefabricated structure will be founded on a mesh reinforced concrete slab. A 2 m concrete apron slab will form a walkway around the building.

The change house will contain both male and female change areas consisting of lockers and benches for the mining staff to store their clothes when they arrive for their shift. At the end of the shift, the staff will return to the change area, remove their dirty work clothes (which will be washed in the adjacent laundry), shower, and then proceed through the change area, where they will retrieve their own clothes.

The MIA change house has been sized to cater for a maximum of 134 workers taking office staff, hourly shift staff, R&R staff, plant shutdowns, third-party workers, and visitors into account.

The male change house area has been sized as follows:

- 104 double lockers in the change area (1 locker per person)
- 20 showers (1 shower per 5 people maximum)
- 10 toilets (1 toilet per 10 people maximum)
- 10 wash-hand basins (1 wash-hand basin per 10 people maximum)
- 10 urinals (1 urinal per 10 people maximum)

The female change area has been sized as follows:

- 30 double lockers in the change area (1 locker per person)
- 6 showers (1 shower per 5 people maximum)
- 3 toilets (1 toilet per 10 people maximum)
- 3 wash-hand basins (1 wash-hand basin per 10 people maximum)

A fully equipped laundry room (5 m × 5 m), where all the overalls from the shifts will be washed, will be provided adjacent to the change house.

A fully furnished first-aid room (5 m × 5 m) will be provided for the treatment of minor injuries sustained within the process plant or as a holding/waiting room before transportation to the nearest hospital from the process plant site.

A fully furnished canteen/training room (10 m × 7 m), designed to accommodate 70 personnel, will be provided. A section of the room can be partitioned off to form a conference/training room as required. The canteen will be utilised as an area where staff can partake of their self-provided meals and refreshments.

18.2.2.3 Mining Main Office and Administration Building

A fully furnished, insulated panel construction mining office building will be provided for the mine management and accounts personnel.

The mining office and administration building will have a total floor area of 329 m^2. The mining office building will comprise the following:

- Safety, health, and environment manager's office
- Geologist manager's office
- Drill and blast manager's office
- Filing room
- Two visitors' offices
- Reception
- Maintenance manager's office
- Mine manager's office
- Mobile plant manager's office
- Boardroom
- Server room
- Kitchenette
- Male and female ablutions

Security fencing will be provided around the MIA, including the infrastructure buildings and parking area perimeters.

18.2.2.4 Mining Supervisor/Survey Mine Operations Building

A fully furnished, prefabricated, insulated panel construction mining supervisor/survey mine operations office building will be provided for the supervisors, survey and drawing office staff.

The supervisor office building will have a total floor area of 175 m^2. The supervisor office building will comprise the following:

- Geology assistant's office – open plan
- Survey and drawing office – open plan
- Drill and blast supervisor's office
- Maintenance supervisor's office
- Pit supervisor's office
- SHE officer's office
- Boardroom
- kitchenette
- Male and female ablutions

18.2.2.5 Mining Fuel Dispensing Office

A prefabricated, insulated panel fuel-dispensing office of 12 m^2 (3.4 m × 3.4 m) will be constructed adjacent to the fuel-dispensing stations and alongside the fuel storage tanks located in the MIA. The office includes an observation window and a fitted desk arrangement.

The fuel-dispensing office will be the central hub for control and recording of all diesel fuel-dispensing transactions within the mining area.

18.2.2.6 MIA Heavy-Vehicle Workshop

A workshop will be established to enable maintenance and repair of the mining truck and heavy-vehicle fleet and is located on the north side of the MIA terrace.

The heavy-vehicle workshop will comprise two sections, namely workshop inspection and maintenance bay floor area and store/laydown pad area and consists of a steel pre-engineered portal frame building enclosed by Chromadek roof and side-wall sheeting. The steel columns will be founded on reinforced concrete plinths and spread footings, and the workshop floor will consist of a mesh reinforced concrete slab.

The heavy-vehicle workshop inspection and maintenance bay floor area will be 1,360 m^2 (40 m × 34 m), which will be split in half into two bays that will be serviced by a 75 t overhead gantry crane and a 25 t overhead gantry crane, respectively. Oils and fuel spillage control includes floor channels leading to an oil trap feeding the oil/water separator system. Access to the four bays is via electrically operated roller shutter doors.

The heavy-vehicle store/laydown pad floor area will be 1,020 m^2 (30 m × 34 m) and will be serviced by a 15 t overhead gantry crane. Enclosed areas will be provided on either side of the store/laydown pad floor area within the steel portal frame structure comprising a component store, tool store, component cleaning booth, parts receiving portal, male and female ablutions, tearoom and four fully furnished offices for staff. Access to the store area is via electrically operated roller shutter doors on either side.

The appropriate workshop tools will be provided for the heavy-vehicle workshop.

18.2.2.7 MIA Tyre Workshop

A workshop will be established to enable maintenance and repair of the mining truck and heavy-vehicle fleet tyres and is located on the north side of the MIA terrace.

The tyre workshop will comprise two areas, namely the tyre store area and tyre workshop area, and consists of a steel pre-engineered portal frame building with enclosed roof and side-wall sheeting. The steel columns will be founded on reinforced concrete plinths and spread footings, and the workshop floor will consist of a mesh reinforced concrete slab.

The tyre workshop floor area will be 378 m^2 (27.4 m × 14 m) and will be serviced by a 7 t overhead gantry crane. Access to the workshop is via electrically operated roller shutter doors on either side.

A lean-to brick-enclosed compressor room (5.6 m × 4 m) will be provided alongside the tyre store, complete with compressor, receiver, compressed air reticulation piping and outlets located throughout the workshop.

The appropriate workshop tools will be provided for the tyre workshop including large tyre handling equipment.

18.2.2.8 MIA Welding Workshop

A workshop will be established to enable welding maintenance and general repair of the large mining equipment steel components, such as the FEL buckets, bins, hoppers, loading arms, and hydraulic systems.

The welding workshop will consist of a steel pre-engineered portal frame building enclosed by Chromadek roof and side-wall sheeting. The steel columns will be founded on reinforced concrete plinths and spread footings, and the workshop floor will consist of a mesh reinforced concrete slab.

The welding workshop floor area will be 660 m² (30 m × 22 m) and will be serviced by a single 75 t overhead gantry crane. Oils and fuel spillage control includes floor channels leading to an oil trap feeding the oil/water separator system. Access to the workshop is via electrically operated roller shutter doors on either side.

The appropriate workshop tools will be provided for the welding workshop.

18.2.2.9 MIA Warehouse

A 330 m² enclosed steel and cladded structure warehouse will be provided at the MIA for the purpose of storing spares and equipment for the mining operations. Suitable shelving and racking will be incorporated in this building to store and manage the stored items.

An unfenced laydown area is located adjacent to the MIA warehouse.

The warehouse includes a brickwork stores dispatch room, with a service counter and waiting area (5.3 m × 4 m), and a warehouse management office (4.3 m × 2.4 m).

18.2.2.10 MIA Wash Bay

The mining fleet wash bay floor area will be 280 m² (20 m × 14 m) comprising a mesh reinforced concrete slab and a 6 m container on either side featuring a staircase to an elevated steel platform housing the water wash cannon swivel nozzle arrangements. Wash bay water pumps will be housed in the 6 m containers. Dirty water management, oils and fuel spillage control includes primary and secondary floor channels leading to an oil trap feeding the oil/water separator system for water recirculation back to the MIA wash bay.

18.2.3 MIA Site Drainage and Pollution Control

The MIA terraced areas will be constructed with berms and side drainage as required to ensure that any storm or dirty water runoff not contained in the bunded areas is diverted via a series of dedicated drainage channels and then routed via the oil/water separator system and finally discharged into the main contact (dirty) water catchment pond located north of the Colnic pit.

18.2.3.1 Storm Water Berms

Storm water cut-off berms will be constructed to prevent storm water from entering lower lying areas from areas with a higher elevation. The berms will be constructed using the material from the bulk excavations when the bulk earthworks are carried out.

18.2.3.2 Surface Drainage Side Drains

Surface drainage of the MIA area will be achieved by using side drains. The surface of the plant will be sloped to allow the water to flow freely away from the plant. The plant roads will be built to have a single cross fall of between 2 % and 4 % in the direction of the side drain.

18.2.3.3 Oil/Water Separation System

The MIA terrace will feature an automated 15 m^3/h oil/water separation system comprising pumps, oil skimmers, debris strainers and separator units that will enable the following pollution control measures with excess water being drained into the storm water drainage system:

- Separation of hydrocarbon waste, i.e. fuels, oils, and greases from the MIA workshop and wash bay facilities assuming a mean oil droplet size of 25 µm, a high surfactant load, a high suspended solids load, and a high feed oil load.
- Separation of non-hydrocarbon waste, i.e. soil, stones, sawdust, plastic, and general paper.
- Circulation of water for reuse at the wash bay facility comprising a 20 m^3 collection sump arrangement.

18.2.4 MIA Infrastructure Sewage Collection and Disposal

A 21 m^3/d containerised biological sewage treatment plant will be provided south of the MIA terrace for the treatment and disposal of domestic raw sewage and change house wastewater that is generated by and collected from the various infrastructure buildings.

Fats, oils, and greases will also be treated with fat traps at the source, as required, and will not exceed 10 mg/L before entering the sewage plant.

Sewage reticulation piping and manholes will be provided to facilitate the flow of sewage under gravity to a collection manhole located adjacent to the sewage treatment plant. The sewage will be pumped via a submersible pump into the containerised treatment plant.

18.2.5 MIA Security

The MIA site will be fully enclosed by a 2.4 m high mesh fence to keep out range animals and unauthorised people. Access to the plant site will be restricted to one access point at the main gate, which will be equipped with a gatehouse that is manned 24 h/d. Other emergency and maintenance access gates will be provided but will be kept locked at all times.

Booms to control vehicle access will be provided at the entry gate to the MIA.

18.2.6 MIA Raw Water Supply

The raw water stored at the process plant will also be distributed to the MIA via a dedicated raw water storage tank (150 m^3), which is located on a hill adjacent to the process plant alongside the ROM crushed ore stockpile.

Raw water from this process plant storage tank will be gravity distributed, by means of a buried overland 280 mm OD HDPE pipeline approximately 1.8 km in length (see Figure 18.2) , to the raw water tank (350 m^3) and dedicated fire water storage tank (500 m^3), both located at the MIA, which will be used for raw water and fire water distribution, respectively, at the MIA.

18.2.7 MIA Potable Water Supply

The treated raw water at the process plant potable water treatment facility will be distributed to the MIA and surrounding villages via a dedicated potable water storage tank (150 m^3), which is located on a hill adjacent to the process plant alongside the ROM crushed ore stockpile.

Potable water from this process plant storage tank will be gravity distributed by means of a buried overland 90 mm OD HDPE pipeline to the MIA potable water tank (350 m^3). This pipeline will be approximately 1.8 km in length. The potable water will be redistributed throughout the MIA by means of two 37 kW centrifugal pumps (20 m^3/h flow) and also pumped to two dedicated potable water storage tanks (100 m^3 each) located on hills in close proximity to the Rovina and the Merișor villages, respectively. These storage tanks, in turn, will be used to distribute potable water under gravity to the respective villages.

18.2.8 MIA Fire Water Protection System

The MIA will be fitted with a containerised electric and diesel-powered fire water pumping system directly connected to the dedicated 500 m^3 MIA fire water storage tank. The electric-powered pump will be used in the event of a fire, and the diesel pump will be a backup in case of power failure due to the unavailability of MCCs. An electrically operated jockey pump will be provided to maintain the pressure in the fire water header during normal plant runs.

The fire water protection system is fully automated and integrated with the MIA gatehouse control system (whereby alarms will be sounded at the MIA for low system pressure, etc.) and interfaced with fire detection and alarm systems that will be installed at the workshops, tyre store and infrastructure offices.

The fire water pump arrangements will be housed in a containerised unit, complete with its own automated fire control system and dedicated internal sprinkler system.

The fire water system reticulation at the MIA will consist of buried fire water piping loops and strategically placed hydrants, complete with hose cabinets and hose reels, and supplemented with portable fire extinguishers placed within the respective infrastructure facilities.

Foam monitors with foam nozzles will be installed at the fuel depot and dispensing area in the MIA. The diesel receipt fire system consists of a receipt spill slab featuring a fixed water/foam monitor with local foam concentrate tanks.

18.2.9 MIA Fuel Depot

The diesel for the mining fleet and the process plant will be supplied by the fuel supplier and operated by the owner.

The storage of the diesel will be at the mining fleet dispensing station, located in the MIA, where allowance has been made for three days' storage of diesel for usage by the mining fleet and plant support vehicles.

The fuel depot comprises the following:

- Diesel storage tank farm consisting of two 68,000 L self-bunded, all steel construction, double-wall insulated containerised diesel storage tanks, complete with staircases at either end for access to filling points, built-in leak detection and vehicle impact protection, and a hand pump for sampling. The storage tanks will meet all the tank refill/receiving and decanting requirements such as diesel filter arrangements, large water coalescer and cartridges, fill pipe arrangements camlock couplings with Y-strainers and cap featuring fire safe ball valves, anti-syphon valves and check valves with overfill valves as required for safe filling and dispensing.
- Automatic tank gauging/fuel monitoring system that detects water and leakage and supervises real-time tank volume(s).
- Diesel pumping system, which will be located within the containerised diesel storage tank arrangements and comprise duty and standby loading pumps, complete with diesel filters to 5 µm with a coalescer for heavy vehicle/truck high-flow filling at 275 L/min, and low-flow dispensing for light vehicles at 80 L/min.
- Four high-flow and two low-flow metered diesel dispensing stations all situated on dedicated islands for heavy and light vehicles, respectively, and located 20 m from the self-bunded diesel storage tanks, complete with 12/24 V pump kit(s) and including all interconnecting piping, pressure relief and isolation valves, heavy-duty stop nozzle arrangements, automatic shut-off and flow-stop devices, 4 m (minimum) diesel hose, and suitably sized positive displacement meter resettable analogue register for data acquisition.
- Diesel fuel dispensing software to interface with dispensing controllers.
- Designed, built, and certified in accordance with CE and ISO standards; tanks built, tested and certified to UL142 standards as a minimum.
- Dedicated store provided for storage of all vehicle lubricants.

18.2.10 Explosives

The blasting activity will be subcontracted to an authorised firm according to the requirements of the Romanian national regulations. Explosive emulsions, boosters, wicks, etc. will not be stored on the site as only non-explosive materials or equipment necessary for blasting can be stored on site.

18.2.11 MIA Communications

A fibre optic network backbone will be provided, interfacing with all the infrastructure and systems and enabling data acquisition and communications. Infrastructure buildings will be equipped with data trunking and data outlets to facilitate data communications, including voice-over-internet-protocol for telephonic communications.

The mine management information and communication systems will be specified by ESM.

18.2.12 Off-Site Infrastructure Power Supply and Distribution

Power reticulation will consist of the following nominal voltages for the electrical systems and comply with IEC 60038:

- 20 kV ± 5 %, three-phase, three-wire, solidly earthed

- 400/230 V ± 5%, three-phase, four-wire, solidly earthed (TN-S system)

The system's three-phase fault levels will not exceed the following values:

- 20 kV system: 21 kA
- 400 V system: 65 kA

The off-site infrastructure power supply and distribution will cater for the following:

- ROM primary crushing
- MIA infrastructure
- WWTP
- Raw water pump station
- Waste facility conveyors

The power for the off-site infrastructure will be supplied from the main intake substation located at the process plant area. A buried 20 kV power line will be routed down the valley next to the overland conveyor to service the ROM primary crushing area substation. From the ROM primary crushing area substation, another 20 kV power line will be routed along the Colnic to Rovina pit haul road feeding the MIA and tie into a ring main unit that will serve as a termination point for the future power line extension to the Rovina pit.

The first Colnic pit dewatering substation will be positioned at the haul road entrance to the pit and shall feature a separate 20 kV power line feed main intake substation. This substation will be sized for the required future load when the mining waste rock conveyor systems are installed. From this intake substation, a power line will be routed through the underground water diversion tunnel to the WWTP located at the catchment dam. The reticulation at the WWTP will also supply power to the raw water pumping station also located at the catchment dam.

18.2.12.1 Power Demand

The average annual power demand for the off-site infrastructure will be **7,289 kW** during the mining phase of the Colnic pit and will increase to **8,079 kW** when the mining activities move to the Rovina pit due to both pits requiring dewatering for the same period. The loads are given in Table 18.3.

Table 18.3: Average Annual Power Demand

Project Load	Colnic Pit Average Annual Power Demand (kW)	Rovina Pit Average Annual Power Demand (kW)	Source of Supply
MIA	412	412	Buried power line (1.5 km)
WWTP	230	230	Buried power line (0.8 km)
Raw water pump station	510	510	At WWTP
Waste facility conveyors	5,347	5,347	Buried power line (1 km)
Mining and dewatering – Colnic pit	790	790	Buried power line (1.2 km)
Mining and dewatering – Rovina pit	–	790	Buried power line (1.8 km)
Totals	**7,289**	**8,079**	

18.2.12.2 Lighting

The lighting for the mining activities will be provided by using mobile diesel power lighting rigs. These rigs will be operated by the mining personnel and be moved to their required locations to provide adequate lighting as and when required.

High mast lighting will be available at each of the pit dewatering pump station substations, MIA, and water treatment plant to provide area lighting. The high masts will also serve a dual function by providing lightning protection to the surrounding infrastructure and personnel.

18.2.12.3 Earthing

The earthing for off-site infrastructure will be established through a localised equipotential earth system at each location. The earthing system will be formed using a combination of buried 70 mm bare copper earth conductors and grounding rods to achieve the desired resistivity values.

18.3 WASTE MANAGEMENT FACILITIES

The general arrangement of the waste management facilities is shown in plan in Figure 18.7.

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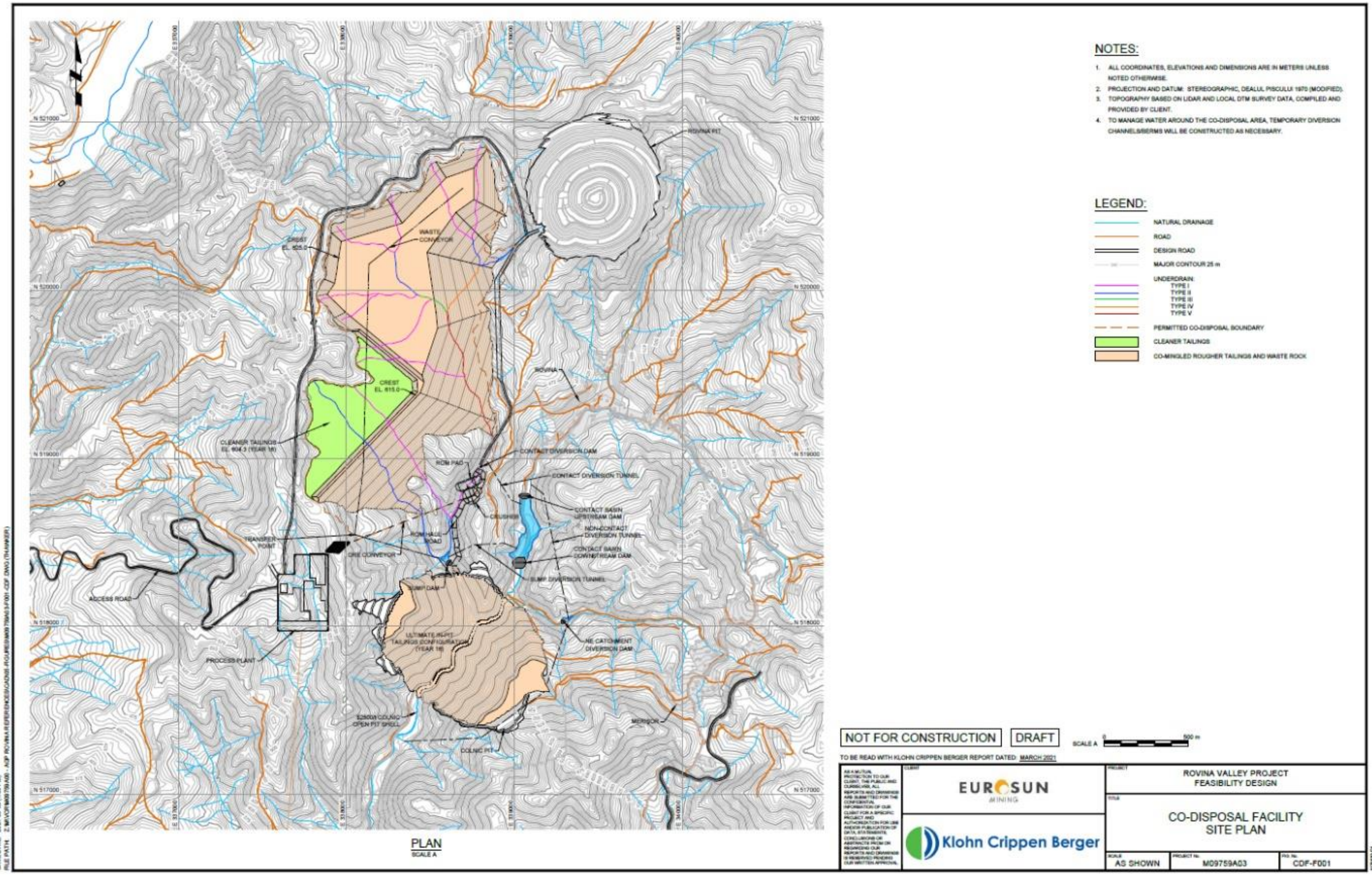


Figure 18.7: Waste Management Facilities – Plan

18.3.1 Co-Disposal Facility

The co-disposal facility (CDF) is designed to maximise storage capacity for the wastes processed from the Colnic pit and Rovina pit within the permitted boundary for waste storage. The processed wastes consist of cleaner tailings and co-mingled waste rock and rougher tailings (co-mingled waste) transported via conveyors to the CDF.

The CDF configuration extends north–south between the Rovina pit and the Colnic pit and stores cleaner and co-mingled waste on an annual basis to accommodate the production schedule. The geometry of the CDF footprint was constrained to an area already permitted by ESM, bounded by the access and haul road alignments, and was developed with a design philosophy to store all cleaner tailings in one area.

Underdrains will be constructed in stream beds within the footprint of the CDF to collect groundwater seepage and precipitation infiltration, and thereby draw down the phreatic surface within the CDF.

18.3.1.1 Cleaner Tailings Cell

Cleaner tailings will be transported to the CDF via a covered conveyor segregated from the rougher tailings stream. Cleaner tailings will be deposited at the southwest corner of the CDF in a segregated, lined cell throughout the LOM. The liner will consist of a primary geosynthetic liner, with a secondary low-permeability soil liner.

Cleaner tailings will be placed in horizontal lifts in the south corner from the start of production to Year 16 as per the annual production schedule. The co-mingled rougher-scavenger tailings and waste rock will be used to construct a stable perimeter around the cleaner tailings cell in advance of each year's placement to provide sufficient storage capacity for the cleaner tailings. Placement of each year's liner extension will be undertaken in parallel with the cell raising.

At project initiation (prior to tailings or waste placement), a starter embankment will be raised at the location of the cleaner tailings cell. This starter embankment will provide the storage capacity for Year −1 production of cleaner tailings. This starter embankment will be constructed from locally sourced material as the material used in the construction of the starter embankment will need to be non-potentially acid generating (Non-PAG) and non-ML.

18.3.1.2 Co-Mingled Waste Storage

Waste rock sourced from the Rovina pit and the Colnic pit will be delivered via an uncovered conveyor to a transfer point where it will be mixed, on the conveyor, with the rougher-scavenger tailings that have been delivered from the plant to the transfer point via a covered conveyor. From the transfer point, the co-mingled waste will then be transferred via a covered conveyor to the CDF.

The annual co-mingled waste staging plan is as follows:

- **Year −1 to Year 3:** Annual stages to be constructed by downstream raises with an approximate upstream slope of 2H:1V and an approximate downstream slope of 2.5H:1V.

- **Year 3 to Year 9:** Annual stages to be constructed to a constant elevation of 625 m from south to north, filling up the valley with an approximate downstream slope of 2.5H:1V.

The co-mingled waste will be placed in horizontal layers from ground to crest (bottom to top) to the required elevation. To meet the short-term (during construction) and post-earthquake stability criteria, a downstream shell of rougher tailings compacted in horizontal layers to a dilative condition will be constructed.

18.3.1.3 Stability Analyses

Stability analyses were conducted for sections across the CDF selected along the valley sections to provide maximum fill height and steep bedrock slopes which represent critical conditions for the CDF stability.

The minimum target factors of safety (FoS) for geotechnical stability of the CDF are as follows:

- Static, Long-Term FoS: 1.5
- Static, During Construction FoS: 1.3
- Post-Earthquake FoS: 1.2

Preliminary seepage analyses were conducted along the chosen sections based on assumed hydraulic conductivities for the fill and foundation materials. Laboratory testing on three rougher tailings samples indicated a permeability of 4×10^{-7} m/s, recognising that the field value may differ from the laboratory value, with the field value potentially being greater than the laboratory value. The seepage analyses indicated that the phreatic level in the CDF fills is sensitive to the assumed hydraulic conductivity of the co-mingled waste and the assumed infiltration. In comparison to the phreatic surface obtained from the seepage analyses based on the laboratory-derived permeability, a higher phreatic level was assumed for the DFS design stage. This higher phreatic surface assumed for the CDF was based upon a review of the performance of another filtered tailings disposal facility in a wetter climate. The design piezometric level was assumed to be at a height of 30 m above ground, below the crest of the CDF, and then falls towards the drain constructed along the toe of the CDF.

The stability analyses for two of the three loading conditions were conducted with strength and phreatic level assumptions as follows:

- The static, long-term stability analysis was conducted with drained strength parameters for all materials.
- The post-earthquake stability analysis was conducted with undrained strength parameters for co-mingled waste with different strength ratios for uncompacted and compacted materials.

Due to limitations on the available laboratory testing data, it was difficult to predict to what degree excess pore pressures would build up during construction of the CDF. Based upon the oedometer testing on the rougher tailings, however, it is expected that some build-up in excess pore pressures could occur. To allow sufficient dissipation of such pore pressures within the structural toe zone of the CDF, it may be necessary to adopt a staging plan that allows construction of this structural toe zone in advance of the higher parts of the slope and to possibly flatten the temporary north-facing slope as it advances northwards. A detailed construction staging plan, in conjunction with stability analyses for the during-construction condition, will be undertaken during the next design stage of the CDF.

The CDF meets the stability criteria given the following design conditions:

- Residual soils are removed during the foundation preparation activities, and the CDF is founded on bedrock.
- The operating and final slopes of the CDF are maintained at no steeper than 2.5H:1V.
- The downstream shell of rougher tailings is built of compacted layers to provide a dilative, drained strength response for short-term and post-earthquake conditions.
- The phreatic level in the CDF does not exceed the phreatic level assumed.
- The geometry and rate of rise of the CDF is controlled to limit the excess pore pressure generated during construction and enhance dissipation rates within the outer structural zones.

18.3.1.4 Foundation Preparation

The foundation preparation beneath the CDF will involve excavation of topsoil and residual soil to expose competent bedrock foundation prior to fill placement. Foundation preparation is to be completed at least a year in advance of placement in order to facilitate the construction of underdrains before the placement of the co-mingled waste. Material from excavation and stripping should be stockpiled for future use in designated areas, including for use in the progressive reclamation of the mine areas, such as the finished outer slopes of the CDF.

18.3.1.5 Liner Protection

As the cleaner tailings material is potentially acid generating, the bottom and top of the cleaner tailings cell will be lined with a combined soil/geosynthetic liner, constructed as follows:

- Upper layer: Geomembrane (minimum 1.5 mm (60 mil) thickness)
- Lower layer: Compacted clay (minimum 300 mm thickness)

This combined liner will be installed at the base of the cleaner tailings cell prior to start of operation and will be expanded each year prior to the next year's placement of material. Following the final lift of cleaner tailings, the top surface of the final lift of the cleaner tailings cell will also be lined.

18.3.1.6 Seepage Control and Underdrainage

The seepage control system was designed to collect seepage from the CDF and convey it to the contact water management system. Underdrains will be constructed to collect both water that enters the base of the CDF as seepage from the stream beds and precipitation infiltration through the CDF. Base flow estimates from the streams within the footprint of the CDF to allow prediction of the seepage rates and tailings testing to allow prediction of infiltration were not available for the DFS design work.

The critical period for the underdrain flows will be prior to the placement of the waste material, when runoff from the upstream catchment can directly enter the drains. As the CDF is advanced, runoff from the natural catchment will be diverted away from the underdrains to the maximum extent practical by a series of temporary diversion ditches and catchment berms. It is expected, however, that some flow cannot be practically diverted, and this flow will be captured and led into the underdrainage system in a controlled manner. The underdrains have therefore been sized for a storm event with a return period of 1/10 year and a 1-hour intensity derived from the total 24-hour precipitation. An FoS of 3.0 has been applied to the sizing of the underdrains to account for groundwater seepage, runoff from the natural catchment and other uncertainties, as well as for potential reductions in the conveyance capacity of the drains over time.

Underdrains will be constructed by the placement of select, durable, non-acid generating screened rockfill. Above the rockfill, two filter layers will be installed consisting of gravel in the lower layer and sand in the upper layer.

To minimise flow from the natural catchment into the underdrainage system, underdrains will be constructed only a short-time in advance of the placement of co-mingled waste.

18.3.1.7 Construction Methodology

The construction methodology for the CDF is summarised as follows:

- Pre-stripping required for the subsequent year's construction will be completed a year in advance.
- Diversion of the runoff from the natural catchment will be established to minimise clean water entering the underdrainage system and to minimise the potential for erosion at the toe of the CDF.
- Underdrains will be constructed in advance of the placement of co-mingled waste, with the time period between placement of the underdrains and covering with co-mingled waste being as short as practicable. The drain geometry will be adjusted in the field to accommodate the stream geometry in order to achieve the required drainage area.
- Cleaner tailings and co-mingled waste will be conveyed via separate streams of conveyors from the plant to the CDF.
- Cleaner Tailings will be placed in one location throughout the LOM, at the south corner of the CDF. The base of the CDF, the upstream face of the starter fill and ongoing fill raises, and the surface of the final lift of the Cleaner Tailings cell (at closure) will be lined. The liner will comprise a primary geosynthetic liner (upper) and a secondary compacted clay liner (lower).
- A Starter Embankment will be constructed a year in advance of Cleaner Tailings placement in Year −1, to allow for initial storage of the cleaner tailings. This Embankment will have an upstream slope of 2H:1V and downstream slope of 2.5H:1V.
- The final downstream slopes for the CDF will be built with benches and steeper inter-bench slopes to form an overall slope of 2.5H:1V.
- For the first three years, co-mingled waste will be placed at the south end of the CDF before switching to fill the valleys, proceeding south to north. Interim stages of co-

mingled waste are built at a 2.5H:1V slope (or flatter as required by during-construction stability).
- The outer structural compacted shell of the CDF will be built of compacted layers of rougher tailings to provide drained and dilative behaviour under short-term and post-earthquake conditions.
- The inner areas of the CDF will be constructed using co-mingled waste and are not expected to be formally-compacted other than for equipment accessibility and to reduce infiltration.
- All materials will be placed and compacted (where required) in horizontal layers from ground to crest (bottom to top).

A detailed review of construction staging and methodology, should be conducted in future studies.

18.3.1.8 Closure and Reclamation

The reclamation strategy for the CDF involves placing a 0.5 m thick soil and vegetative cover followed by grass mixes for long-term cover. The reclamation strategy will be executed separately for the cleaner tailings and co-mingled waste:

- For the cleaner tailings, the closure design involves the installation of a liner over the exposed tailings surface after Year 16. The liner will consist of a primary geomembrane and secondary clay layer with an appropriate soil and vegetative protective cover to encapsulate the cleaner tailings.
- For the co-mingled waste areas, progressive reclamation will be carried out as soon as the final CDF exterior slopes are established, and will consist of applying a vegetative cover to these slopes. This activity will be executed as an on-going operation on the final exposed slope.

18.3.2 Colnic Pit Infill

The Colnic pit will be backfilled with co-mingled rougher-scavenger tailings and waste rock once the storage capacity of the CDF is reached, following Year 9. Cleaner tailings will continue to be stored at the CDF during this period. The geometry of the Colnic pit infill was developed to allow for the return of the Rovina Valley drainage to its natural course at project closure; a commitment made by ESM in project permitting. To achieve this arrangement, the CDF was designed to allow for a 20 m wide channel to pass through the facility, and with slopes that mesh well with natural topography at closure.

18.3.2.1 Co-Mingled Waste Storage

Waste rock sourced from Rovina pit will be delivered via an uncovered conveyor to a transfer point where it will be mixed, on conveyor, with the rougher-scavenger tailings. The co-mingled waste will be transferred via a covered conveyor to the Colnic pit infill.

The annual co-mingled waste staging plan is as follows:

- **Year 10 to Year 12 (Filling to Channel Bottom):** Colnic pit backfilling begins in Year 10, filling the pit with the co-mingled waste material until placement reaches the design channel bottom, mid-year in Year 12.

- **Year 12 to Year 13 (Infilling the Far Side):** The area southeast of the channel is then filled starting mid-year in Year 12. By mid-year in Year 13, placement in this area is complete.
- **Year 13 to Year 16 (infilling the Near Side):** Once the southeast side of the channel is filled, placement begins on the northwest side of the channel (mid-year Year 12), until completion of filling.

Some limited areas outside of the pit boundary will have co-mingled waste placed in them. Downstream slopes for the Colnic pit infill will be built with benches and steeper inter-bench slopes to form an overall slope of 2.5H:1V.

18.3.2.2 Foundation Preparation

The foundation preparation will be carried out beneath the areas of co-mingled waste placement outside the Colnic pit and will involve excavation of topsoil and residual soil to expose competent bedrock foundation prior to fill placement. The foundation preparation will be completed a year in advance to allow for clearing, grubbing, stripping, and ground preparation activities. Material from excavation and stripping should be stockpiled for future use in progressive reclamation of the exposed slopes.

18.3.2.3 Closure and Reclamation

At closure of the Colnic pit infill, after Year 16, the reclamation strategy involves placing a 0.5 m thick vegetative cover followed by grass mixes for long-term cover to the final exterior slopes. This activity will be executed annually for a portion of the final exposed slope after each annual raise. A final closure rehabilitation programme will be executed for the co-mingled tailings area slopes and crest after Year 16.

After completion of the Colnic pit infill, and as part of the Closure operations, the 20 m wide channel will re-establish the Rovina Valley stream flow back to the streambed downgradient of the Colnic pit.

The construction of the channel for the Rovina Valley drainage has been developed with a nominal width of 20 m and channel side slopes of 2.5H:1V. For the chosen channel cross-section, the longitudinal shape of the channel was developed with a sinuosity that is reflective of the natural channel that existed prior to excavation of the Colnic pit. The channel sinuosity was implemented in the design in an effort to manage the velocity of the water routed through the channel, thereby reducing the potential for excessive erosion, as well as to restore the original fluvial geomorphic processes that took place prior to construction.

18.3.3 Geochemistry

18.3.3.1 Geochemical Characterisation of Mining Wastes

Geochemical characterisation of mining wastes is concerned with the influence that the weathering of waste rock, process residues, such as flotation tailings, and pit walls might have on water quality in the downstream environment. For sulphide orebodies, a major concern is the potential for ARD and ML. Geochemical characterisation studies are carried out to assess these potentials and involve specialised tests and analyses to classify whether a mined rock or process residue is potentially acid generating (PAG), acid consuming (non-

PAG), and in either case, whether the material could present a risk to water quality. The results of studies are used as inputs to the design and operation of waste and water management systems, and to assess the need for mitigation measures.

18.3.3.2 Characterisation of Waste Rock and Low-Grade Ores

Pyrite, FeS_2, is often the most prevalent mineral in a deposit responsible for ARD due to its potential oxidation to sulphuric acid, which creates conditions in which ML can be exacerbated, although some elements are mobile in neutral pH conditions. Typical of porphyry deposits, the Colnic and Rovina orebodies have phyllic alteration haloes containing disseminated pyrite, some of which will be present in the waste rock. To characterise rock that will be mined, geochemical static tests were carried out on over 300 samples of Colnic and Rovina waste rock and low-grade ore selected from exploration drill core based on lithological and alteration abundance (LCL, 2012). Samples were found to be predominantly potentially PAG, with Colnic rock having a higher percentage of PAG rock than the Rovina rock. The low-grade ore samples generally had a slightly higher ARD potential than the waste rock samples.

Additional kinetic testing of 24 samples of mostly PAG waste and low-grade ore was carried out in laboratory humidity cells for up to 60 weeks to assess the rates of oxidation, neutralisation, and metal release (LCL, 2013 and 2014). Test leachates remained neutral, and leachate chemistry became stable or with decreasing parameter concentrations for both waste rock and low-grade ore. Elements of possible environmental concern that were present in leachates included aluminium, manganese and selenium. The oxidation and neutralisation rate data suggested that the generation of ARD is possible but that the delay to the onset of acidic conditions could be estimated to be measured in years or tens of years due to the low rates of sulphide oxidation and the sufficiently high carbonate content of the rock. Sulphide oxidation rates were generally declining during testing, and it is likely that many of the lithological units might not produce ARD, or at least not during the life of mining operations. Waste pile management to mitigate ARD in the long term should be readily achievable.

Tests on low-grade ore are carried out to assess the possibility that stockpiles could become the source of ARD and contaminants to the environment. Since pit walls will contain exposures of some ore-grade material, the rates of metal release from the exposed walls during and after mine closure can also be assessed. Estimates of metal loadings and water quality from the pit walls can be determined when more project details become available during basic and detailed engineering.

18.3.3.3 Characterisation of Flotation Tailings

A geochemical static test programme was carried out to evaluate samples of the tailings produced in a flotation test programme on the Colnic and Rovina ore composites (LCL, 2021). The results showed that the rougher-scavenger tailings are predicted to be non-PAG, and their disposal will not produce acidity or significant ML. The cleaner tailings, however, are strongly PAG, will apparently oxidise readily, and will represent a high risk for the generation of acidic drainage containing significant metal loadings. Based on the flotation mass balances, a combined tailings stream would also be PAG.

18.3.3.4 Geochemistry of the Waste Disposal Facility

As described in Section 18.3, the rougher-scavenger and cleaner tailings streams produced in the ore processing plant will not be combined for disposal. The non-PAG rougher-scavenger tailings will be co-mingled with the waste rock before disposal in a waste disposal facility. The PAG cleaner tailings will be disposed of in a dedicated zone within the waste facility.

Co-mingling the non-PAG rougher-scavenger tailings with the waste rock prior to deposition and compacting it after placement will be beneficial in lowering the overall sulphide content and adding neutralisation capacity. The tailings will also reduce the permeability of the waste rock-tailings mix, which will be further enhanced by compaction after placement. These factors will lower the risk of acid generation and associated ML from the waste rock, at least in the short to medium term during operations, even in the absence of mitigation measures. The application of mitigation measures, including progressive rehabilitation and/or rehabilitation of the waste facility at the end of operations, should prevent ARD in the long term. In any event, the waste facility design includes the management of surface and seepage water and the provision for collection and treatment of contact water to prevent the release to the environment of any contaminated water that might be produced both during operations and post-closure.

While oxidation of the cleaner tailings in the dedicated zone of the waste facility can be expected, reactions will be limited to the exposed surface layer due to a very fine particle size and will be further mitigated by the facility design, operational practice, and a number of geochemical factors. The disposal area will be lined, and surface runoff and any seepage will drain to a dedicated collection pond for treatment as required. The zone will also receive a permanent cover upon completion of the operations to close out the facility.

18.4 WATER MANAGEMENT

The RVP is located in an area characterised by moderate, forested slopes with significant vegetation, in the form of grasses, shrubs, and tree coverage. The project infrastructure is located in two major valleys, with the eastern valley (Rovina Valley) containing a significant stream which crosses the planned location of the Colnic pit.

The Colnic pit is located at the confluence of a few small streams in addition to the main Rovina Valley drainage. Surface water management upstream of the Colnic pit is focussed on diverting water away from the pit and preventing flooding of the Colnic pit.

The Rovina pit is located at the headwaters of the western Rovina Valley; there are no significant watercourses intercepted by the Rovina pit and runoff into the pit will be managed by pumping from sumps within the pit.

The general arrangement of the water management facilities is shown in plan in Figure 18.8.

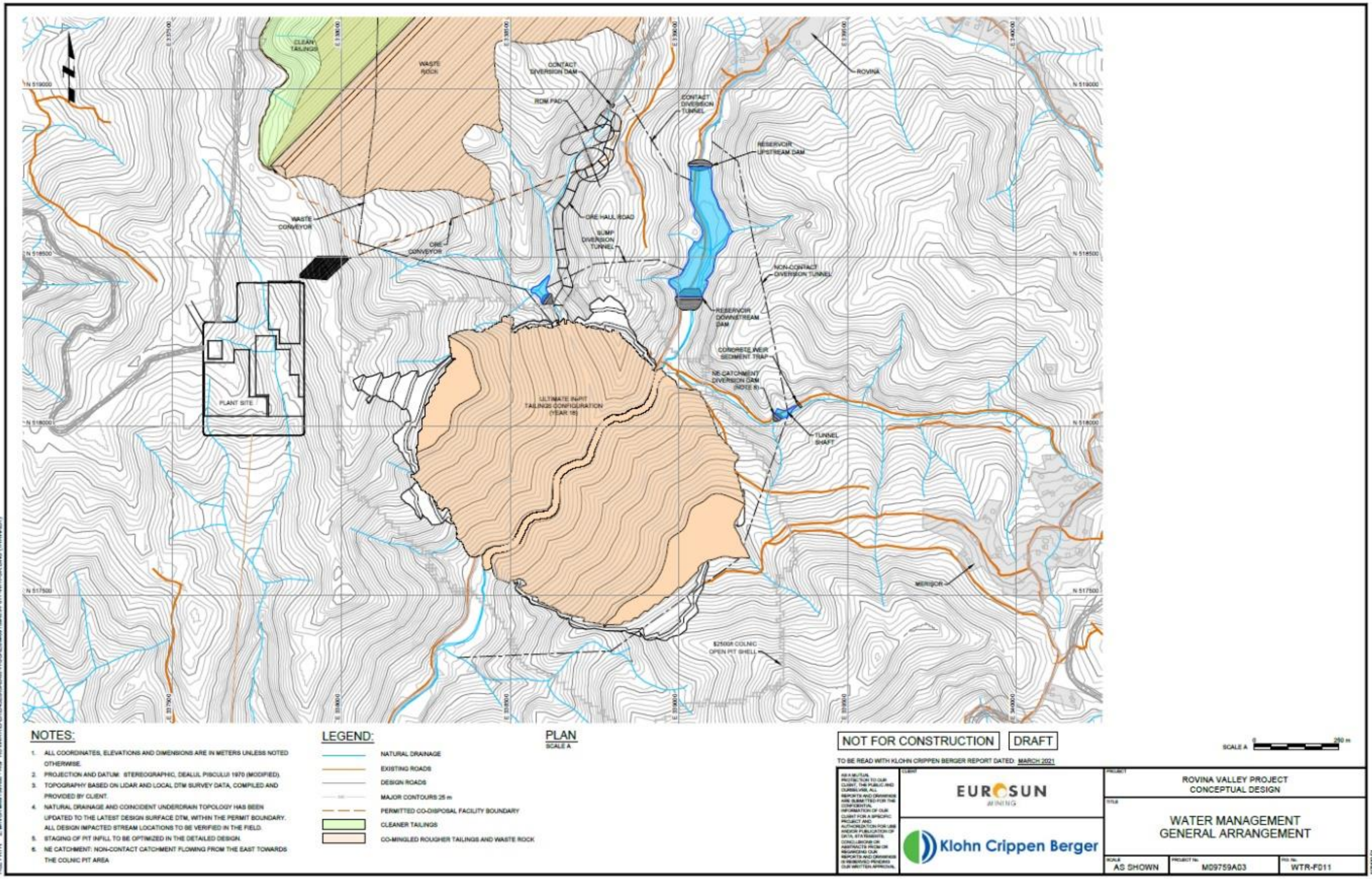


Figure 18.8: Water Management Facilities – Plan

18.4.1 Water Management Infrastructure

The water management infrastructure at the RVP is categorised as managing either of the following:

- Non-Contact Water: Runoff that originates from natural ground or areas undisturbed by the project.
- Contact Water: Runoff that originates from mine development areas such as the CDF.

18.4.1.1 Non-Contact Water Management

The proposed non-contact water management plan consists of diverting the two natural catchments located upstream of the mine operations:

- The Rovina Valley and its catchment (approximately 8.9 km^2)
- The Merișor catchment, located northeast of the Colnic pit (approximately 1.1 km^2).

Drainage from the Rovina Valley catchment will be diverted around the mine infrastructure, including the Colnic pit, through the non-contact diversion tunnel, which discharges into the original Rovina Valley drainage, approximately 280 m downstream of the Colnic pit.

In addition to the Rovina Valley stream flow, the non-contact diversion tunnel will receive flows from the Merișor catchment. The diversion arrangement for this catchment consists of a diversion dam and a shaft excavated through rock to intercept the non-contact diversion tunnel. Upstream of the shaft, there is a concrete weir that acts as a sediment trap to reduce the sediment loading to the shaft and tunnel.

18.4.1.2 Contact Water Management

The proposed contact water management system consists of the following

- The contact water pond, a collection pond located in the Rovina Valley (between the non-contact water diversion tunnel and the Colnic pit).
- Conveyance structures to route water from the contact water catchments to the contact water pond.
- A pump station to reclaim contact water for mine processes from the contact water pond.
- A treatment plant to treat water for mine processes or for release to the natural environment, downstream of the pit (discharged via the non-contact diversion tunnel, following treatment).

Excess water from the pits will be pumped to the contact water pond. Runoff from the CDF will be conveyed to the contact water pond via tunnels. Outflows from the contact water pond will be actively managed through pumping to meet the demand of the processing plant, as well as to remove excess water for treatment and discharge to the environment.

18.4.2 Conveyance Structures

Sizing of the tunnel and channel has been developed based on a routing assessment to confirm that these are sufficiently sized to convey peak flows from the selected

Environmental Design Flood and Inflow Design Flood events from the respective catchments. The estimated hydrographs were used to size the water conveyance structures. The tunnels will be concrete lined.

18.4.3 Contact Water Pond

The contact water pond will receive flows from mine developed areas as well as dewatering pump flows from the pits. The pond will be contained by an upstream dam and a downstream dam, and by high topography on the east and west portions of the pond. The pond and dams were sized with consideration given to pumped outflows and additional storage requirements potentially required under a dry-year scenario.

The downstream dam will have a spillway to safely route excess flows that cannot be contained by the contact water pond. A spillway will also be located in the upstream dam as part of the contingency system (see Section 18.4.4). However, this upstream dam spillway will have a higher invert elevation than the downstream spillway, and the upper spillway is, therefore, only expected to function in the case of a rare event (see Section 18.4.4).

18.4.4 Contingency Considerations

The assumed design event for the non-contact diversion tunnel was the peak flow associated with an event with a return period of 1/1,000 years, but the tunnel was also sized to pass peak flows estimated for events with return periods up to 1/10,000 years.

In addition, given the importance of the non-contact diversion tunnel to the overall water management system and site, a contingency plan to prevent failure of the mine infrastructure in the event that the non-contact diversion tunnel is unable to convey the design flows (e.g. due to blockage or collapse of sections of the tunnel) was developed. The proposed contingency plan consists of spillways at the contact water pond dams to safely convey excess flows from the Rovina Valley stream flow and from the mine to the Colnic pit area. The spillway inverts were selected to reduce the potential for backwater of upstream catchments and dwellings.

18.5 LOGISTICS

Logistics and transport studies were conducted to

- Define the possible access routes to site.
- Identify port facilities and capabilities at the point of discharge.
- Determine the most efficient routing and method of transport to site.
- Determine road/bridge upgrade requirements to ensure the safe delivery of all shipments.
- Investigate project insurance requirements.
- Determine total logistics budget to complete the movement to site of all project cargo.
- Complete a methodology to enable control of all movements.
- Establish a shipping procedure specific to the project.
- Identify staff resource requirements along the supply chain and at the project site.
- Determine customs and excise requirements in Romania and their effect on the project programme/budget.

18.5.1 Freight Forwarder

The services of a freight forwarder will be contracted during project execution to ensure timeous receipt of goods on site and include the following:

- Customs documentation
- Sea freight: placing of containers at the suppliers' or nominated warehouse for loading and ensuring the timeous delivery of the container(s) to the relevant port of loading.
- Road freight: the use of suitable vehicles, routing to the final destination, and customs clearance if applicable.
- Airfreight: the use of suitable airline or cargo carrier(s) to the final destination.
- Cross-trade: arranging the collection of goods with the supplier and advising the Procurement Director of the cost and time frame before proceeding.
- Tracking of cargo/container(s).
- The Shipping Controller will be responsible to ensure that all orders are completely delivered with the required documentation.

18.5.2 Road Freight

Refer to Doc. No. SP0829-0000.00-3W35-001 - Logistics Route Survey Brad – Holleman Romania – May 2012 in which specific routes were surveyed for road transportation of goods to the RVP site from the nearest sea and river ports.

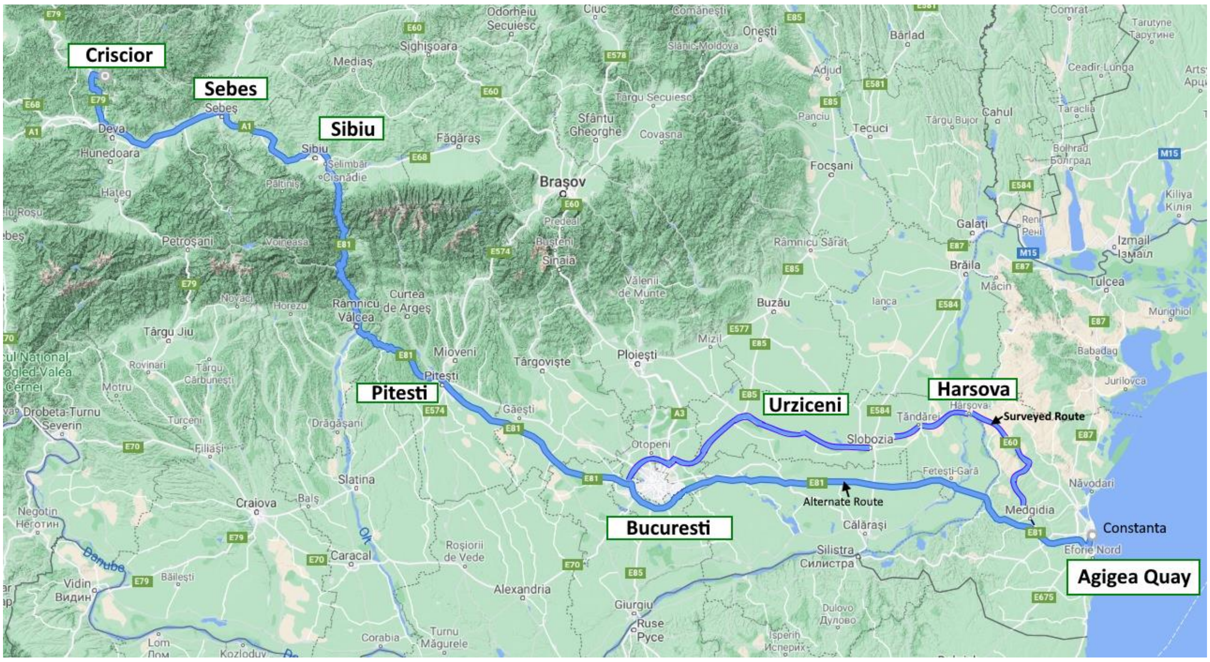
The road transportation routes identified and studied are as follows:

- Port of Constanta to Bucureşci
- Port of Orșova to Bucureşci
- Trans-European Transport Network (TEN-T)

18.5.2.1 Port of Constanta to Bucureşci

Sea port (Black Sea) route to the RVP site (see Figure 18.9):

Agigea Quay → Constanta → Medgidia → Silistea → Tepes Voda → Stupina → Harsova → Tandareni → Slobozia → Urziceni → Sinesti → Afumati → Centura Bucuresti → Titu → Gaiesti → Pitesti → Ramnicu Valcea → DJ703F → DJ703L→ Centura Calimanesti → Baraj Turnu → DN7 → Talmaciu → Centura Sibiu → DN1+DN7 → Salistea → Miercurea Sibiului → Sebes → Orastie → Deva → Brad → Criscior → Bucureşci (Rovina Valley Site) = **total 740 km**.



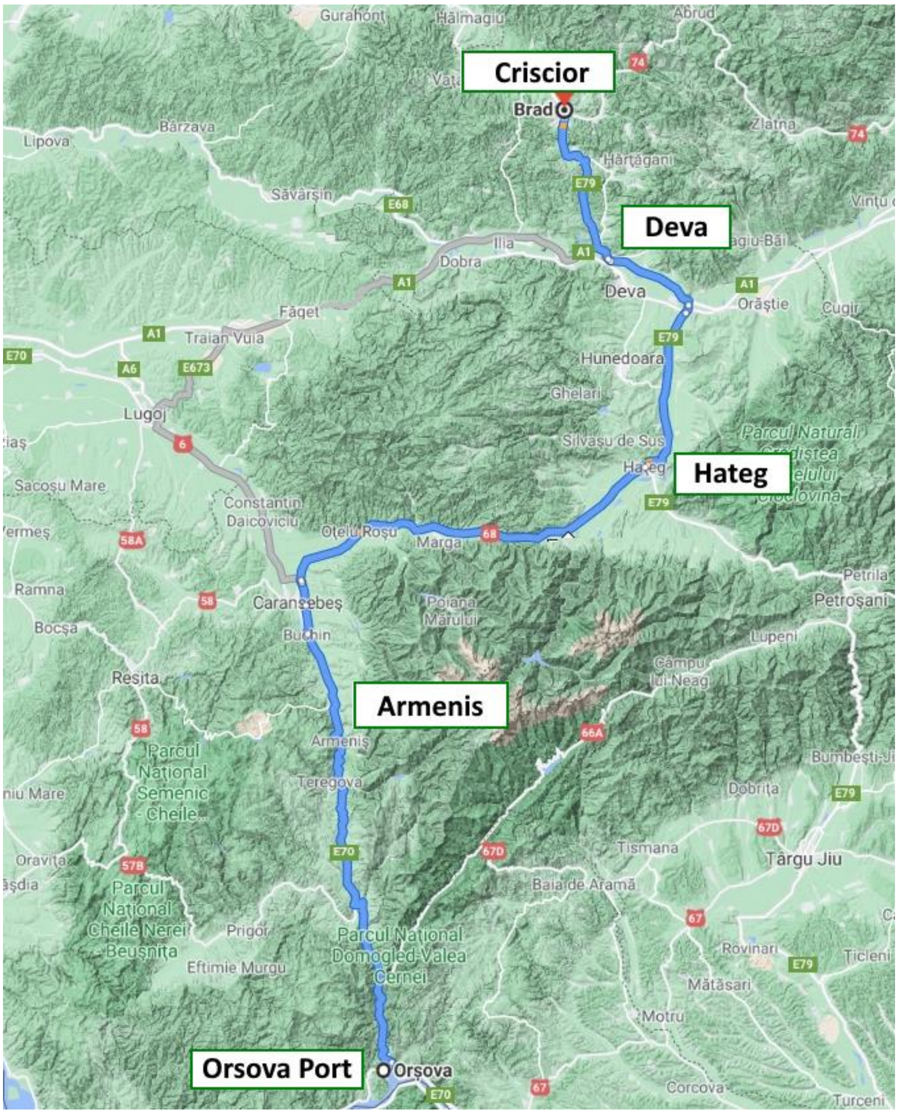
Source: www.holleman.ro

Figure 18.9: Port of Constanta to Bucureşci

18.5.2.2 Port of Orșova to Bucureşci

River port route to the RVP site (see Figure 18.10):

Port of Orșova → Barza → Herculane → Mehadia → Armenis → Buchin → Caransebes → Obreja → Glimboca → Otelul Rosu → Vama Marga → Sarmizegetusa → Totesti → Hateg → Calan → Simeria → Deva → Soimus → Brad → Criscior→ Bucureşci (RVP) = **total 250 km**.

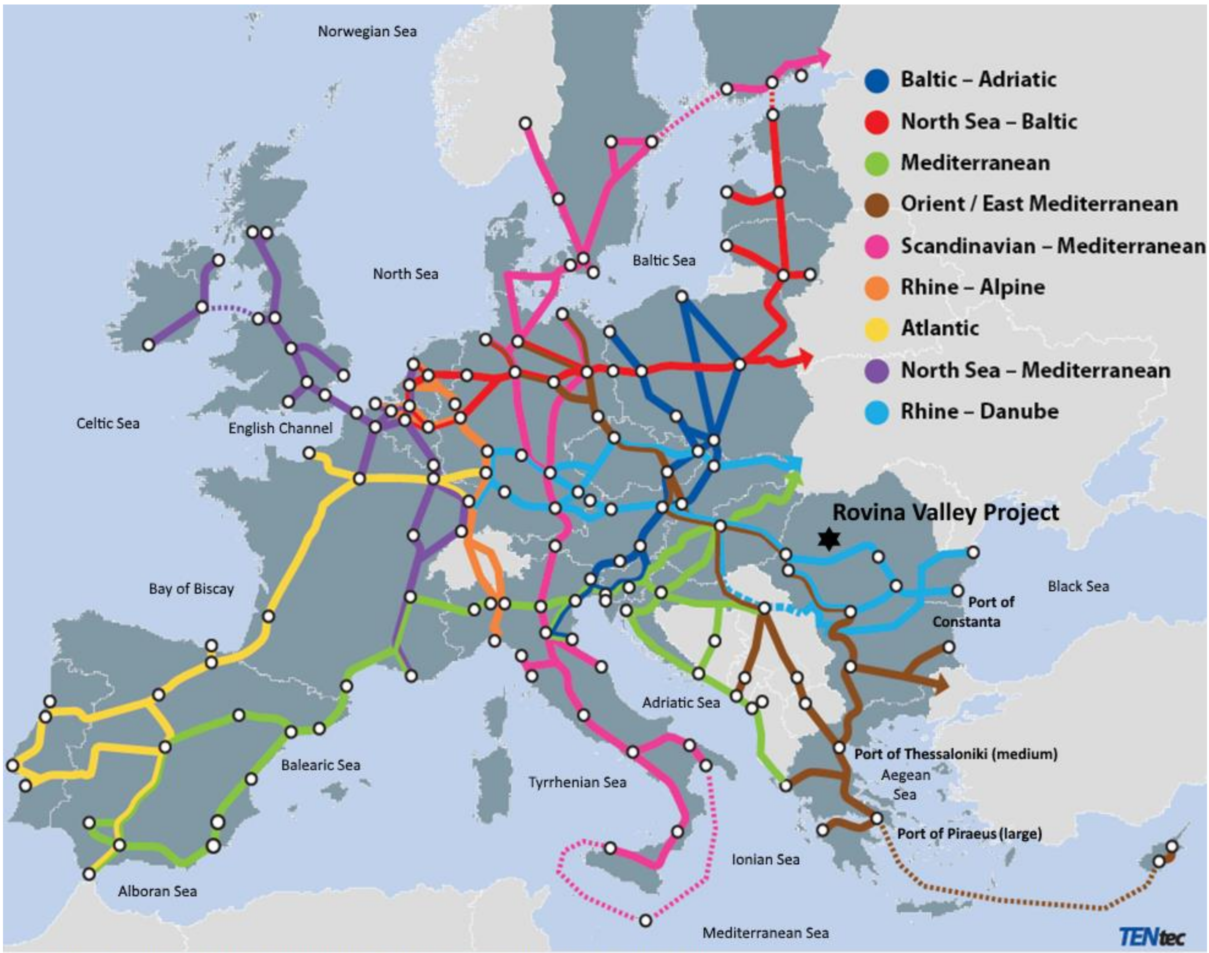


Source: www.holleman.ro

Figure 18.10: Port of Orșova to Bucureşci

18.5.2.3 Trans-European Transport Network (TEN-T)

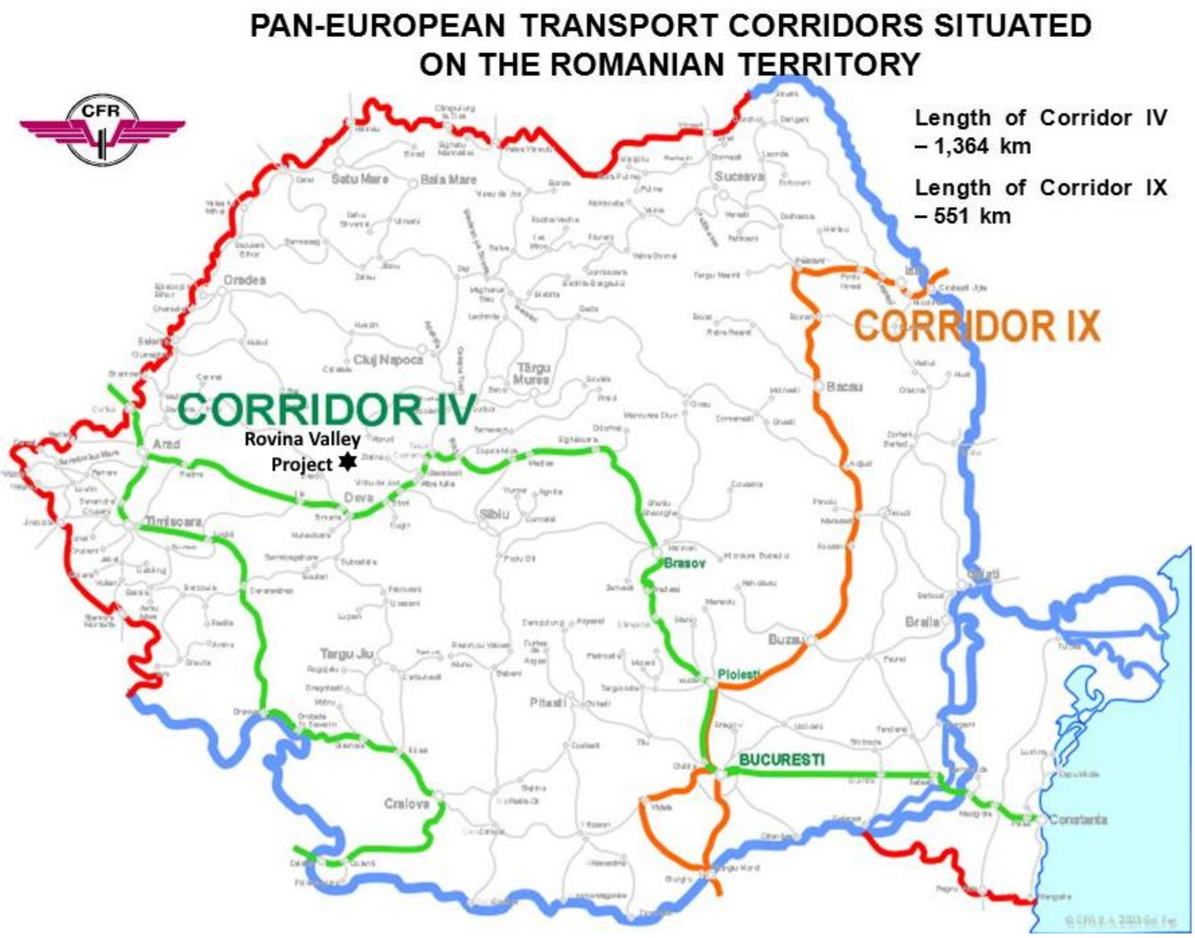
Figure 18.11 shows the extent of the TEN-T transport network in Europe.



Source: European Commission, Directorate-General for Mobility and Transport, TENtec Information System

Figure 18.11: Pan-European Transport Network (TEN-T)

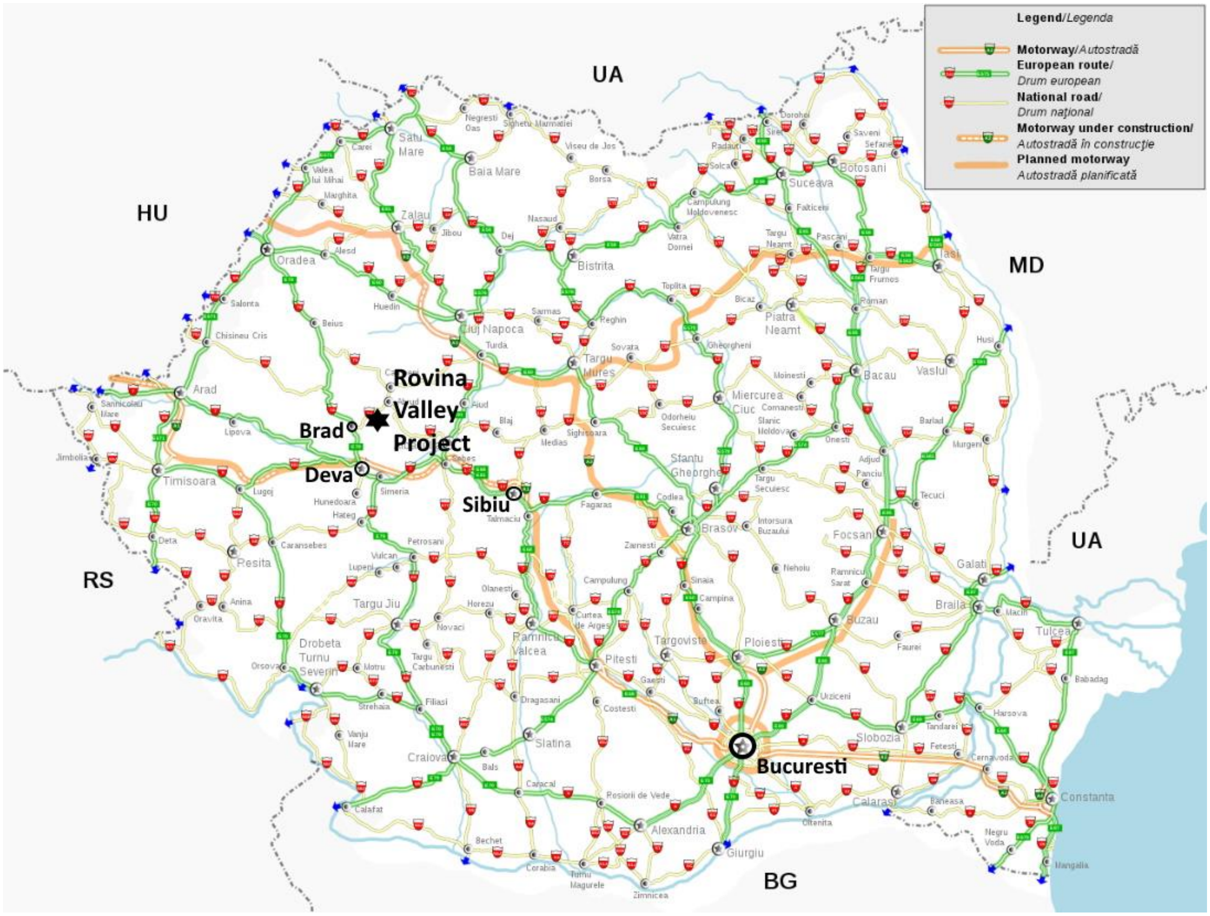
The Pan-European transport corridors situated on Romanian territory are shown in Figure 18.12.



Source: Flanders Investment & Trade Market Survey; Dec 2016

Figure 18.12: Pan-European Transport Corridors Situated on Romanian Territory

The major routes within Romania are shown in Figure 18.13.



Source: https://commons.wikimedia.org/wiki/File:Romania-drumuri.svg

Figure 18.13: Major Road Network – Romania

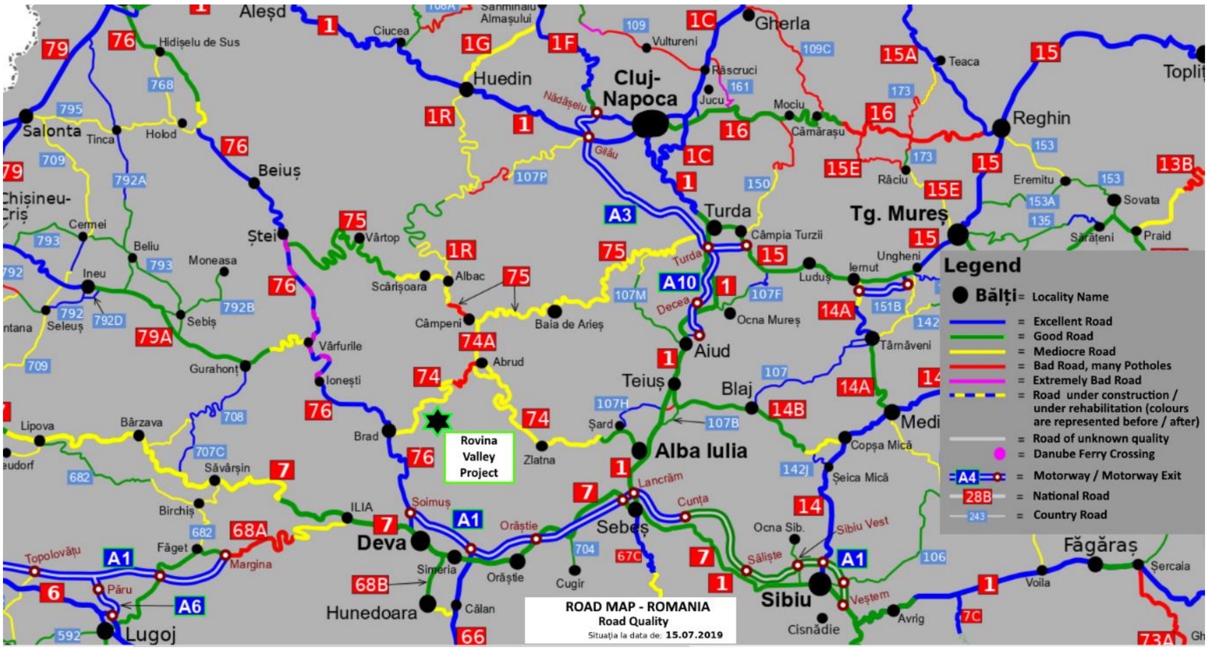
From Sibiu (see Figure 18.14), the principal access to the property is via a four-lane highway to Deva and then another 40 km via a paved two-lane highway leading to the historical gold mining town of Brad, followed by secondary paved roads eastward for 7 km, passing through the town of Crişcior and on to the village of Bucureşci, which is located within the Rovina mine property. These roads provide the principal access to the Rovina and Colnic deposits. The western boundary of the Rovina Licence is located less than 1 km east of the town of Crişcior (population approximately 3,000), where the ESM main office is located.

Source: https://commons.wikimedia.org/wiki/File:Romania-drumuri.svg

Figure 18.14: RVP – Site Access Roads Network

Access to other portions of the Rovina Licence is via various paved and gravel roads, with tracks suitable for four-wheel-drive vehicles, or along footpaths. Access to the mine property by road is possible year-round; however, short periods of blockage are possible in winter due to snow, especially in the higher areas of the Apuseni Mountains.

The quality of the access roads to the RVP site is defined in Figure 18.15.



Source: https://www.reddit.com/r/MapPorn/comments/dx4mi9/road_quality_map_of_romania/

Figure 18.15: RVP – Site Access Roads Quality

18.5.3 Airfreight

In the event that the project programme requires an aircraft to swiftly transport goods into the country, aircraft can be deployed to one of three international airports in close proximity to the RVP site as shown in Figure 18.16. These costs would have to be negotiated at the time of shipment.

The air infrastructure in Romania consists of 14 airports permanently open for traffic. They offer more than 130 direct flights to 76 destinations in 31 countries and also ensure a strong internal connection providing easy access to every region in the country. The most important airports are the ones in Bucharest (major international airport), Timisoara and Constanta (big international hub airports), which are operated by the Romanian government. The others are operated by local counties.

The project lies near three international airports (see Figure 18.16) within approximately a 2 h to 3 h drive from the cities of Sibiu, Targu Mures (or Timisoara), and Cluj Napoca. The city of Sibiu has the nearest international airport to the property with the most regularly scheduled commercial flights from various European destinations.

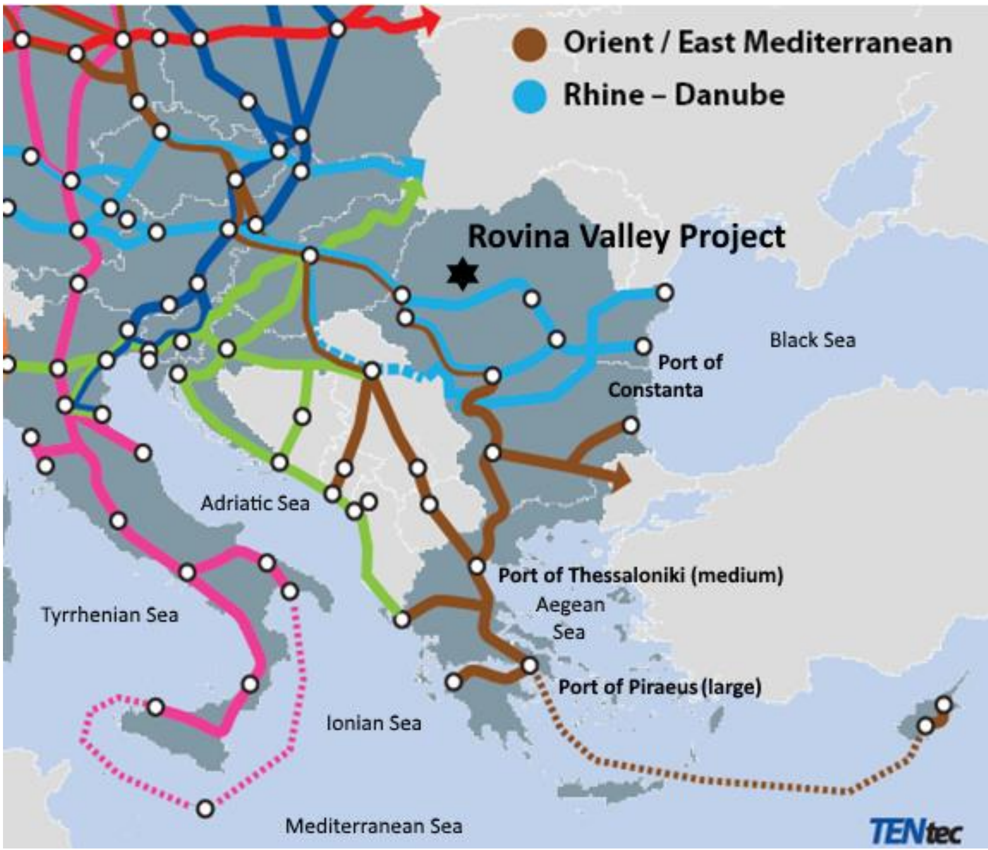
Source: https://commons.wikimedia.org/wiki/File:Map_of_airports_in_Romania.svg

Figure 18.16: Airports – Romania

18.5.4 Sea Freight and Port Facilities

The Port of Thessaloniki is one of the largest Greek seaports and one of the largest ports in the Aegean Sea basin, with a total annual traffic capacity of 16 Mt. The Port of Thessaloniki contains the second largest container port in Greece, after the Port of Piraeus.

The Port of Constanta (see Figure 18.17) is located in Constanta, Romania, on the western coast of the Black Sea, 179 nautical miles from the Bosphorus Strait and 85 nautical miles from the Sulina Branch, through which the Danube river flows into the sea.



Source: https://commons.wikimedia.org/wiki/File:Pan-European_corridors.svg

Figure 18.17: RVP – Nearest Sea Ports

18.5.4.1 Port of Constanta

Port specifications (source: www.portofconstantza.com):

- Water location: Black Sea
- Anchorage depth: 23.2 m
- Cargo pier depth: 6.4 m to 7.6 m
- Oil terminal depth: 2.5 m to 13.7 m
- Dry dock: Medium
- Harbour size: Medium
- Railway size: Large
- Harbour type: Coastal Breakwater
- Max. size: Up to 152 m (500 ft) in length
- Repairs: Moderate

- Shelter: Good
- Latitude: 44°10’0.00” N
- Longitude: 28°38’60.00” E

The Port of Constanta (see Figure 18.18) is a container hub and is the most important container terminal in the Black Sea with a throughput capacity of 1,5 million TEU (twenty-foot equivalent unit) per year.

Port service time is 56 h per week. Considering the average number of non-operational days due to adverse weather conditions such as rain, fog, and heavy storms, the number of weather working days varies between 330 and 350 per year.

The port area is utilised through a total number of 21 terminals for commercial cargo handling operations.

The port is located at the crossroads of the trade routes linking the markets of the landlocked European countries to Transcaucasus, Central Asia and the Far East. The port has connections with Central and Eastern European countries through Corridor IV (rail and road), Corridor VII – Danube (inland waterway), to which it is linked by the Danube–Black Sea Canal, and Corridor IX (road), which passes through Bucharest.

The port is both a maritime and a river port. Daily, more than 200 river vessels are in the port for cargo loading or unloading or waiting to be operated. The connection of the port with the Danube river is made through the Danube–Black Sea Canal, which represents one of the main strengths of the port. Important cargo quantities are carried by river, between Constanta and Central and Eastern European countries: Bulgaria, Serbia, Hungary, Austria, Slovakia, and Germany.

With regard to rail infrastructure, the port is connected to the Romanian and European rail network, with the port being a starting and terminus point for Corridor IV, a Pan-European corridor. Corridor IV follows the following route: Dresden/Nuremberg → Prague → Vienna → Bratislava → Győr → Budapest → Arad → Bucharest → Constanta/Craiova → Sofia → Pernik → Thessaloniki or Plovdiv → Istanbul.

With regard to road infrastructure, the port is located close to Corridor IX, passing through Bucharest. Corridor IX follows the following route: Helsinki → Vyborg → Saint Petersburg → Moscow → Kiev → Chişinău → Bucharest → Ruse → Dimitrovgrad → Alexandroupolis.

The total length of the roads in the port amounts to 100 km. Highway A2 connects the port with the national road network.

Source: www.constantza-port.ro

Figure 18.18: Sea Port of Constanta, Romania

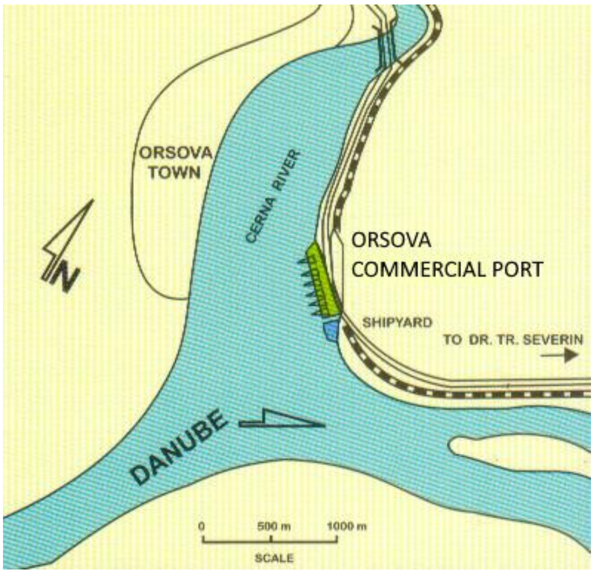
18.5.4.2 Port of Orșova

The Port of Orșova is one of the largest Romanian river ports, located in the city of Orșova on the Danube River.

Port specifications (source: https://shipnext.com/port/orsova-roorv-rou):

- Port type: River port
- Port size: Small
- Water depth: 20 m in the basin
- Max. draft: 4.2 m
- Max. vessel size: 3,000 deadweight tonnage
- Latitude: 44° 43' 21" N
- Longitude: 22° 24' 22" E

The river Port of Orșova (see Figure 18.19) is modern and well equipped, and offers open storage areas, and suitable maintenance and disposal facilities including four gantry cranes up to 16 t single use.



Source: www.apdf.ro

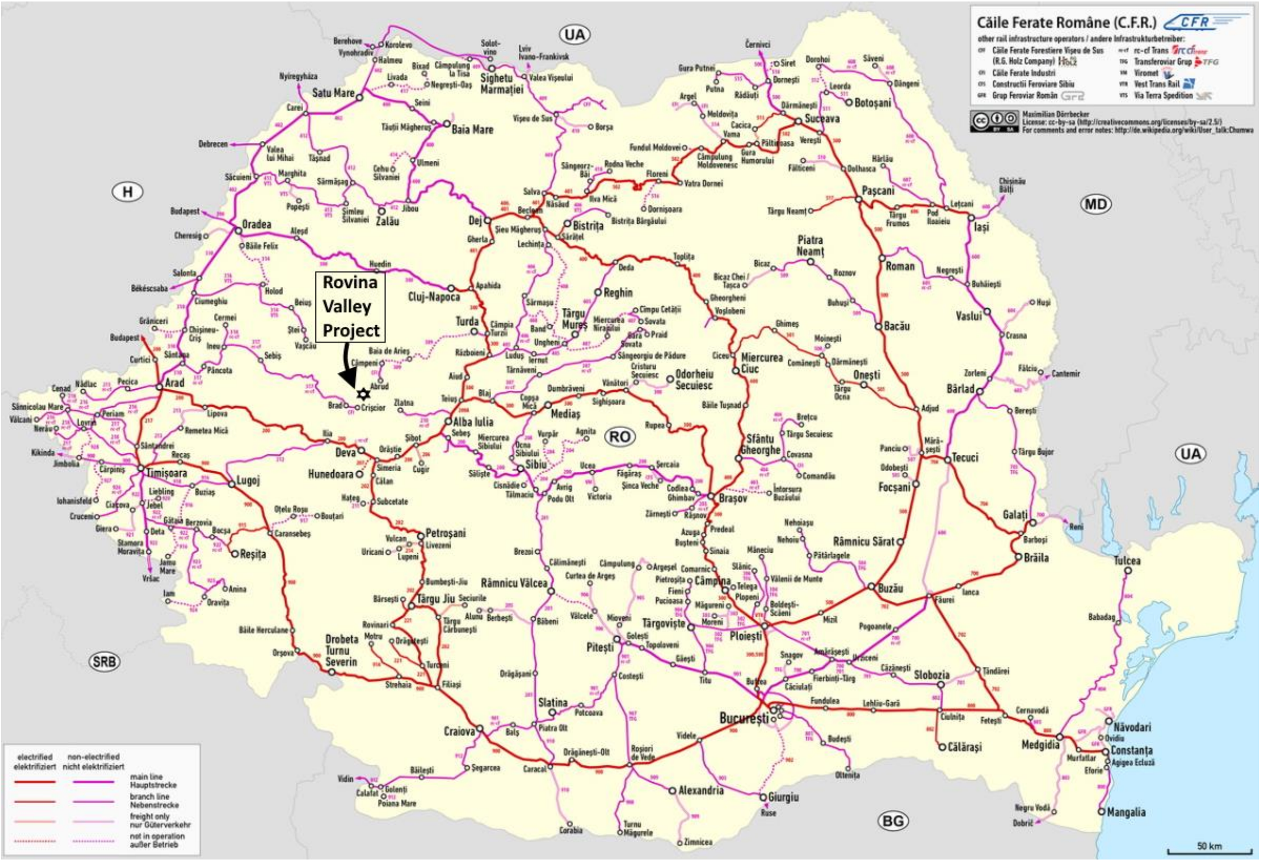
Figure 18.19: River Port of Orșova, Romania

18.5.5 Rail Freight

The rail network in the Port of Constanta is connected to the Romanian and European rail network, with the port being a starting and terminus point for Corridor IV, a Pan-European corridor. Round-the-clock train services carry high volumes of cargo to the most important economic areas of Romania and Eastern Europe, and the port is also an important transport node of the TRACECA, the transport corridor providing the connection between Europe, Caucasus, and Central Asia. The total length of railways in the port amounts to 300 km.

The total Romanian rail network under exploitation amounts to approximately 11,000 km (the 7th largest in the EU), of which approximately 8,000 km (72 %) is single track (the average in the EU is 59 %) and approximately 3,000 km (27 %) is double track. It comprises approximately 1,000 stations, 200 tunnels and 4,000 bridges. Only 37 % of the rail network is electrified compared to the EU average of 54 %.

Figure 18.20 shows the railway network in Romania, and Figure 18.21 shows the nearest railways links to the RVP.



Source: https://commons.wikimedia.org/wiki/File:Railway_map_of_Romania.png

Figure 18.20: Railway Network – Romania

Source: https://cms.rne.eu/rail-freight-corridors

Figure 18.21: RVP – Nearest Railway Links

18.5.6 Documentation

In order to ensure effective management of logistics and that all parties' expectations of the project are met, a written project logistics guide and execution plan will be required. This logistics execution plan will outline the responsibilities of all the stakeholders (contractor/company/suppliers/other interested parties) and will indicate how cargo management and control from time of receipt by the contractor to time of delivery will be achieved. Comprehensive project logistics documents and an execution plan applicable to the RVP were developed during the study.

18.5.7 Project Cargo

Table 18.4 provides an estimated summary of the project cargo, outlining tonnages, number of containers and shipping method.

Table 18.4: Summary of Project Cargo

Item	Description	Freight Mass	Quantity	Shipping Method
		t	Containers	
1	Structural Steelwork (including Conveyors)	3,625	302	Containerised
	Plate Work (including Conveyors and excluding Tanks)	330	28	Containerised
	Sundries, i.e. Liners, Rails, Handrailing, Bolts, and Guards	367	30	Containerised
2	Mechanical:			
	Mechanical Equipment – General (excluding Mills)	595	58	Containerised
	Gyratory Crusher	243	20	Containerised/Break Bulk
	Pebble Crusher	60	5	Containerised/Break Bulk
	Apron Feeder	23	2	Containerised
	Filter Presses	952	80	Containerised/Break Bulk
	Cyclone Cluster	71	9	Containerised
	Pumps	258	22	Containerised
	Conveyor Belting	234	20	Containerised
3	Mills and Components:			
	Ball Mill	900	8	Containerised/Break Bulk
	SAG Mill	1,300	12	Containerised/Break Bulk
	Regrind Mill	90	10	Containerised/Break Bulk
4	Plant/Overland Piping and Valves:			
	Piping	520	65	Containerised
	Valves	28	4	Containerised
	Piping Mills	20	3	Containerised
5	Electrical including MV cables, VSDs and MCCs	675	73	Containerised/Break Bulk
6	Control and Instrumentation	72	6	Containerised

Item	Description	Freight Mass	Quantity	Shipping Method
		t	Containers	
7	Civil/Earthworks	18	2	Containerised
8	Plant First Fills:			
	Ball and SAG Mills First Fill – Grinding Media	348	18	Containerised
	Regrind Mill First Fill	7	1	Containerised
9	Water Treatment Plant	24	2	Containerised
10	Sewage Treatment Plant	24	2	Containerised
11	WWTP	48	4	Containerised
12	Fire Pump Station	16	2	Containerised
13	Tanks, Bins and Steel Sumps	1,113	93	Containerised/Break Bulk
14	Infrastructure:			
	Pre-Engineered Steel Buildings (Steel)	402	34	Containerised
	Pre-Engineered Steel Buildings (Sheeting)	104	10	Containerised
	Prefabricated Buildings and Furniture	312	26	Containerised
	Workshop Tools	70	12	Containerised
	Weighbridge	17	4	Containerised
	Gantry Cranes (Process Plant and MIA)	155	12	Containerised
	Security and Perimeter Fencing	185	51	Containerised
	Diesel Storage Tanks and Dispensing	45	4	Containerised
15	Waste Facility and Raw Water Catchment Dam:			
	HDPE Lining and Geotextiles	82	15	Containerised
16	Spares	114	13	Containerised
TOTAL		**13,447**	**1,062**	

19 MARKET STUDIES AND CONTRACTS

19.1 INTRODUCTION

The RVP, when in operation, is expected to produce an average of 11,038 t/a or 24.3 Mlb of copper in concentrate form. This concentrate will also contain saleable gold. The presence of gold, together with the acceptable levels of impurities, makes the RVP concentrate marketable.

A study was conducted to identify copper smelters which might provide the best commercial terms in the European and Asian markets for the concentrate produced by the RVP

Key findings included the following:

- The market for copper and gold concentrate is strong.
- Smelters have specific preferences regarding the copper:gold ratio in the concentrate and differing levels of tolerance for impurity elements; these concerns will be directly reflected in the terms offered.

The study also concluded that, while there is no generally accepted standard format for the smelter contracts, the main components of the contract usually comprise the following:

- **Assay and handling charges**: usually a charge per lot for assaying, weighing, handling, etc. A maximum lot size or value may apply. These may also include customs charges.
- **Treatment and smelter charges**: usually quoted in terms of United States dollars per troy ounce or euros per kilogram, etc. These may be quoted as a single charge or as separate treatment and smelter components.
- **Penalties**: charges imposed for specific impurities. The contract will specify permissible maximum levels of deleterious elements and may set out a scale of charges for impurities in excess of these limits.
- **Payability, or Return Rate**: specifies the percentage of copper and gold contained in the concentrate for which the miner will be credited.
- **Outturn**: the time period between receipt of copper and gold concentrate by the smelter and payment to the miner.

The transport and refining costs used in the economic analysis of the RVP are based on the indicative quotations obtained in the 2021 Transport Study: “Proiect Transport Minereue Concentrat” prepared for SAMAX Romania SRL by Traffic Plan SRL.

19.2 COPPER CONCENTRATE PRODUCT SPECIFICATION

19.2.1 Production Forecast

The RVP is designed to produce an average of 64,682 t/a of copper and gold concentrate ranging from 62,050 t/a during the Colnic pit period to 72,126 t/a during the Rovina pit period at an average grade of 21.7 % m/m Cu and 38.6 g/t Au.

19.2.2 Product Specification

Table 19.1 shows the RVP concentrate specification from the Colnic and Rovina orebodies. It can be noted that in both instances the concentrate presents acceptable levels of penalty element levels and also saleable content of Cu and Au (i.e. > 1 g/t).

Table 19.1: RVP Concentrate Specification

Element	Colnic Final Concentrate	Rovina Final Concentrate
Cu (%)	21.13	22.70
Fe (%)	33.72	31.80
S (%)	39.40	35.60
Au (g/t)	83.05	11.73
Ag (g/t)	140	62
Al (g/t)	8,300	10,000
As (g/t)	< 50	< 50
Ba (g/t)	37	56
Be (g/t)	0.1	0.2
Bi (g/t)	< 200	< 200
Ca (g/t)	10,000	6,000
Cd (g/t)	120	32
Co (g/t)	62	130
Cr (g/t)	210	56
K (g/t)	1,600	2,900
Li (g/t)	< 10	< 10
Mg (g/t)	1,800	2,600
Mn (g/t)	190	130
Mo (g/t)	2,600	530
Na (g/t)	880	2,700
Ni (g/t)	130	65
P (g/t)	< 200	< 200
Pb (g/t)	1,200	450
Sb (g/t)	< 10	< 10
Se (g/t)	130	65
Sn (g/t)	< 200	< 200
Sr (g/t)	20	32
Ti (g/t)	1,600	1,700
Tl (g/t)	< 30	< 30
U (g/t)	< 40	< 40
V (g/t)	20	26
Y (g/t)	6	4.9
Zn (g/t)	16,000	3,600

19.2.3 Penalties

It is commonly known that copper smelter terms include penalties if the concentrations of impurity elements, such as arsenic, mercury, bismuth and fluorine, exceed stipulated limits. The behaviour of these elements in the smelters can depend on the processes used to treat the concentrate. The principle behind the penalties is to compensate the smelters and refineries for additional costs caused by the presence of the penalty elements, but these costs will vary depending on the process used and the location of the smelter.

Table 1.16 lists typical penalties for deleterious elements in copper concentrates and the magnitude of the penalty when limits are exceeded.

Table 19.2: Typical Penalty Schedule for Copper Concentrate

Element	RVP Expected Concentrate Range (%)	Threshold (%)	Penalty (US$/t per extra 0.1 %)
Arsenic	< 0.005–0.015	0.2	2
Antimony	< 0.001	0.05	15
Bismuth	< 0.022	0.02	25
Cadmium	0.003–0.013	0.03	30
Cobalt	0.006–0.013	0.5	1.0
Fluorine	Not measured	0.03	15
Lead	0.049–0.13	1.00	0.3
Mercury	Not measured	0.0005	3,000
Nickel	0.007–0.014	0.5	1.0
Selenium	0.007–0.024	0.03	15
Zinc	0.39–1.763	3.00	0.3

19.2.4 Moisture

The concentrate moisture is required to meet the threshold moisture limit of < 10.65 %. The RVP concentrate design moisture content is < 8.50 % and will, therefore, be acceptable for bulk transportation and for smelter requirements.

19.2.5 Credits

Typical elements attracting a credit are gold, silver, and other precious metals such as platinum group metals (PGMs) if they are present in significant quantities. Payments are normally made if the amount contained in the concentrate exceeds a certain level. The price of the raw materials shall be the sum of the values of the payable metals less the sum of the deductions. This must be stipulated in any long-term agreements.

In the case of the RVP, the copper concentrate contains gold. For gold, typically the buyer shall pay for a minimum of 97.50 % of the final gold content, subject to a minimum deduction of 1.0 g/dmt.

19.2.6 By-Product Credits

It is not common for smelters to pay for by-products in copper concentrates; however, for those that do, the main by-product for copper concentrates is sulphuric acid.

The rule of thumb is that the value of sulphur = 1 dmt of copper concentrate with 30 % of sulphur and gives approximately 1 t of sulphuric acid (H_2SO_4) depending on the recovery rate. The typical recovery rate is 95 % in a copper smelter.

19.3 COPPER MARKET FUNDAMENTALS

19.3.1 Copper Concentrate Demand

The global copper concentrate market was valued at US$77,034.8 million in 2021 and is projected to reach US$96,435.3 million by 2028, expanding at a compound annual growth rate (CAGR) of 3.1 % during the forecast period. The market is driven by factors such as growing awareness regarding the properties of copper and its use in different industries, rising demand for copper, increasing production of copper-based alloys, and the growing use of copper-based products for human health.

More than 80 % of the mined copper is processed in smelters. There has been a marked increase of the copper smelting industry over the past decade, which is largely due to the growth in the Asia Pacific region, influenced mainly by growth in China. Asia Pacific is a promising region for the market. It constituted a 52.3 % share of the market in 2020. The market in the region is projected to expand at a CAGR of 3.3 % during the forecast period. The demand for copper concentrate is expected to rise due to the rapid growth in the construction, automotive, and appliance industries in the region. The market in North America is projected to expand at a CAGR of 3.2 % due to the recent advances in the mechanical industry and the expected launches of new trains and aircraft during the forecast period.

The copper smelting industry consists of the following:

- Smelters that process concentrates from their own upstream facilities
- Smelters that process bought concentrates produced by other facilities where they have no ownership: custom smelting
- Smelters that process a combination of the aforementioned: integrated smelting.

Increasing urbanisation and industrialisation are big catalysts for the continued and high demand for copper. Emerging economies boost the demand for copper. Growing wealth is expected to increase the use of copper in homes and workplaces; therefore, the copper concentrate market is projected to grow due to urbanisation. Electric vehicles (EV) represent nearly 30 % of this increased demand. As per the Deloitte Study (Deloitte Insights, 2020, “Electric vehicles, setting a course for 2030”), the EV forecast is for a compound annual growth rate of 29 % achieved over the next ten years: total EV sales growing from 2.5 million in 2020 to 11.2 million in 2025, then reaching 31.1 million by 2030. EVs would secure approximately 32 % of the total market share for new car sales.

Currently, 55 % of the world's population lives in cities, up from just under 47 % in 2000. According to the United Nations, this global trend is expected to increase to 64.5 % by 2040 driven by India, China, and Nigeria. In terms of consumption, China accounted for a 48 % share of the global refined copper market in 2018, while Europe and North America consumed a 16 % and 10 % share, respectively. From 15 % to 25 % of the total consumption of China's processed goods is re-exported. No other country is expected to have a longer-term growth than China in the Asia Pacific region over the next 20 years. In 2019, China became the world's leading producer of refined copper while consuming approximately 51 % of all copper.

19.3.2 Copper Concentrate Supply

China was a leading consumer of refined copper in 2018. Moreover, it was a key manufacturer of refined copper. The country was responsible for a 36 % share of the refined copper from the total global production in 2018. After China, Chile is the next largest producer of refined copper, responsible for a 10.4 % share of refined copper production worldwide.

In 2019, approximately 20 Mt of copper was produced worldwide. The total demand growth from 2017 to 2040 is forecast at 1.8 % per year and is projected to hit 43.8 Mt due to transportation, buildings, and manufacturing electrification. Additionally, governments continue to set renewable energy targets, resulting in a higher demand for copper. Globally, building and electronics were the main end-users of copper, accounting for over 60 % of the global copper production. However, there are geographical differences in the end usage of copper. In India, for example, the construction industry accounts for approximately 49 % of copper consumption, while in China, the vital use of copper is in the electronics and communications industry, which makes up approximately 42 % of the total copper consumption.

19.3.3 Market Balance

The balance between the supply and demand of concentrates is the key to how much the smelters will charge the sellers. When concentrate supply is lower than the smelters' demand, then smelters need to compete for concentrates, and the market favours the miners. Conversely, if there is an oversupply of concentrate, stocks begin to accumulate, and smelters offer higher terms, which mines accept to offload excess supply. The global import and export trade of the copper concentrate is rising largely to falling copper content.

19.3.3.1 Short-Term Outlook

There's moderate demand due to the impact of Covid-19, and low-to-moderate supply due to supply chain restrictions globally.

Currently, investments have stagnated due to the global pandemic, and each region is facing difficulties in managing several industries while maintaining safety measures for their employees.

Existing investors in the mining and copper industries are investing in short-term opportunities and profits. New investors are investing in the trending sectors, such as healthcare and IT solutions.

The copper mining companies in China need to remain committed to investing in overseas copper deposits to ensure access to high-quality, low-cost resources. For instance, in October 2019, Zijin Mining announced the investment of US$146 million to boost its participation in Ivanhoe Mining. This, in turn, increased the production of copper in the short-term outlook.

19.3.3.2 Long-Term Outlook

In the long term, the number of importers is projected to increase in Asia Pacific. China is expected to continue to hold a major market share in the global total imports of the copper concentrate in the coming years.

India is an emerging country in terms of copper production and is expected to be a leading importer of copper concentrate during the forecast period.

In the long term, many of the new and existing investors are estimated to invest in smart and green mining activities.

19.3.4 Major Customers of Concentrates

Asia Pacific is the largest importer of copper concentrate at > 50 %, with China taking the lead in this regard. Table 19.3 shows the top 10 importing countries globally. Key importers of copper concentrate are China, Japan, Spain, Germany, South Korea, and India. The import of copper concentrate by these countries is expected to continue as many manufacturers have tie-ups with these country-based importers.

In the long term, the number of importers is projected to increase in Asia Pacific. China is expected to continue to hold a major market share in the global total imports of the copper concentrate in the coming years. India is an emerging country in terms of copper production and is expected to be a leading importer of copper concentrate during the forecast period.

Table 19.3: Top 10 Concentrate Importing Countries – Trade Balance

No.	Country	Volume of Concentrate (kt)					
		2015	2016	2017	2018	2019	2020
1	China	13,307	17,050	17,372	19,738	22,024	**
2	Japan	4,828	5,131	4,732,	5,250	4,789	5,228
3	Spain	1,783	1,796	1,958	2,130	1,493	1,203
4	Korea, Republic of	1,769	1,558	1,650	1,820	1,731	2,004
5	Germany	1,165	1,058	1,250	1,186	1,020	1,239
6	India	1,697	1,023	1,468	1,050	772	**
7	Bulgaria	947	754	1,097	1,106	731	**
8	Taipei, Chinese		584	328	217	415	**
9	Brazil	596	555	562	377	450	301
10	Finland	447	436	441	440	458	
11	Russian Federation	187	549	689	629	482	308
** Information not available at the time of publishing this report							

19.3.5 Long-Term Price Forecast

The long-term price forecast for copper and gold is shown in Table 19.4. The forecast shows a steady decline of the copper price over the period, from a high of US$8,529/t in 2021 to a low of US$6,351/t in 2030. The gold price is also forecasted to show a steady decline for the period 2021 to 2030, from a high of US$1,824/oz in 2021 to a low of US$1,421/oz in 2030.

Table 19.4: Price Forecast for Cu and Au

Metal	Description	Jun-21	Sep-21	Dec-21	Mar-22	Jun-22	Sep-22	Dec-22	Mar-23	2023	2024	2025	Long Term (Nominal)	Long Term (real)
													2026-2030	2026-2030
Copper (US$/t)	**Consensus (Mean)**	**8,529**	**8,308**	**8,070**	**7,885**	**7,848**	**7,692**	**7,576**	**7,585**	**7,417**	**7,188**	**7,393**	**7,650**	**6,351**
	High	10,000	9,750	10,000	9,750	9,750	9,435	9,381	9,427	9,500	9,591	9,831	10,476	7,599
	Low	7,606	7,250	6,600	6,393	6,100	5,750	5,732	5,800	5,567	5,800	5,535	6,763	5,535
	Standard Deviation	587	665	821	930	1,055	1,038	1,043	1,021	1,012	950	948	1,006.8	592.18
Gold (US$/oz)	**Consensus (Mean)**	**1,824.6**	**1,821.3**	**1,805.1**	**1,793.5**	**1,766.2**	**1,740.4**	**1,720.7**	**1,682.5**	**1,688.9**	**1,633.3**	**1,634.7**	**1,659.7**	**1,421.1**
	High	2,175.0	2,200.0	2,200.0	2,300.0	2,300.0	2,300.0	2,300.0	2,137.1	2,169.5	2,216.8	2,236.4	2,310.2	1,865.4
	Low	1,680.0	1,645.0	1,610.0	1,450.0	1,450.0	1,450.0	1,450.0	1,475.0	1,452.5	1,355.0	1,344.8	1,326.5	982.9
	Standard Deviation	111.7	134.2	146.0	172.7	177.9	194.2	216.0	180.3	179.7	211.5	240.0	309.32	251.30
	Number of Forecasts	25	25	25	25	25	24	23	16	23	21	18	12	11
Source: Energy and Metals Consensus Forecast, 15 March 2021														

19.4 COPPER MARKETING STRATEGIES

19.4.1 Potential Customers

The market research 'Upmarket Research, March 2021, "Global Copper Concentrate Market Analysis & Forecast, 2018-2028" shows that the copper concentrate market will have sufficient demand for this raw material in the long term. There is a potential for long-term contracts with Asian customers, as the research shows capacity availability in this region. China is expected to continue to be a major player in Asia Pacific and globally, and the potential major customers in this region include the following:

- Jiangxi Copper Corporation
- Jinchuan Non-Ferrous Co.
- Tongling Non-Ferrous/Sharpline Intl./Sumitomo/Itochu

Europe is also a potential customer, with Spain, Germany and Sweden being potential viable options for this project. Figure 19.1 shows the top smelters' capacity and production in 2020. Based on the production during this period, Asia Pacific as a region has more available capacity for further concentrate imports than any other region. Table 19.5 shows the top 40 smelters and their capacity and production for 2018 and 2020.

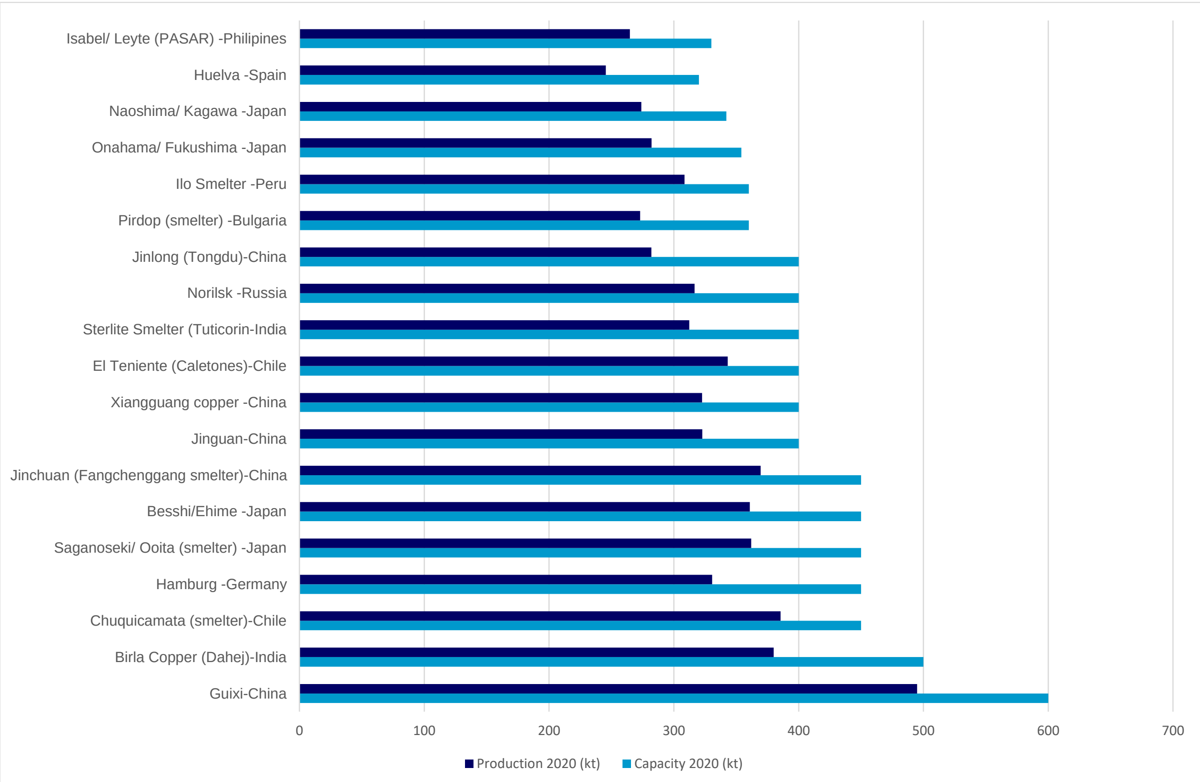


Figure 19.1: Top Smelters Capacity and Production

Table 19.5: Top 40 Smelters – Capacity and Production

No.	Smelter	Location	Capacity (kt)	Production (kt)	2019	2020
			2019	2018		
1	Guixi	China	960*	757.8	814.1	495
2	Birla Copper (Dahej)	India	500**	389	396	380
3	Nippon Mining & Metals Co. Ltd	Japan	470	392	392.1	
4	Chuquicamata (smelter)	Chile	450	395.6	395.6	385.5
5	Hamburg	Germany	450	353.3	353.7	330.7
6	Saganoseki/ Ooita (smelter)	Japan	450	375.3	375.4	361.9
7	Besshi/Ehime	Japan	450	375.8	375.8	360.9
8	Jinchuan (Fangchenggang smelter)	China	450	335.2	379.8	369.5
9	Jinguan	China	400	336.8	336	322.8
10	Xiangguang copper	China	400	336.8	336.8	322.6
11	El Teniente (Caletones)	Chile	400	351.2	351.6	343.1
12	Sterlite Smelter (Tuticorin)	India	400	328	328.4	312.4
13	Norilsk	Russia	400	320.4	333.8	316.6
14	Jinlong (Tongdu)	China	400	296.1	340	281.9
15	Pirdop (smelter)	Bulgaria	360	284.4	284.4	272.9
16	Ilo Smelter	Peru	360	314.6	315.7	308.5
17	Onahama/ Fukushima	Japan	354	295.6	295.6	282.1
18	Naoshima/ Kagawa	Japan	342	286.6	286.3	273.9
19	Huelva	Spain	320	253.8	255.4	245.4
20	Isabel/ Leyte (PASAR)	Philippines	330	271.2	272.3	264.7
21	Mount Isa Mines Ltd	Australia	250	210.5	212	***
22	Caraiba Metais S.A.	Brazil	220	184.4	185.7	***
23	Eliseina Ltd	Bulgaria	14	11.8	11.8	***
24	Falconbridge Ltd	Canada	155	131.1	131.8	***
25	Atlantic Copper S.A.	Spain	320	269.4	269.4	***
26	Hibi Kyodo Smelting Co. Ltd	Japan	263	208.6	209.9	***
27	Hudson Bay Mining & Smelting	Canada	90	72.1	75.1	***
28	Inco Ltd	Canada	135	112.7	112.7	***
29	Kennecott Utah Copper Co.	USA	320	267.2	267.2	***
30	Metallo-Chemique N V	Belgium	150	125.3	125.6	***
31	MMC Norilsk	Russia	100	77.8	79.2	***
32	Montanwerke Brixlegg	Austria	85	74.7	74.7	***
33	Mopani Copper Mines Plc.	Zambia	180	141.3	141.5	***
34	Nippon Mining & Metals Co. Ltd	Japan	470	392	392.1	***
35	Noranda Inc.	Canada	200	175.6	175.8	***
36	Palabora Mining Co. Ltd	South Africa	135	110.7	110.8	***
37	Southern Peru Copper Corp.	Peru	300	237	237	***

No.	Smelter	Location	Capacity (kt)	Production (kt)	2019	2020
38	Umicore	Belgium	190	166.1	166.6	***
39	Zambia Consolidated Copper Mines Ltd	Zambia	200	168.4	169.6	***
40	KGHM Polska Miedz, S.A.	Poland	220	181.1	181.5	***
*Capacity reduced to 600kt/a in 2020 ** Capacity reduced to 350kt/a in 2020 *** Information not available at the time of publishing this report						

19.4.2 Potential Markets of Growth

Figure 19.2 represents the global copper concentrate market size for the period 2017 to 2028, where 2017–2020 are historical years, 2020 is the base/actual year, and 2021–2028 are forecast years. The market is expected to grow at a year-on-year rate of 0.98 % between 2020 and 2021 owing to the recovery from the effect of the 2020 Covid-19 pandemic on the copper concentrate market. The return of copper production facilities to their previous capacity and the restart of normal transportation activities will result in a strong recovery in the market.

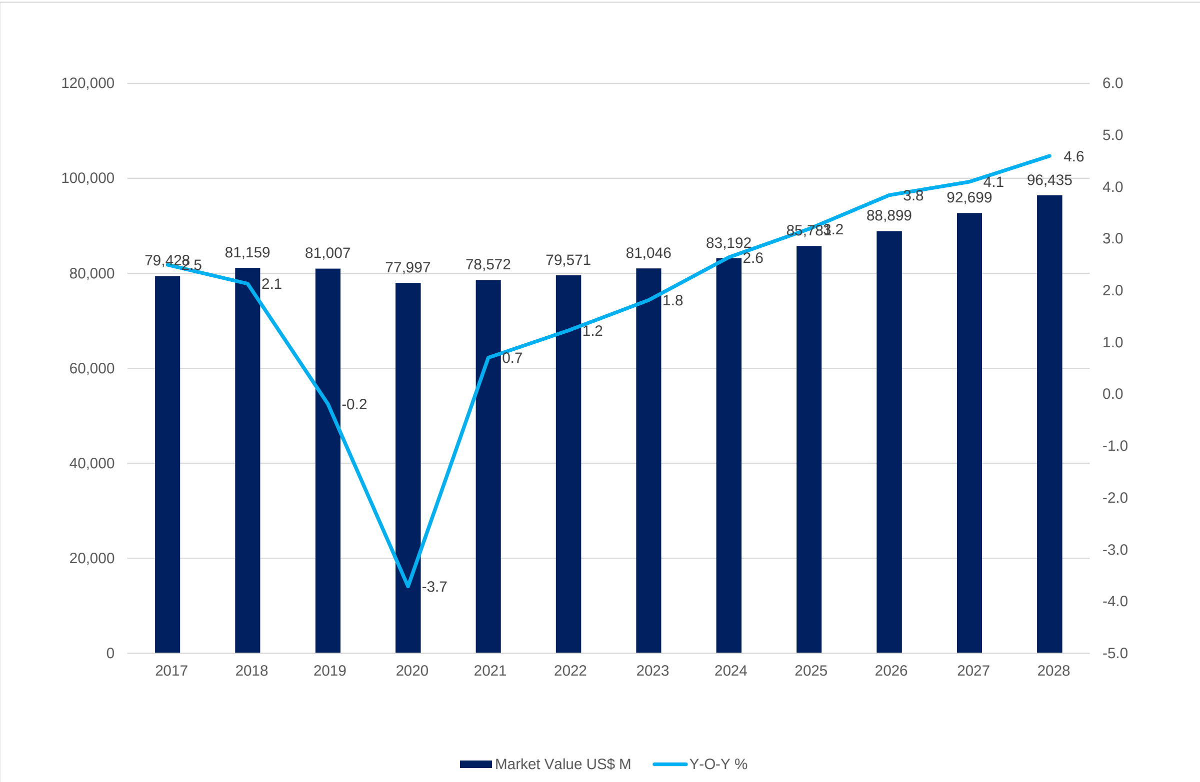


Figure 19.2: Market Volume Forecast

19.5 COPPER CONTRACT AND PRICING STRATEGIES

19.5.1 Contract Parameters

The RVP concentrate does not contain unacceptable amounts of any deleterious elements, based on this and the valuable metals (e.g. gold) contained within the concentrate, it would be considered marketable in the main international markets with no penalties for impurities. Long-term contracts often contain favourable commercial components such as options on the quotation periods and flexibility on the freight/destination. It is assumed that 100 % of the sales under long-term contracts would be to customers in Asia, mainly China, and Europe. As per industry standards, in the case of sales to destinations such as India and China, sampling and weighing operations would be carried out at the load port, and it is assumed that a weight allowance of 0.2 % would be applicable. Based on the handling requirements from the mine to port and port to vessel loading, an in-transit loss of material of 0.3 % to 0.5 % should be considered. Loss mitigation procedures will be implemented to minimise these losses.

19.5.2 Commercial Variables

The pricing for copper concentrates is established through a process of negotiation between buyer and seller through a copper concentrate smelting contract covering a certain time frame, typically renegotiated annually. The contract provides a framework covering a range of terms and conditions including treatment charges (TCs) and refining charges (RCs), payables for copper and gold, penalties for impurities, quotation periods for payable metals and payment terms. A number of standard commercial contract clauses covering the logistics of delivery, taxes and duties, insurance, assaying and force majeure conditions are also normally included. According to the industry standard, the key commercial variables to be considered for profitability calculation include metallurgical deductions, TC/RCs, penalties, quotation periods, price participation, freight parity, payment and concentrate trading during a backwardation.

19.5.3 Treatment Charges and Refining Charges

TCs and RCs are negotiated annually between miners and smelters. Typically, negotiations start in October and normally finish by year-end. Benchmark terms are normally established when the major mines and smelters have agreed the annual terms. The TC and RC for the top 40 smelters for 2020 are shown in Table 19.6.

Table 19.6: Top 40 Smelters TC and RC

No.	Smelter	TC (US$/t)	RC (US$/t)
1	Guixi	71.70	72.20
2	Jinchuan (Fangchenggang smelter)	54.10	55.80
3	Jinguan	70.40	74.60
4	Jinlong (Tongdu)	61.10	65.10
5	Xiangguang copper	71.70	74.80
6	Huelva	75.50	78.90
7	Norilsk	74.90	78.00

No.	Smelter	TC (US$/t)	RC (US$/t)
8	Besshi/Ehime	80.60	84.20
9	Onahama/ Fukushima	73.50	76.90
10	Naoshima/ Kagawa	76.50	80.10
11	Birla Copper (Dahej)	53.40	56.00
12	Chuquicamata (smelter)	55.00	57.70
13	Hamburg	72.00	75.70
14	Saganoseki/ Ooita (smelter)	73.60	77.30
15	El Teniente (Caletones)	58.30	61.20
16	Sterlite Smelter (Tuticorin)	53.60	56.30
17	Pirdop (smelter)	74.30	78.30
18	Ilo Smelter	57.10	60.10
19	Jinlong (Tongdu)	66.30	70.30
20	Isabel/ Leyte (PASAR)	66.40	70.30
21	Mount Isa Mines Ltd	68.90	73.10
22	Caraiba Metais S.A.	55.50	58.80
23	Eliseina Ltd	73.90	78.40
24	Falconbridge Ltd	75.70	80.30
25	Atlantic Copper S.A.	78.70	83.90
26	Hibi Kyodo Smelting Co. Ltd	73.10	77.90
27	Hudson Bay Mining & Smelting	74.50	79.40
28	Inco Ltd	75.20	80.20
29	Kennecott Utah Copper Co.	78.00	83.40
30	Metallo-Chemique N V	80.40	86.10
31	MMC Norilsk	80.10	85.90
32	Montanwerke Brixlegg	79.60	85.30
33	Mopani Copper Mines Plc.	65.90	70.80
34	Nippon Mining & Metals Co. Ltd	73.90	79.60
35	Noranda Inc.	74.60	80.30
36	Palabora Mining Co. Ltd	63.80	68.80
37	Southern Peru Copper Corp.	66.00	71.40
38	Umicore	76.90	83.20
39	Zambia Consolidated Copper Mines Ltd	66.10	71.60
40	KGHM Polska Miedz, S.A.	73.50	79.90

Based on the TCs, RCs and capacity availability, the following smelters may be considered as viable buyers for the RVP concentrate and should be investigated further for potential contracts:

- Jinchuan (Fangchenggang smelter) – China
- Birla Copper (Dahej) – India
- Sterlite Smelter (Tuticorin) – India

The following smelters that are within the RVP region and have capacity for custom concentrate may be investigated:

- Bor Smelter, Serbia
- Pirdop Smelter, Bulgaria
- Hamburg Smelter, Germany
- Huelva Smelter, Spain

19.5.4 Typical Pricing Clause

The price of the raw materials is the sum of the values of the payable metals less the sum of the deductions. The buyer should pay for a percentage of the absolute copper content, dependent on a minimum deduction of 1.0 unit at the official London Metal Exchange (LME) Grade A Settlement Copper Quotation, i.e., an average of over the quotation period. The percentage paid depends on the copper concentrate grade. Figure 19.3 shows the price points at various copper concentrate grades.

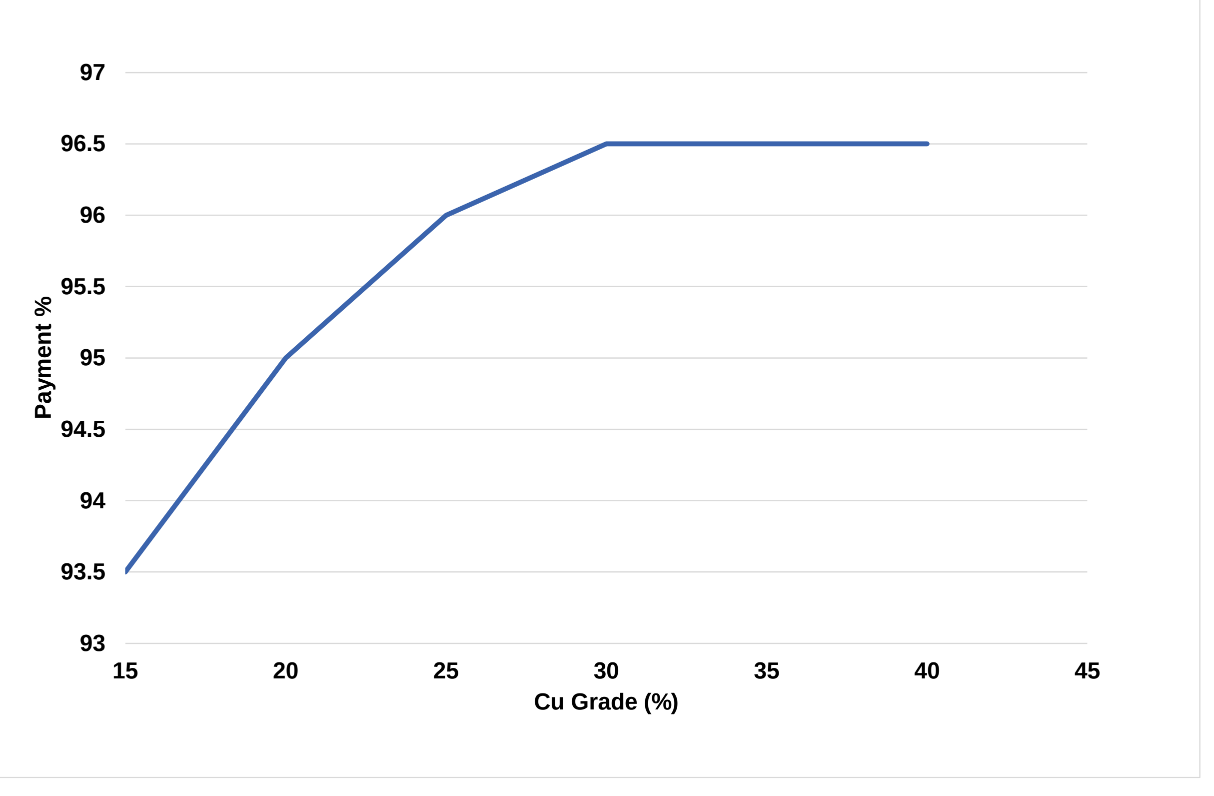


Figure 19.3: Copper Concentrate Grade vs Payment Percentage

In the case of the RVP, with a copper concentrate grade of ~22 %, the payable copper value is as follows:

22 % × 95.2 % × US$6,492/t Cu = US$1,174/dmt

19.5.5 Price Participation

In addition to the TC/RC, term contracts contain a price participation (PP), whereby the buyer receives a proportion of the copper price above a certain level. PPs, as with TCs and RCs, are negotiated annually between miners and smelters, typically between October and year-end.

The extent of the PP is normally 10 % of the difference between the actual price and the neutral price. In some contracts, the PP could work both ways, so the buyer concedes a proportion to the seller below the neutrality point. Given the projected supply shortfall and forecast growth in Chinese smelting capacity, it is unlikely that the smelters will be in a position to force the reinstatement of the PP clause.

19.5.6 Quotation Period

The quotation period for copper is usually the month following the delivery month of the copper concentrate. The average price for any such quotation period shall be calculated by totalling the US dollar equivalents of the daily prices and dividing such a total by the number of pricing days in such period. The quotation period shall be agreed upon between the buyer and seller of the concentrate.

19.5.7 Freight Parity

Freight parity applies when the buyer decides to take delivery at a port other than a main port, and the seller is left in a neutral position compared with delivery to a main port. If the freight cost to the other port is higher than to a main port, the buyer pays the difference to the seller; if it is lower, the buyer receives the difference from the seller.

19.5.8 Payment

The period of payment varies from contract to contract. For a spot contract, the sale is typically 100 % bill of lading plus 30 days; for a long-term contract, payment is 90 % on arrival at the discharge port plus 15 days.

19.6 COPPER MARKETING RESOURCES AND ORGANISATION

International trade in copper concentrate typically requires a marketing team consisting of the following:

- Commercial Manager to supervise the whole marketing process as well as to be in charge of looking for new potential markets/customers
- Negotiator to be in charge of trading the product and maintaining the commercial relationship with customers, local customers' representatives, and traders
- Commercial Controller to control and ensure that all processes of the back office are carried out free of errors
- Contract Administrator to be in charge of invoicing and all administrative issues regarding product sales
- Logistics Coordinator to be in charge of coordinating logistics/shipping requirements for product sales

19.7 COPPER PRODUCT LOGISTICS

The Contract of Affreightment is established on a long-term basis (three to five years) to ensure stable, best freight rates. Distribution does not apply to this case. The copper concentrate will be produced at the process plant at a rate of 120 t/d average, requiring a small fleet of trucks to handle the load. The copper concentrate can be trucked 395 km from the process plant to Bor, Serbia where the possible selected smelter is located. The copper concentrate cake will be filtered and stockpiled in a copper concentrate storage building (total capacity of 2,000 t). Concentrate truck loading will take place inside the concentrator-filter building using FELs. The concentrate will be loaded into 1,000 kg bags, which will be loaded into 40' containers and transported by trucks to reach the destination smelters.

The transport of the concentrate from the process plant to the smelter will be carried out by a fleet of five trucks with a maximum capacity of 25 t.

For more details, refer to the report “Proiect Transport Minereue Concentrat” prepared for SAMAX Romania SRL by Traffic Plan SRL.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 INTRODUCTION

The Rovina Valley Project (RVP) is composed of three gold-copper porphyry deposits located in close proximity to each other in the Rovina Valley and Garzi Valley of Hunedoara County in the South Apuseni Mountains of western Romania. The deposits are Rovina to the north, Colnic centrally, and the Ciresata deposit to the south. This Definitive Feasibility Study (DFS) incorporates only the Rovina and the Colnic deposits. While the Ciresata deposit can be brought into the project for development later, it is not discussed further in this section.

The RVP has undertaken permitting efforts, environmental studies, and community engagement since 2004. Initial environmental and social baseline work commenced in 2012 and continues to the present day. A preliminary Environmental Impact Assessment (EIA) was produced by SC ECO TERRA ING SRL for SAMAX in 2014, but changes in project design and the need for more complete baseline information and assessment has continued the work programme.

This section summarises the work undertaken for social and environmental studies, permitting, and community engagement, and highlights key risks and mitigations for these different aspects that will be more completely covered in the Project Environmental and Social Impact Assessment (ESIA) to be undertaken in 2021.

20.2 PROJECT DESCRIPTION

The property is centred at latitude 46° 07' N and longitude 22° 54' E or 515,000 N and 340,000 E using the Stereo70 projection of the Romanian National Geodetic System. Elevations on the property range from 300 masl to 940 masl. The project footprint, with a rectangular boundary, extends over an area approximately 8 km north–south and 7 km east–west (see Figure 20.1).

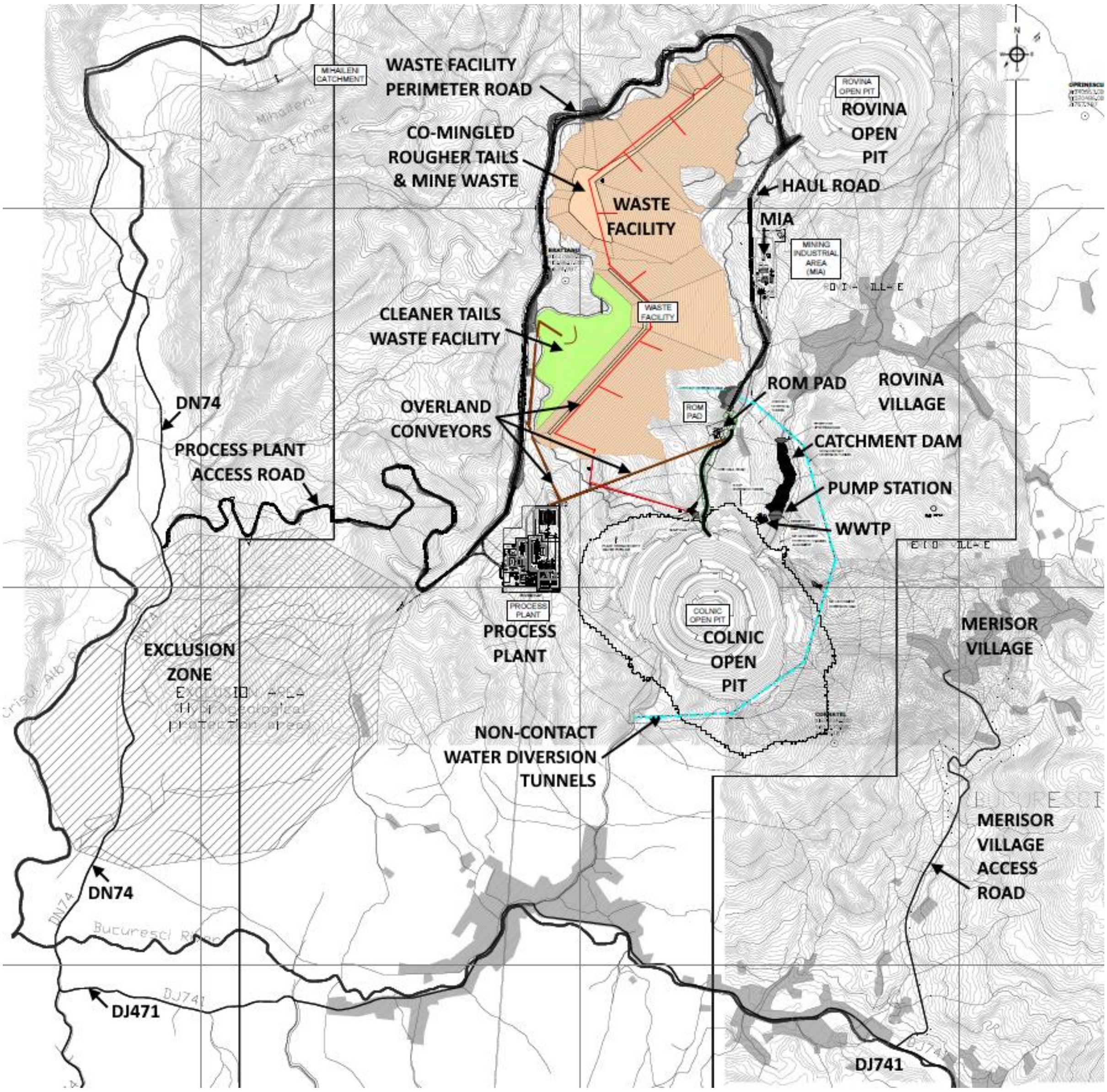


Figure 20.1: RVP Area of Influence and Infrastructure

The deposits to be exploited in the RVP are Rovina to the north and Colnic to the south. The deposits will be mined with a standard open-pit mining method using trucks and a hydraulic loader. The open-pit mining operation will last 16.7 years during which ore will be crushed at the primary crusher and then conveyed to the process plant for treatment.

The ore processing method will use industry-standard crushing-grinding and flotation technology to recover a copper-gold concentrate, without the use of cyanide, at a processing plant located to the west of the Colnic open pit. The processing plant area will also include an office, laboratory, transformers, and separate water treatment facilities for both raw water and potentially contaminated contact water. Process water needs will be met by recycling used process water, using the water from the pit dewatering sumps, and making up the balance with raw water. Flotation tailings will be filtered and co-disposed with waste rock within the waste facilities situated between the Rovina and Colnic pits. The mining industrial area (MIA) will be constructed in the Rovina Valley, east of the waste facilities, and will comprise a workshop, truck wash, tyre store, fuel storage and dispensing area, warehouses, offices, and parking. Haul roads will serve the two open pits, the primary crusher, and the waste facilities.

The processing plant area will be serviced by a dedicated underground power line, a raw water pipeline from the contact water pond, and access roads. As well as providing some of the required process water, the raw water will be treated and made potable for use at the plant and maintenance areas, and also serve a domestic mains water supply network specially constructed for the village of Rovina, which is not currently connected to mains water. See Section 18.4 for further details on water management.

The waste rock and filtered rougher scavenger tailings will be co-mingled and deposited in the waste facility by conveyor placement in raises. As each section is completed and contoured, a compacted clay layer, up to 1.5 m thick, will cap the section and be covered with a further layer of stockpiled topsoil before being planted to forest. The lower volume cleaner tails waste stream will have a sulphide-rich content that makes it prone to ARD and ML. This will be deposited in a special lined cell within the waste facility west of the main facility to isolate it from any percolating water and ensure geochemical stability in the long term. See Section 18.3 for further details on the waste management.

The waste facilities will have surface run-off drains and underdrains that will collect contact water and redirect it through concrete-lined channels to a contact water pond. The Colnic pit straddles the Rovina Valley Stream, so the 2.2 km long non-contact diversion tunnel will be constructed to allow the stream to bypass the pit outline and rejoin the original course to the south of the pit. Both the non-contact diversion tunnel and the contact water pond have been designed to tolerate a 24 h 1:1,000-year flood inflow. Ephemeral and permanent watercourses in the upper western Rovina Valley (Catchment H) will be overlain by the waste facilities and so diversion channels will be created around the perimeter of the waste facilities and pits to rejoin the Rovina River via the non-contact diversion tunnel. A wastewater treatment plant will be located by the contact water pond. Wastewater will only be discharged from the treatment plant once it has achieved the regulated chemical quality and will be piped to the non-contact diversion tunnel, which then flows on to the Crişul Alb River at Crişcior. See Section 18.4 for further details on Water Management.

Topsoil will be stockpiled during land clearance of the pits and waste facilities to be used in closure as restoration material. The Colnic pit will be progressively backfilled in Years 10 to 16 of operations, and during closure the landscape will be restored, including the course of the Rovina Valley Stream. The Rovina pit will be stabilised during closure and allowed to flood to form a lake with the surrounding restored landscape.

Non-mineral waste will be segregated for recycling and sent to licensed waste management third parties, and non-recyclable waste will be sent to licensed waste disposal facilities. A Waste Management Plan will cover the monitoring and management of these activities.

There are two villages within the project footprint: Rovina and Bucureşci. An existing county road connects Bucureşci, Rovina and Merișor. Eleven households that will be physically displaced have been identified in the construction footprint. They are all located in the footprint of the waste facility, in the Remetea neighbourhood of the Rovina village.

20.3 PERMITTING

In April 2004, SAMAX was granted a non-exclusive Prospecting Permit, which includes the present area of the RVP. In August 2005, following a public tender and bid process, SAMAX was granted the Rovina Exploration Licence (Rovina no. 6386/2005) for a period of four years by the National Agency for Mineral Resources (NAMR). In August 2009, SAMAX was granted a three-year extension to the Rovina Exploration Licence.

On 27 May 2015, SAMAX obtained Mining Licence no. 18174 for the RVP (100 % owned by ESM), approved by the Romanian Government by Government Decision no. 900/09.11.2018, published in Official Gazette no. 970/16.11.2018, Part I of which set the following conditions:

- Name of the perimeter: ROVINA
- The surface of the exploitation perimeter is 27,678 km^2, in which the mining objectives will occupy 4.73 km^2.
- The duration of the negotiated concession is 20 contractual years, with the right of extension for successive periods of 5 years each.
- The mining product made by the holder is copper concentrate with gold content.

The RVP is subject to European Union (EU) environmental laws, such as the environmental impact assessment directive (2011/92/EU), water and waste framework directive (2008/98/EC), industrial emissions directive (2010/75/EU), management of waste from extractive industries directive (2006/21/EC), and the related Best Available Techniques (BAT) Reference Document for the Management of Waste from Extractive Industries, in accordance with Directive 2006/21/EC (EUR 28963 EN 2018). The RVP has also been aligned with national legislation and good international practice such as the performance requirements of the Equator Principles, EP4 (2020), the International Finance Corporation (IFC) Performance Standards on Environmental and Social Sustainability (2012), and the World Bank Group Environmental, Health, and Safety General Guidelines (2007).

The Environmental Agreement for construction (*Acord de mediu*), exploitation and closure of the mines is a key milestone, setting out the area and terms for exploitation. This requires various precursor permits and permissions to be obtained, including the following:

- Approval to change the land use for mining by approving a Plan Urbanistic Zonal (PUZ)
- A Decision on the EIA Screening and Scoping
- Potential additional regulator conditions for water, cultural heritage and nature protection
- Approval from the National Institute of Public Health

Key permits and an indicative target schedule for the permitting process are summarised in Table 20.1.

The following risks associated with potential project delays have been identified: potential changes to regulation, regulator delay, transboundary impact assessment at the request of neighbouring countries, public challenge to the PUZ, EIA, operation permits stages, and administrative appeals, etc.

Table 20.1: RVP Permitting Timeline and Schedule

<table>
<tr><th>Type of Permit</th><th>Date Issued</th><th>Description</th></tr>
<tr><td rowspan="5">Mining</td><td>2004</td><td>Non-exclusive Prospecting Permit</td></tr>
<tr><td>2005</td><td>Exploration Licence (Rovina no. 6386/2005) by the NAMR</td></tr>
<tr><td>2009</td><td>Extension of Exploration Licence</td></tr>
<tr><td>2014</td><td>Preliminary EIA prepared for mining licence purpose only</td></tr>
<tr><td>2015, 2018</td><td>Mining Licence no. 18174 for the RVP (100 % owned by ESM), approved by the Romanian Government by Government Decision no. 900/09.11.2018</td></tr>
<tr><td rowspan="5">Land use change</td><td>2020</td><td>Urban Certificate no. 122/11.06.2020 issued by Hunedoara County Council for land use change by the PUZ approval</td></tr>
<tr><td>2020 - Ongoing Process</td><td>Opportunity Agreement Approval to be issued by Hunedoara County Council for the PUZ</td></tr>
<tr><td>April 2021 (to be further confirmed)</td><td>Start the Strategic Environmental Assessment (SEA) procedure to obtain the environmental licence for the PUZ approval, issued by the Hunedoara Environmental Protection Agency</td></tr>
<tr><td>April 2021 – ongoing</td><td>Start the permitting procedures to obtain the other approvals and permits required for the PUZ approval: Army Central Administration, Romanian Intelligence Service, Hunedoara Police Inspectorate, regional electricity service provider (E-Distribuţie Banat), Hunedoara Forestry Direction, Hunedoara Cultural Direction, Water Management Administration Crişcior, Road Administration Direction of Hunedoara County Council, Road Administration of Local Roads from Buceş, Bucureşci and Crişcior, Hunedoara Real Estate and Cadastre Office, Land Improvement National Agency – Hunedoara Office, affected land and property owners, Ministry of Agriculture and Rural Development, Chief Architect of Hunedoara County Council, Buceş, Bucureşci and Crişcior Mayoralties</td></tr>
<tr><td>To be determined. Exact timeline depends on the issuance of the above mentioned approvals</td><td>PUZ approval by Hunedoara County Councill</td></tr>
<tr><td rowspan="4">Construction</td><td rowspan="4">Once the PUZ is approved</td><td>Urban Certificate (UC) for construction</td></tr>
<tr><td>EIA Study and permitting process to obtain the environmental agreement to be issued by Hunedoara Environmental Protection Agency, based on the Decision of the Romanian Government and considering the proposal of the central public authority for environmental protection</td></tr>
<tr><td>Other approvals required in the UC (majority from regulators mentioned above at PUZ approval stage)</td></tr>
<tr><td>Construction permit by Hunedoara County Council</td></tr>
<tr><td>Operation</td><td>Once constructions are finalised, equipment is installed, and the mine is ready to operate</td><td>Environmental Permit, Water Management Permit, Emergency Situation and Firefighting Permit, Health Permit, Labour Permits, etc.</td></tr>
</table>

According to the current permitting plan, SAMAX estimates finalising the above-mentioned permitting steps and starting the construction phase in May 2023, and starting production in December 2024.

SAMAX has workstreams under way and will work closely with the plan developer, and in line with its communication strategy and stakeholder engagement plan, to facilitate these processes and manage project risks. Environmental information gathered by the RVP will be shared with the plan developer to ensure that the SEA (following Governmental Decision no. 1076/2004 to implement the 2001/42/EC EU SEA Directive) will accompany the spatial plan. The environmental information provided at the SEA stage will be compatible with the project-specific EIA (following Law no. 292/2018 to implement the 2011/92 EU EIA Directive), which will be required under the Romanian and EU legislation, and an ESIA study in line with international standards (EP4, 2012 IFC Performance Standards), which will be voluntarily developed by SAMAX to fully identify and mitigate the impacts at the level of international lenders' expectations.

Applications are needed for regulator conditions on water management and will require background information and engagement with the regulators. These applications form the first part of a three-stage process to obtain the Water Permit (for the PUZ, for construction and for operation), which covers abstraction, use and discharge of water for mining projects and the process plant's operation, and also the storage and discharge of hazardous and other substances that can pollute water.

20.4 PROJECT AND ESIA STUDY OBJECTIVES

The primary objective of the RVP is to safely develop and operate the open-cast mining and ore processing of the Rovina and Colnic deposits, whilst remaining compliant throughout with environmental laws and guidance and maintaining the social licence to operate. The ESIA is the internationally recognised approach, enshrined in EU and Romanian laws, that will:

- Determine the baseline environmental and social conditions.
- Identify known or potential impacts on the environment and society.
- Assess the significance of these impacts.
- Identify mitigations of these impacts.
- Develop environmental and social management plans to implement these mitigations.

An ESIA is a requirement for a large project such as the RVP and is a substantial undertaking, requiring the collection of baseline data over several years. The process of baseline data collection has been under way at the RVP for over a decade, especially since 2012, and the findings of these, and ongoing plans, are summarised later in this section. Following the preliminary EIA in 2014, performed to obtain the mining licence, the EIA for the RVP is expected to be submitted to the regulators early in 2022. In parallel, an international ESIA is being undertaken by SAMAX, to fully comply with international/IFC standards and lenders' expectations. The outstanding baseline programme serving both the local EIA and International ESIA studies includes the following:

- Conducting surface monitoring and developing groundwater models
- Completing terrestrial and aquatic biodiversity assessments
- Completing data collection for the cultural heritage baseline
- Conducting air quality, noise, and vibration monitoring
- Identifying local users of water
- Conducting a social baseline and census and socio-economic survey for landowners and those who will be physically and economically displaced by the project

A key requirement of the baseline and impact assessment process is a detailed understanding of the project layout and activities, and mitigations of impact already inherent or planned for in the project design. These are described in Section 20.5 in relation to the environmental media (soil, surface water, air quality, etc.) or **Section 20.6** in relation to social context.

20.5 ENVIRONMENTAL BASELINE SETTING

Studies and surveys to establish the RVP environmental baseline have been under way since 2012, contributing to the preliminary EIA (2014), and are ongoing. Table 20.2 provides a summary of the environmental studies already completed and those planned in the next 12 months. Baseline studies completed to date have provided an understanding of the key environmental sensitivities in the area. The schedule of future work has been planned to meet national permitting requirements, align with international good practice, and to help further understand or close out environmental risks.

Table 20.2: Environmental Studies Completed and Planned

Environmental Media	Work Completed	Work Planned
Surface Water	Surface water flow and quality monitoring (semi-continuous automated probe readings at 17 locations across the RVP area since 2008) Preliminary surface water and sediment quality monitoring (2006–2010) Comprehensive surface water quality monitoring – first campaign completed (2020) Hydrological studies undertaken in area Catchment scale groundwater model design of diverting channel in Rovina Valley	Comprehensive surface water quality monitoring – second campaign in 2021 Water treatment scheme for drinking water facility Wastewater treatment and discharge scheme, to be determined
Groundwater	Preliminary, historical well and springs monitoring (2006–2010) Installation of five new monitoring wells (2020) Comprehensive groundwater quality monitoring (historical and new wells, springs) – campaign completed (2020)	Develop groundwater flow model for diversion tunnel area
Terrestrial Ecology – Biodiversity	Habitat and species surveys undertaken in 2017–2018 and October 2020 to January 2021. Surveys included habitats and flora, invertebrates, mammals, bats, birds,	Continue surveys during 2021 to update the baseline conditions covering the project area of influence and to identify potential

Environmental Media	Work Completed	Work Planned
	amphibians, and reptiles	species of conservation interest
Aquatic Ecology	Surveys included aquatic macro-invertebrates and fish (started October 2020)	Continue surveys during 2021 to characterise the baseline condition and to identify potential species of conservation interest
Air Quality	Twelve months of air quality monitoring (oxides of nitrogen, nitrogen dioxide, sulphur dioxide, particulate matter (PM10 and PM2.5) and dust deposition with associated metals analysis) in line with good international practice	Complete baseline survey and studies, except for PM10 monitoring, which will continue until October 2021
Noise and Vibration	Surveys and studies are in progress	Complete baseline survey and studies
Landscape	Observations made and photographs taken of sensitive viewpoints	Complete baseline survey and studies
Soils and Geology	Detailed geological mapping since 2005, and shallow soil metal geochemistry detailed surveys in 2007 as part of mineral exploration Soil sampling for land capacity (agricultural) and chemical quality (pollution) in 2020 Health Institute sampling of soils in residential settings in the area of interest (2020)	Report 2020 survey results
Geochemistry	Baseline completed. Sampling and analytical programme to determine ARD/ML capacity of the waste rocks (2012) and two tailings streams (2021)	Further ARD studies may be undertaken during operations

20.5.1 Physical Environment

The property is located in the South Apuseni Mountains, which are mostly gently rolling with some abrupt slopes and cliff-forming rock exposures. The highest peaks near the property are the Duba Peak (969 masl), Coasta Mare Peak (786 masl), and Cornetel Peak (695 masl).

In the Rovina Valley, the terrain is hilly to mountainous, with access through narrow valley floors and moderately steep slopes up to rounded ridges. The minimum and maximum elevation ranges for each of the deposits are 350 m to 540 m for Colnic and 500 m to 680 m for Rovina. The area is mostly forested with deciduous forests (beech and oak) and occasional conifers, particularly at higher elevations.

The regional climate is mild temperate continental, with winter months from December to March and snow accumulation typically less than 30 cm. Mean winter temperatures are in the range of –3 °C to –5 °C, with a minimum of approximately −20 °C. Springtime temperatures of 5 °C to 10 °C may start in early April, but snow is found until mid-May in the forested areas. In the summer months, from June to September, the temperatures range from 10 °C to 20 °C, with rare maximum highs near 35 °C. The typical annual precipitation is 800 mm to 1,100 mm.

20.5.1.1 Geology and Soils

The geology of the South Apuseni Mountains is dominated by Neogene volcanic and sub-volcanic rocks, subdivided into three main groups:

- Early Miocene acidic tuffs and ignimbrites
- Mid-Miocene to Pliocene calc-alkaline stratovolcanoes (associated with epithermal and porphyry mineralisation)
- Pliocene to Pleistocene alkaline volcanic rocks

The RVP area comprises a sequence of Neogene subvolcanic intermediate intrusive rocks. The exploration programmes identified gold-rich porphyry systems (Colnic and Ciresata deposits) and a copper-gold porphyry (Rovina) associated with these Neogene subvolcanic intrusive complexes. The geology and mineralisation are detailed further in Section 7.

Soils are typically either residual or colluvial clays on valley slopes and ridges, or alluvial soils adjacent to watercourses on lower valley slopes and in valley floors. Residual clay and colluvial soils have relict clasts of bedrock of sand and gravel grade, weathered to fresh, tending to increase in frequency and clast size with depth. In the RVP, these classify as various cambisols and leptosols. Alluvial soils include thick (up to 8.5 m) fining-up sequences from basal cobble gravels to gravelly, clayey sands, which classify as fluvisol and luvisol. Topsoil is more organic rich with an average of 1.4 % total organic carbon, ranging up to 4 %.

At the Colnic pit site, there are some abandoned adits and mine waste dumps from 19th century workings. The dumps are relatively small, and the dump material is covered entirely or partly by vegetation.

High-resolution mineral exploration soil surveys in 2007 identified anomalous metal concentrations in the soils that overlie the mineral deposits, but recent chemical quality sampling in 2020 indicates that these do not exceed the maximum level admitted by national regulations for sensitive areas, except for lead, arsenic, copper and manganese in a small number of samples. The same survey identified trace organic compounds, such as petroleum hydrocarbons (up to 94 mg/kg) and traces of poly-nuclear aromatic hydrocarbons. These could be from burning, or peaty soils, rather than pollution. None of the soil samples detected sulphide (< 1 mg/kg), reflecting their already weathered state.

20.5.1.2 Surface Water

The most significant surface water in the RVP is the Bucureşci River (south of Colnic) and its tributaries, especially the Rovina Valley Stream that flows from the north and whose northern tributary valley (Catchment H) is the location of the pits and waste facilities. The Bucureşci River flows west to the Crişul Alb River at Crişcior, thereafter flowing westward to eventually cross the Romania-Hungary border and discharge into the major Tisza River. The Tisza River flows into Serbia and joins the Danube River just north of Belgrade; the Danube eventually enters the Black Sea within Romanian territory. The Rovina Valley Stream has a variable but permanent flow sustained by groundwater baseflow, whilst many of the tributaries are ephemeral, especially in the upper catchments.

There has been a programme of monitoring and sampling on the Rovina Valley Stream and some locations at Bucureşci, with semi-continuous automated probe readings for flow and quality at 14 locations across the RVP area since 2008. The same locations also had preliminary surface water and sediment quality monitoring biannually between 2006 and 2010.

A more comprehensive surface water quality monitoring programme has been developed for the ESIA, consisting of two campaigns for water quality, during dry and wet seasons. The first campaign took place in 2020 at 15 sampling points, located along the Rovina and Bucureşci (and other small tributaries), as well as on the Crişul Alb, in the water supply catchment area. The second campaign was done in the spring of March 2021 when high-flow conditions were expected.

The chemical and physical parameters that were analysed were selected to comply with the national standards as a function of their use:

- Parameters established by NTPA 001 and NTPA 013 (technical norms for discharge into a natural water body and the quality standard that must be met by surface water used for drinking purposes).
- Parameters for surface water quality by Order 161/2006 (the norm regarding the classification of surface water quality in order to establish the ecological status of water bodies).

The results from the first campaign indicate that most of the surface water analysed is in the best quality category for ecological status.

20.5.1.3 Groundwater

Alluvial soils in the valley floors, and adjacent to smaller watercourses, form high-permeability aquifers, limited in yield by their relatively thin strata and the narrow widths of the valleys in the hilly terrain of the RVP. Elsewhere, the clayey residual soils are thin and of relatively low permeability such that they act as aquitards, as does the underlying weathered bedrock. Monitoring wells installed through these often show groundwater to be confined by these aquitards. The fresh igneous bedrock has extremely low intrinsic porosity and permeability, but is lightly to moderately fractured, forming a high-permeability but low-yielding fractured bedrock aquifer. An aquifer abstraction zone is present in the Crişul Alb valley, north of Crişcior, in the Crişul Alb River catchment, less than 1 km west of the process plant area.

Typical for the region, communities often have springs or fountains that once served the water needs of local residents, and still do in many parts of the RVP, including the Rovina Valley. Five of these fountains, along with three wells, were monitored periodically between 2006 and 2010. It is not known which aquifers these different wells, springs and fountains are producing from.

In 2020, a more comprehensive groundwater quality monitoring programme was developed for the ESIA. Five new monitoring wells were installed in autumn 2020: two at low points of the waste facilities area; one between the waste facility and the Colnic pit; and two within the footprint of the Colnic pit (near the Rovina diversion tunnel location). The sampling campaign took place in December 2020 for all monitoring wells and the five communal springs/fountains.

The chemical and physical parameters that were analysed were selected to comply with the national standards as a function of their use:

- Parameters established by the law for drinking water (Law 458/2002, republished), for the five communal springs/fountains.
- Parameters for the quality of underground water bodies (G.D. 53/2009, amended).

The results indicate that the water from the communal springs/fountains does not meet drinking water quality standards. The groundwater sampled from the monitoring wells has the typical characteristics for the area with no evidence of elevated concentrations of metals or other pollutants, and so is considered to be of good quality.

20.5.1.4 Air Quality

The RVP does not have major pollution sources or industrial zones within or close to its boundaries. The only current air pollution sources are related to the use of solid fuel for household heating in the localities of Crişcior, Bucureşci, Merișor and Rovina, and road traffic along the DN 74 Abrud-Brad and DJ 741 Crişcior-Curechiu routes.

Industrial activities are not well represented in the wider region and do not significantly affect the air quality. Mining was historically important in the region until recently, and there are a number of old waste facilities and tailings deposition ponds outside of the RVP; however, most of them are partly covered by vegetation and raise no significant concerns in terms of airborne dust emissions at the RVP.

The baseline surveys for air quality or noise surveys are in progress. Greenhouse gas emissions have yet to be assessed.

20.5.2 Biological Environment

There are no designated protected areas for biodiversity in or around the main infrastructure footprints of the RVP. However, the RVP area is characterised by a high diversity of species and habitats, including some of conservation interest, typical for forested areas in hilly regions of Romania.

The dominant habitat is deciduous forest, interspersed woodland and grassland areas, as well as agricultural land comprising pasture and orchards with scattered homesteads. The majority of the pastureland is disused, mainly reverting to meadow and woodland, and supports good species diversity. Some areas are composed of mature forest and likely to sustain high species diversity. A seasonal watercourse supports remnant areas of the alluvial forest, which is of conservation interest.

During the baseline surveys, ten types of forest habitats have been identified based on the Romanian classification system, out of which nine may have Natura 2000 correspondence. Semi-natural grassland was identified as an important habitat for orchids and potentially could support other rare and protected plant species.

Five amphibians and five reptiles were recorded during the surveys. All species identified on site are listed in the IUCN Red List (Global) as Least Concern (LC). Based on the Romanian Red Book of vertebrates, three amphibians and three reptiles are listed as Vulnerable (VU). All amphibians and four reptiles are listed in Annex IV of the EU Habitats Directive (92/43/EEC) for species of community interest in need of strict protection at a European level.

Surveys identified 70 bird species within the RVP area. All species recorded as present on site are listed in the IUCN Red List (Global) as LC. Of the species recorded, 14 are listed in Annex I of the EU Birds Directive (2009/147/EC) as requiring special conservation.

The 2017–2018 survey identified 23 mammals recorded or likely to be present within the RVP area, one of which is listed as Endangered (Bicolored shrew *Crocidura leucodon*), three as VU and two as Near Threatened (NT), namely the Eurasian otter *Lutra lutra* and Brown bear *Ursus arctos*, based on the Romanian Red Book of Mammals. Six mammal species are listed in Annex IV of the EU Habitats Directive.

In a limited bat survey in autumn 2020, the presence of three species was indicated: *Barbastella barbastellus*, *Nyctalus noctula* and *Pipistrellus pipistrellus*. All bat species are protected under Romanian law and are listed in Annex IV of the EU Habitats Directive. Aquatic ecology surveys undertaken in autumn 2020 identified the presence of eight fish species, all listed in the IUCN Red List (Global) as LC.

No protected or internationally recognised sites for biodiversity were identified within 1 km of the RVP. The closest protected area lies approximatively 1.4 km from the exploitation perimeter (Natura 2000 Special Area of Conservation, ROSPA0132 Munții Metaliferi). Further investigations and assessment will determine if the protected site will be impacted by the project. To inform further assessment, additional baseline studies are planned for 2021, including surveys for bats, protected flora and aquatic ecology, which will further inform the EIA.

20.6 SOCIAL BASELINE SETTING

Studies and surveys to establish the RVP social baseline have been under way since 2009, contributing to the preliminary EIA (2014), and are ongoing. Table 20.3 provides a summary of the social studies already completed and those planned in the next 12 months. Baseline studies completed to date have provided an understanding of the key social sensitivities in the area. The schedule of future work has been planned to meet national permitting requirements, align with international good practice, and to help further understand or close out social risks.

Table 20.3: Social Studies Completed and Planned

Aspect	Work Completed	Work Planned
Social Baseline	Household surveys, focus group discussions, informant interviews	No further works are currently planned
Land Acquisition	Survey of land to be acquired undertaken in 2012, including details of ownership, and potential costs	Census and socio-economic survey, in line with good international practice

Aspect	Work Completed	Work Planned
Cultural Heritage	Extensive archaeological surveys across the RVP area during 2009, 2013 and 2016	Further field-based surveys of the project areas not covered by the current baseline and additional surveys to include architectural (built) heritage, industrial heritage or intangible cultural heritage

20.6.1 Main Economic Activities

There is a strong and long-standing mining tradition in the Apuseni Mountains, and many local community residents regard themselves as “proud miners”. However, mining activities in the region were abandoned 15 years ago, resulting in a steady economic decline, high rates of unemployment, and out-migration, particularly of skilled workers and youth.

In the urban area of Brad, 6 km west of the RVP, the primary economic and livelihood activities include waged labour in the private sector such as medium-sized industries (supporting the automotive sector), commercial shops and services. The public sector is also a key employer through the mayoralty, fiscal authority, educational and health systems. Similarly, the Crişcior settlement on the border of the RVP, which is a semi-urban area and the heart of the former mining activities, displays a mix of economic activities and livelihoods similar to those of Brad. A number of private companies, including the RVP developer, have registered offices and activities in Crişcior. The commercial sector is represented through supermarkets and shops, coffee shops as well as transport (taxi) services. However, it is noted that most waged labour occurs in Brad and people from the other settlements mostly commute to Brad for work. Secondary livelihoods consist of agriculture and some animal husbandry activities; these are undertaken by those living at the town periphery and are mainly conducted for direct household consumption.

20.6.2 Population

The RVP footprint (approximately 470 ha) covers the community areas of Bucureşci (pop. 506), Rovina (pop. 128), Merișor (84) and Crişcior (2,684). The town of Brad (pop. 15,361) represents the socio-economic centre of the area.

20.6.3 Household Structures

Most of the households are multigenerational; typically, there will be a retired miner and at least one employed person. With retirement pensions higher than wages paid in the area, most households are heavily dependent on the retired miners as the main source of income. Remittances from relatives living abroad are not widespread or significant.

20.6.4 Electricity

All the settlements in the area are connected to the national electricity distribution grid through 0.4 kV overhead or underground transmission lines. Within the households, electricity is used mainly for lighting and to a small extent for cooking (electrical ovens) and heating via small units.

The RVP will install its own separate electricity power lines from the national electricity grid, and there will be transformers at the process plant and maintenance areas.

20.6.5 Water Supply

Table 20.4 presents the water supply arrangements in the RVP area.

Table 20.4: Access to Water and Sanitation

Municipality/ Commune	Settlement	Urban/Rural	Water Supply Modality			Wastewater Treatment Type	
			Percentage of Households with Access to the Centralised Supply System	Percentage of Households Using Spring Water Captured Gravitationally in Tanks	Percentage of Households Using Wells/ Fountains	Percentage of Households with Access to Centralised Sewage System	Percentage of Households with Indoor Bathrooms which have Access to Own Septic Tanks
Bucureşci Commune	Bucureşci	Rural	65 %	30 %	5 %	5 %	55 %
	Curechiu	Rural	0 %	70 %	30 %	0 %	40 %
	Merişor	Rural	0 %	70 %	30 %	0 %	25 %
	Rovina*	Rural	0 %	70 %	30 %	0 %	30 %
	Şesuri	Rural	0 %	70 %	30 %	0 %	45 %
Crişcior Commune	Crişcior	Rural*	90 %	0 %	10 %	60 %	40 %
	Valea Arsului	Rural	0 %	10 %	90 %	0 %	100 %
	Zdrapţi	Rural	0 %	20 %	80 %	0 %	100 %
	Barza	Rural	15 %	15 %	70 %	0 %	100 %
Buceş Commune	Buceş	Rural	73 %	NA	100 %	0 % – sewage system not operational	NA
	Mihăileni	Rural	NA			0 % – sewage system not operational	NA
	Stănija	Rural	NA	NA	NA	NA	NA
Brad Municipality	Brad	Urban	70 %	0 %	50 %	40 %	40 %
	Tărăţel	Rural	NA	NA	NA	NA	NA

* Although classified as a rural settlement, the Crişcior settlement has a semi-urban character due to its previous development at the height of the mining sector operations.

Source: ERM Social Field Survey, November 2020

20.6.6 Community Health and Sanitation

Healthcare facilities are concentrated primarily in Brad, and to a lesser degree in Crişcior. The Buceș, Bucureșci and Crişcior villages host medical centres servicing the entire administrative unit. The Brad Municipal Hospital is the main secondary healthcare facility in the area, complemented by the Deva Emergency Hospital. Emergency and some inpatient services are also available at the Crişcior Permanence Centre. Medical facilities in the study area are as shown in Table 20.5.

Table 20.5: Medical Facilities in the Study Area

Facility Type	Brad	Crişcior	Bucureşci	Buceș
Hospital	2			
Polyclinic	1			
CMI/Clinic	9	1	1	1
Dentist	2	1		
Pharmacy	8	2		
Ophthalmologist	1			
Medical Laboratory	1			
Permanence Centre		1		
Occupational Risk Unit		1		
CMI Family medical clinic				
Source: ERM fieldwork, 2020				

20.6.7 Education

Schools in small towns and rural areas tend to have fewer students and classes and are not able to attract highly qualified teachers, even if they receive more funding per student. Decreasing numbers of students lead to closure of schools in rural areas and raise challenges in accessing education. Consequently, education facilities are located in the three commune centres (Crişcior, Bucureşci and Buceş). With one exception (in the Zdrapţi village, Crişcior commune), schools in the other villages of the communes were closed over the past decade, due to decreasing numbers of pupils.

The following types of education are available:

- In the Bucureşci and Buceş settlements, schools offer preschool, primary and lower secondary education.
- In the Crişcior settlement, Crişan Technical High School offers preschool, primary and technical secondary education. It offers the following specialties: mechanics, electro mechanics, tourism, and food industry.
- In Brad, Avram Iancu Theoretical High School offers science and humanities as key study directions.

The closest universities to the area are in Deva, Petroşani (Hunedoara County) and in the Arad, Timiş and Cluj counties.

Mining tertiary education is available through the Mining Faculty of Petroşani University, which includes the following departments: Department of Mining Engineering, Topography and Constructions, the Department of Environmental Engineering and Geology, and the Department of Management and Industrial Engineering.

20.6.8 Cultural Heritage Sites

There are no designated protected areas for cultural heritage in or around the main infrastructure footprints of the RVP. Archaeological investigations have been conducted by the Deva Museum of Dacian and Roman Civilisation during three field campaigns (2008–2009, 2012–2013, 2015–2016) with a total of 166 archaeological trenches excavated.

Detailed archaeological investigations were also carried out in 2016 over two areas previously identified as having archaeological potential. Here, the archaeologists identified some bronze-age artifacts around what has been considered two seasonal habitation areas. All the artifacts have been collected by the Deva Museum; the conclusion being that these areas can be archaeologically discharged. No other prehistorical or Roman period vestiges have been identified in the RVP.

An 18th century wooden church in the Rovina village has been identified as a cultural-heritage site. The ESIA studies will address potential impacts on both tangible and intangible heritage.

20.7 MINE CLOSURE

Mine closure is a critical element in the life cycle of the mine and is the phase in which many of the long-term, large-scale impacts, such as those to soil, biodiversity and water, are mitigated.

ESM is committed to following international best practices regarding Environmental, Social and Governance (ESG) principles throughout the development and operation of the project. Some examples of these best practices are that ESM has consciously designed the project with a dry-stack tailings deposition facility and eliminated the use of cyanide in the ore processing. ESM have chosen a high degree of stakeholder engagement from the outset, with consultation on the layout of infrastructure, and have responded to community concerns, such as tailings stability and the use of cyanide. The decision to undertake early, comprehensive surveys enabled ESM to receive an Archaeological Discharge Certificate and plan a layout that avoids impacting any sites in this historical region. Together, these decisions by ESM mark their commitment to responsible and sustainable mining.

A Mine Closure Plan (MCP) was developed in 2014 at the same time as the preliminary EIA, to a slightly different project design plan than the current RVP. However, much of the infrastructure type, location and parameters are similar or the same, and the 2014 study forms the basis of the MCP under development for the RVP.

The RVP MCP is designed to be in line with national requirements and international good practice, e.g. EU Extractive Waste Directive, EU Best Available Techniques, IFC Health and Safety Guidelines on Mining, and the International Council on Mining and Metals Good Practice Guide. Romanian Mining Law (85/2003, and the norms and orders relating to it) requires a financial guarantee for environmental rehabilitation, to be constituted at a bank approved by the NAMR, to cover the operations for each subsequent year, to be renewed annually. Currently, this guarantee is proportional to the works proposed in the annual exploitation programme (e.g. geotechnical drilling and geophysics), but once operations commence, a substantial proportion of the mine closure cost estimate will have to be guaranteed.

Mine closure costs are estimated to be US$20 million, although post-closure costs and passive care for the surface water management systems are included in the water management OPEX (see Section 21).

The approach to closure will be to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use. The current MCP vision is to restore most of the site to pre-mining land use and status, namely mixed upland forest. The aim of the MCP will be to minimise or eliminate long-term active aftercare such as water treatment requirements.

Monitoring of groundwater and surface water (levels and chemistry) will be undertaken throughout the LOM to monitor the impacts of mine dewatering and develop an appropriate closure strategy to ensure that adverse impacts on hydrogeology and hydrology do not occur post-closure. Upon closure, there will be a phase of further site investigation, risk assessment and regulatory liaison to identify any sensitivities or requirements relating to post-closure soil or water quality and to develop remedial action plans that may be required in this regard, including detailed scopes and costs for work programmes. This will be followed by pre-demolition assessments to develop a demolition contract package (based on structural surveys and the collation of structural details of all buildings) indicating which buildings, facilities and equipment will be decommissioned. At this stage, community continuity projects will assess the impact of the closure on the community, and mitigation will be developed in a programme of initiatives.

Decommissioning will be undertaken by specialist contractors who will be contractually required to operate at all times in accordance with the relevant legislation and international best practice.

The mine closure activities will vary between the four main project elements: the processing plant and maintenance areas, the two pits, and the waste facilities. These are discussed separately below, as well as the site monitoring during closure.

20.7.1 Processing Plant and MIA

Following decommissioning, structures no longer required will be demolished, foundations removed, and voids backfilled. Demolition rubble and any waste will be removed from site, and the land will be levelled and restored using stockpiled soils and planted to forest.

Drinking water treatment facilities and pipelines in the plant area that served the Rovina Valley communities will be retained and transferred to the local authorities such that the mains water supply continues after the active closure period.

20.7.2 Colnic Open Pit

Once the pit has reached the maximum design extent, it will be progressively backfilled with waste rock and co-mingled filtered rougher scavenger tailings from the ongoing operations to the design elevation, prior to being contoured and capped with stockpiled topsoil and planted to forest. A new streambed for the Rovina Valley Stream will be laid, mimicking the width and sinuosity of the natural bed and, once completed, the non-contact diversion tunnel will be decommissioned and sealed.

20.7.3 Rovina Open Pit

The Rovina pit will intersect the northern upper headwaters of Catchment H. Once the pit has reached the maximum design extent, the closure concept is to form a lake. To achieve this, the pit slope gradients will be adjusted, and subsoil and topsoil will be deposited on the upper part of the pit slopes, which are above the design lake elevation, and planted to forest.

20.7.4 Waste Facilities

The mineral waste facilities will cover a large area of the western Rovina Valley (Catchment H) and will be composed of a dry stack of waste rock co-mingled with filtered rougher scavenger tailings, deposited in raises by conveyors. ARD studies (see Section 18.5) indicate that these are geochemically stable over the short and medium term with the co-mixing of tailings reducing porosity, airflow, and water permeability within the waste rock pile. The closure concept is to provide a low-permeability cap and maintain diversionary channels and underdrains such that the waste facility is permanently unsaturated and geochemically stable in the long term. Caps will be emplaced progressively as sections of raises are completed.

As each section of the waste facility is completed, geotechnical stability will be achieved by slope adjustment works and clearing of oversized material from the surface, followed by stabilisation and consolidation of slopes with fences along the contour lines, support walls and perimeter ditches. The cap will be constructed by the deposition of a compacted clay layer, up to 0.5 m thick, to provide a low-permeability surface, which will be overlain by stockpiled topsoil and then planted with trees to achieve a full reforestation of the waste facility platforms and slopes.

Following these active closure works, there will be ongoing monitoring work, including deformation monitoring and visual surveys to ensure that the cap is not undergoing erosion. Maintenance will also be required post-closure, including the upstream diversion ditches, the contour and perimeter drains on the waste facility, forestry management, and the infilling of gaps that may develop over time.

The waste facility will also include a disposal facility cell for a separate, lower volume, waste stream of cleaner tails that will have a sulphide-rich content that makes it prone to ARD and ML (see Section 18.3 and 18.3.3). These tailings will be filtered and deposited in a special lined cell within the waste facility to isolate it from any percolating water and ensure geochemical stability in the long term. The basal seal and the cover cap will include both a compacted clay layer and geomembrane, and will be located on the upper, south-western side of the co-mingled rougher tails and waste rock waste facility.

20.7.5 Monitoring in Closure

Throughout decommissioning and into the closure period, site monitoring will include a programme of surface water and groundwater, air emissions and ecological monitoring. Contact water treatment from the waste facility underdrains will continue, although volumes produced after the caps are installed are expected to be low. Monitoring will be conducted after closure for a defined period or until the predefined success criteria are achieved, such as an appropriate concentration in the discharge is guaranteed (as defined by competent specialists) without active water treatment.

20.8 KEY ENVIRONMENTAL AND SOCIAL IMPACTS

Key environmental and social risks are similar to those associated with other gold mining projects and include the social licence to operate; safeguarding rivers, groundwater and biodiversity and mitigating permanent effects; and the risks associated with resettlement.

An EIA undertaken for the RVP in 2014, to a slightly different project design plan, identified the following aspects as key risks:

- **Air Quality**
  Air quality would be impacted, both in ambient quality and as dust deposition. Even with strict dust suppression and monitoring, it cannot be ensured that adequate air quality will be maintained in the area of households at relatively short distances from the haul roads and the Rovina and Colnic pits.
- **Noise and Vibration**
  Noise and vibration impacts that are unacceptable would occur to households at relatively short distances from the haul roads and Rovina and Colnic pits.
- **Biodiversity**
  Although no critical habitats are within the impacted area, there would be a large magnitude loss of habitat at the pits and waste facilities that needs to be carefully managed and offset.
- **Community Impacts**
  Ten plots with inhabited houses need to be resettled, plus other households in the Rovina Valley close to the pits and haul roads would be impacted by dust, noise and vibration at what may be high levels. Elsewhere, there are no significant impacts from changes to environment, and only positive impacts from economic development.

The section below reassesses the RVP impacts on the different environmental and social aspects previously described. Key mitigations for risks and impacts are presented, many of which will occur during the mine closure period.

20.8.1 Surface Water

The RVP will have impacts on surface water in terms of both availability and quality. These are described below, but the overall conclusion is that, with careful management, these are not likely to be significant, despite permanent alterations to the hydrological system.

20.8.1.1 Surface Water Availability

The RVP water supply will be provided by process recycle water, pit dewatering and make-up raw water gathered in the contact water pond from across Catchment H. The water will be used for process water and other RVP needs (such as dust suppression, truck washing and the firefighting system), and a large proportion will be treated for potable water use in the RVP and for a new mains water supply network to residents in the Rovina Valley. Baseline water sampling identified the poor water quality used by residents in the Rovina

Valley, and the installation of mains water is a positive impact that will persist after the closure.

The development of the waste facility will affect most of Catchment H. Diversion channels will have reduced catchments and consequently less water will enter the Rovina Valley Stream. Rainfall on the waste facility and pit areas will be collected as contact water and used as raw water, bypassing much of the Rovina Valley Stream. Dewatering of the pits during operations will reduce baseflow from groundwater to the Rovina Valley Stream and its tributaries. However, as the dewatering supply is used in process water and other uses, much will return to the river network following treatment. Water flow will be monitored during operations and active closure to ensure that the ecological baseflow is maintained.

The Rovina pit will be converted to a new lake during the closure period, whilst the Colnic pit will be infilled, and the landscape restored. The active closure period will end when active water treatment is no longer required and, at this point, water from the run-off drains and underdrains can directly discharge to the Rovina Valley Stream, re-establishing flow conditions to near baseline along the Rovina and tributary valleys.

20.8.1.2 Surface Water Quality

Diversion channels will separate contact from non-contact water in the waste facility and pits area, with the non-contact water directed to the Rovina Valley Stream and contact water from run-off drains, underdrains and pit dewatering directed to the contact water pond to be used for process water. Wastewater will be treated to national NTPA 001 standards, prior to being piped to the non-contact diversion tunnel for discharge.

The active closure period will end when active water treatment is no longer required and, at this point, water from the run-off drains and underdrains can directly discharge to the Rovina Valley Stream.

20.8.2 Groundwater

Groundwater is currently an important resource in the Rovina Valley, where most households are dependent on wells or communal fountains for water supply. Recent baseline surveys have found that residential water sourced from these wells and communal fountains does not meet drinking water quality standards and the RVP plans to install a new potable water network in the Rovina Valley, removing dependence on groundwater.

Groundwater is present in thin and laterally restricted aquifers in the alluvial soils and is also present in a high-permeability but low-storage fractured bedrock aquifer, both of which are of good quality. The discharge of hazardous substances into the groundwater is prohibited (Government Decision No. 516/2016 amending Annex 2 to the national plan for the protection of groundwater against pollution and deterioration, approved by Government Decision No. 53/2009), and so potential consequences of impacts to groundwater quality are high.

The greatest potential risk to groundwater quality is from ARD and ML at the waste facilities. Extensive ARD testing (see Section 18.3.3) demonstrates that the waste rock is PAG but that the rougher-scavenger tailings are non-PAG, will provide additional alkalinity, reduce the net sulphide mineral content, and their fine particle size will reduce the permeability of the mix. The co-mingled material will, therefore, not be PAG in the short to medium term. Underdrains will be laid out to collect percolating rainwater that will occur during operations, which will be treated as contact water. The waste facilities will be capped with low-permeability clays, and contour drains will rapidly conduct rainfall off the waste facility surface. Caps will be installed progressively during operations and completed during closure so that the long-term risks of ARD and ML will be mitigated. A specially designed waste facility will seal in the lower volume cleaner tailings with high ARD potential.

Groundwater availability will be affected at various points during the construction and operational period due to dewatering of the open pits and the diversion tunnels. This may lead to reductions in baseflow to the Rovina Valley Stream and tributaries, wells and springs, especially in the area around the Colnic pit. The installation of a mains water supply system by the RVP to the residents of Rovina removes their dependence on substandard groundwater and also frees them from the effects of any impacts on the groundwater (e.g. from accidental spills or reduced flow in wells and fountains due to dewatering).

Taken together, these measures should ensure that there is no significant deterioration of the groundwater quality, and in the closure period, the groundwater should be re-established to near baseline levels.

20.8.3 Soils, Land Capability and Land Use

The RVP infrastructure will require large areas of topsoil stripping at the footprints of the two pits, the waste rock facilities, the process plant and MIA, and along the new access and haul roads. This topsoil will be stockpiled in managed areas for later restoration purposes.

There are further risks to the soil quality at the process plant and the MIA from potential leaks of fuel, lubricant, solvents, flotation reagent, solid and liquid wastes and fugitive dusts to the soil. Industry standard controls are incorporated in the design, such as covered and sealed hardstanding for ore processing, fuel, reagent and waste storage areas, as well as bunding of liquid storage and usage areas, and controlled drainage directed to contact water treatment facilities (see Section 18). Good operational management for dust control and good housekeeping will also protect soil quality, which has an ongoing protective effect on surface water run-off and groundwater quality.

Erosion of soils may occur on exposed soils on road and platform cuttings, around the diversion channels, at the diversion tunnels and the discharge point on the Rovina Valley Stream, and on the restored topsoil surface of the waste facilities as they are capped. Careful management to reduce loss of soils into the surface waters will be required throughout the construction, operation and closure phases in these areas.

Soil quality, land capability and land use will be permanently altered in these large footprints; however, the main mitigation for this is in the closure period where most of the areas will be capped and laid with stockpiled topsoil prior to planting to forest. A relatively small area will be permanently lost as part of the Rovina pit will be restored as a lake. The closure plan will develop as the RVP proceeds and will include stakeholder engagement with local communities and planners. Some areas of soils could be restored with alternative land uses in mind (e.g. amenity and agriculture) according to a redevelopment plan as part of this closure planning.

20.8.4 Air Quality

Air quality at the RVP is not affected by pre-existing industry or affected significantly by transport, and it is only affected to a minor extent by domestic solid fuel emissions. The RVP will generate air pollutant emissions from the use of explosives in the pits, waste rock and ore loading into trucks, truck transport on haul roads, ore primary crushing and stockpiling, waste rock and tailings conveyance and deposition in the waste facilities, fugitive dusts from the processing plant, dusts from the uncapped co-mingled waste surfaces in dry conditions, and dusts from exposed soils in cuttings, roads and capped areas prior to forestation.

The dust-generating activities carried out within the RVP will be considered and reviewed for permitting purposes as surface sources generating diffuse air pollution. The primary ore crushing and stockpiling operation will be assimilated with a point air emission source. The ore transport operations by trucks will be considered linear/mobile pollution sources. All other air pollution sources related to the RVP operations will be diffuse air pollution sources. The nature of the air pollution sources within the RVP operations, apart from grinding in the processing plant, does not allow for the use of emission control installations. Mitigation by dust suppression on roads and plant will be undertaken.

Air quality is likely to be impacted, both in ambient quality and as dust deposition. Even with strict dust suppression and monitoring, it might not be possible to maintain adequate air quality in the area of households at relatively short distances from the haul roads and the Rovina and Colnic pits.

Greenhouse gas emissions have yet to be assessed.

20.8.5 Noise

The project will generate noise from construction and operational activities and closure. This is likely to affect the people living permanently in the area, as well as the fauna. A full assessment of noise and vibration will be required as part of the EIA, which will also need to take account of the existing background and historical conditions. It might not be possible to maintain an acceptable noise level in the area of households at relatively short distances from the haul roads and the Rovina and Colnic pits.

20.8.6 Terrestrial Ecology

The RVP infrastructure will occupy a large footprint characterised by deciduous forest, interspersed with woodland and grassland areas in mostly abandoned agricultural land, all supporting a high diversity of species and habitats, including some of conservation interest such as mature forest and remnant patches of alluvial forest. These are typical for forested areas in hilly regions of Romania, and whilst not unique or critical habitats, ten types of forest habitats have been identified based on the Romanian classification system, nine of which may have Natura 2000 correspondence.

Baseline surveys at the RVP have identified three amphibians and three reptiles listed as VU, and 14 species of birds listed in Annex I of the EU Birds Directive as requiring special conservation. Of the 23 mammal species identified, one is listed as EN, three as VU, and two as NT; a further six are listed in Annex IV of the EU Habitats Directive of species of community interest needing strict protection at a European level. Bat surveys have not been completed, but three species are likely present, and all bat species are protected under Romanian law and are listed in Annex IV of the EU Habitats Directive.

Based on available data, the presence of critical habitats is unlikely within the project area; however, the potential presence of Natura 2000 equivalent habitats and species listed in Annex IV of the Habitats Directive and Annex I of EU Birds Directive may meet the criteria for European Bank for Reconstruction and Development priority biodiversity features and an IFC natural habitat. These will require further investigation, evaluation, and management, including application of the mitigation hierarchy and careful biodiversity-led management of reforestation during the closure phase. Further baseline studies are planned for 2021.

20.8.7 Aquatic Ecosystem

Several rivers and watercourses cross the RVP, but the focus will be of impacts on the Rovina Valley Stream and tributaries (permanent and ephemeral), especially Catchment H. The stream hosts eight fish species, classed as being of LC, but the stream will experience considerable disruption to the hydrological system and aquatic habitats from multiple impacts during construction and operation of the RVP. These include the removal of Catchment H under the waste facilities, new diversion channels around the waste facilities and two pits, removal of the stream valley in the Colnic pit area, diversion through a tunnel around the Colnic pit, new discharges into the lower reaches of the Rovina Valley Stream, and reduced or altered groundwater baseflow from pit dewatering.

Many of these effects may be restored after closure, although some, such as the upper catchment diversion channels, will be permanent.

20.8.8 Socio-Economic

A strong positive impact on the economic aspects of the community is expected from the RVP due to the following:

- Development of new jobs and new economic activities
- Strong decrease of unemployment in the area
- Diversification of household incomes due to new job categories created directly and indirectly
- Increase of general income of population in the area
- Construction of a new drinking water supply and sanitation in the Rovina Valley
- Contribution of the RVP infrastructure, such as to road maintenance
- Regional and national tax revenue and royalties paid to the State

20.8.9 Land Acquisition and Resettlement

The RVP footprint (including the Ciresata deposit to the south) will affect 915 ownership plots: 719 are privately owned agricultural plots, 121 are private forests, and 75 are state forests.

The RVP will acquire land inside the planned construction footprint area in order to build the different mine elements and associated infrastructure. The mine will be developed in multiple phases and land plots will also be acquired and accessed in stages over the next 20 years, according to the mine development plan. The majority of the land acquisition is expected to occur over the next four years. Land for the establishment of the main linear infrastructure elements (water pipeline, access and haul roads, and power line) will need to be accessed first, followed by land for the processing plant and waste and tailings conveyors.

There is no right to expropriation for mining activities in Romania. Land acquisition for the project will, therefore, be conducted through voluntary land transactions with registered landowners and with the State for State (forest) lands. For private land, a number of steps need to be completed before a parcel of land can be acquired by the project and accessed for construction.

At this stage, 187 landowners of agricultural, forest, and pasture plots will be affected, including 43 landowners residing outside the area (in Bucureşci and Crişcior). Eleven households that will be physically displaced have been identified in the construction footprint. These households are all located in the footprint of the waste facility in the Remetea neighbourhood of the Rovina village (Bucureșci Commune). The extent to which these households will also be impacted by economic displacement is unclear at this stage and will be further assessed.

An RVP Resettlement Framework document is in preparation that will provide the foundation for land acquisition and the resettlement activities. Resettlement will be undertaken to restore the standards of living and livelihoods of those affected by physical or economic displacement, in alignment with national legislation and international good practice. The Resettlement Framework is dependent on engagement with key stakeholders and will require the informed participation of affected households and their representatives and the establishment of an appropriate institutional and governance structure.

20.8.10 Social and Community Engagement

Opposition to the RVP can be expected, for example, in 2015, Mining Watch Romania and the Save Roșia Montana Campaign strongly criticised the Romanian Government for granting the exploitation licence at the Rovina deposit, and Greenpeace and Friends of the Earth Hungary triggered the Espoo convention for the Rovina mine. Information produced by national non-governmental organisations (NGOs) was also referred to by mainstream media.

At the local level, however, previous engagement during the exploration phase has allowed SAMAX to establish a good relationship with the local community during which land was accessed for exploration without any resistance from landowners. During this time, the company supported various cultural and infrastructure projects in Rovina and the larger area.

SAMAX has maintained a Community Relations Office in Rovina since 2007 and has held presentations and consultation meetings regularly since then. Issues raised by stakeholders in relation to the project reflect the sensitive and complex context of gold mining in Romania and include the following:

- Concerns around the use of cyanide in gold mining projects (not being used by the RVP)
- Concerns around the large-scale deforestation needed for the project and overall long-term effects on human health
- The RVP is perceived as being a beneficiary of a simplified permitting process

- Concerns around the fact that mining means mono-industrial development and would thus mean that no other industry can thrive in the area
- An identified need for local employment opportunities with associated concerns around access to jobs and benefits by locals and impacts on local cohesion from this perspective
- Challenges associated with land acquisition, physical and economic displacement (Currently, a voluntary willing-buyer willing-seller approach is being pursued)
- Overall environmental impacts, particularly associated with destruction of landscape, air pollution, noise, soil contamination, pollution of water supply sources and rivers

The RVP will be subject to scrutiny by regulatory authorities and other stakeholders during the permitting process. There will be formal hearings and associated consultation periods included in the spatial plan, strategic environmental assessment and EIA processes, and a range of other points where the public and other interested parties could comment. The RVP will supplement these with dedicated stakeholder engagement activities.

SAMAX has developed an internal Stakeholder Engagement Strategy to support the RVP vision and its founding pillars, considered essential to manage social risk and to build and maintain a social licence to operate. The Stakeholder Engagement Strategy will be complemented by the Social Engagement Plan (in preparation), which will give a more detailed outline of the approach to engagement, specific for each project phase, and will present the process of managing community grievances.

20.9 SUMMARY OF KEY ENVIRONMENTAL AND SOCIAL RISKS AND MITIGATIONS

This section summarises the issues that *could* materially impact the RVP's ability to extract the mineral resources or reserves, and highlights the key mitigation measures planned to address these.

20.9.1 Social Licence

There is strong and well-organised NGO opposition to mining generally in Romania that has focused on the Rovina deposits in the past. However, local communities have a more nuanced view, coming from a strong mining heritage and seeing the potential socio-economic benefits to their communities. They also raise valid concerns on the overall environmental impacts, particularly associated with changes to landscape, deforestation, air pollution, noise, and possible effects on soil, water supply sources and rivers, and consequent effects on human health. The RVP has had a long period of community engagement that is ongoing and is well placed to communicate the mitigations that will be applied to the issues they are concerned about. However, there are numerous steps in the permitting process where public consultation occurs, where local community or external NGO concerns may raise the need for further studies, leading to project delays and unforeseen operational limitations or mitigations to the RVP.

A strong and effective Stakeholder Engagement Strategy and Stakeholder Engagement Plan, linked to the community concerns and the other issues in this section, and reflecting the mitigations these issues and concerns will require, will support building and maintaining a social licence to operate.

20.9.2 Resettlement and Land Acquisition

Eleven households are within the RVP infrastructure footprints and will require resettlement, a voluntary negotiated process in Romania. A similar process will be required for the landowner plots that will be required, step-wise, through the mine development, with the majority in the early construction phase. It is also possible that unacceptable impacts on air quality (from dust) and noise to households very close to the pits and haul roads may result in the need for further resettlement. All of these will interact with the social licence issue described above, potentially in a negative way, and could subject the RVP to project delays or unforeseen costs, operational limitations or mitigations.

The RVP has developed a Resettlement Framework, subject to national legislation and international good practice, to manage this complex and sensitive process.

20.9.3 Air Quality and Noise

Air quality is likely to be impacted, both in ambient quality and as dust deposition. Even with strict dust suppression and monitoring, it might not be possible to maintain adequate air quality in the area of households that are very close to the haul roads and the Rovina and Colnic pits. When considered with the noise issue that may affect the same households, it is possible that these households may also need to be considered for resettlement. Air quality in general, and potential further resettlement, will interact with the social licence issue described above in a negative way. Air quality and noise issues could subject the RVP to project delays or unforeseen costs, operational limitations or mitigations. Further baseline studies are planned for 2021.

20.9.4 Biodiversity

Recent surveys have found that, whilst neither critical or unique, the diverse forest habitats in the RVP contain areas that are of conservation interest and host one endangered and nine vulnerable species, as well as several species listed in the annexes of the EU Habitats and Birds Directives. These may meet the criteria for EBRD priority biodiversity features and IFC natural habitats, and will require further investigation, evaluation and management, including application of the mitigation hierarchy and careful biodiversity-led management of reforestation during the closure phase. Further baseline studies are planned for 2021.

20.9.5 Mitigated Risks

The RVP will cause high-magnitude impacts on soils, surface water and groundwater. However, these impacts and risks can be mitigated, to the extent that they are not likely to be significant, by the designs adopted by the RVP and by careful management of the specific structures, practices and plans that have been developed to protect them. The mine closure process is a critical element of this mitigation, based on a closure concept of reforestation of the waste facilities, plant areas and one of the pits, with the other pit being converted to a lake. In the long term, once geochemical and geotechnical stability has been achieved, the Rovina and tributary valleys, though altered, can flow once again without requiring active treatment.

20.10 ENVIRONMENTAL AND SOCIAL MANAGEMENT PLAN

Environmental and Social Management Plans (ESMPs) are critical in identifying, organising and implementing those specific studies, monitoring programmes and mitigating activities developed during construction, operation and closure to monitor and manage environmental and social impacts and key project risks.

The ESMP will provide an overview of the processes to identify, avoid, mitigate and manage the Project Environmental, Social and Health and Safety (ESHS) risks during the construction stage. The ESMP is the central document of the Project ESHS management system and is supported by a series of subordinated ESHS management plans and procedures implemented at company and contractor levels:

- Company Level ESHS Management Plans: These plans lay out the processes implemented by SAMAX to ensure that project policies, standards and commitments are attained during the construction stage of the project and to guide the EPC Contractor on the requirements and management plans to be implemented for the project as part of their ESHS management system.
- Contractor Level ESHS Management Plans: These plans are to be put in place by the EPC Contractor to ensure implementation of the project policies, standards and commitments during their own project construction activities.

In addition to the baseline data gathering listed in Table 20.2 and Table 20.3, the RVP will continue to enhance and develop the following plans and programmes:

- Resource Efficiency and Pollution Prevention and Control Plan
- Landscape Management Plan
- Biodiversity Management Plan
- Geohazards Management Plan (including soil erosion, landslides, debris flow, rock fall)
- Aggregate Management Plan
- Cultural Heritage Management Plan (including Chance Finds procedure)
- Emergency Preparedness, Response and Contingency Plan
- Occupational Health and Safety Management Plan
- Waste Management Plan
- Traffic and Transportation Management Plan
- Stakeholder Engagement Strategy
- Stakeholder Engagement Plan
- Worker Management and Recruitment Plan
- Community Health, Safety and Security Management Plan
- Community Development Framework Plan
- Outline Mine Closure Plan
- Change Management Procedure
- Contractors Management Plan
- Auditing and Continuous Improvement Plan

An Environmental and Social Commitments Register will be maintained by the operator. This register will document the environmental design and performance commitments made during the DFS and permitting process and be included in the international ESIA study.

21 CAPITAL AND OPERATING COSTS

21.1 PROJECT REQUIREMENTS

21.1.1 Introduction

The purpose of this CAPEX and OPEX estimate is to provide costs to an accuracy of +15 % −10 % that can be utilised for the economic analysis of the RVP.

21.1.2 Responsibilities

The project's CAPEX and OPEX estimate breakdown with associated responsibilities consists of the following:

- Mining costs estimated by DRA
- Process plant and on-site infrastructure costs estimated by SENET
- Dry tailings and mine waste facility development and closure costs estimated by KCB
- Other supporting infrastructure costs estimated by SENET
- Owner's pre-production costs estimated by SENET
- Environmental management: rehabilitation and closure costs estimated by KCB

21.1.3 Escalation

No escalation has been allowed for in the CAPEX and OPEX estimate. Capital costs are as of the first quarter (Q1) of 2021. The EPCM Contractor's rates reflect the rates expected in Q2 2021.

21.1.4 Exclusions

The following were not included in this CAPEX estimate:

- Financing costs
- Taxes and duties
- Permits
- Sunk costs
- Currency fluctuations
- Ongoing exploration costs
- ESM corporate costs

21.1.5 Exchange Rates

The costs of the project are reported in the United States dollar. The exchange rates used are shown in Table 21.1.

Table 21.1: Exchange Rates

Currency	Unit	Amount
Euro to United States Dollar	€/US$	0.85
South African Rand to United States Dollar	ZAR/US$	15.38
British Pound (GBP) to United States Dollar	£/US$	0.75

21.2 CAPITAL COSTS

21.2.1 Scope of the Estimate

The initial CAPEX estimate consists of direct and indirect costs, including Owner's costs and contingency costs, to be expended during the implementation phase, which shall extend from the approval by ESM of the NI 43-101 until the start of the commercial production.

The sustaining CAPEX estimate, which also consists of direct and indirect costs and Owner's costs and contingency costs, covers all the costs to be expended during the period starting at commercial production and extending until the end of the LOM.

The initial and sustaining CAPEX includes labour, permanent material and equipment, and subcontractors' costs required for the mine pre-production development, the processing facilities, the tailings storage facilities as well as all infrastructures, facilities, and utilities necessary to support the operation.

The initial CAPEX prepared for the NI 43-101 qualifies as a Class 3 estimate as per the Association for the Advancement of Cost Engineering (AACE); Recommended Practice 47R-11. The accuracy of the initial CAPEX estimate is assessed at ± 15 %. The sustaining CAPEX does not qualify as a Class 3 estimate; the accuracy of the sustaining CAPEX is assessed at ± 30 %.

21.2.2 CAPEX Summary

The total CAPEX for the RVP was estimated to be **US$446,976,422,** which includes project execution, EPCM, contingency and sustaining capital costs. The initial CAPEX and Total CAPEX by work breakdown structure (WBS) areas is summarised in Table 21.2. and Table 21.3

Table 21.2: Initial CAPEX Summary Per Area

WBS Area	Area Description	Initial CAPEX US$
1000	Mine	47,659,761
2000	Feed Preparation	12,369,609
3000	Process Plant	157,606,769
4000	Waste Management	28,294,405
5000	Plant Infrastructure	24,401,210
6000	Off-Site Infrastructure	21,420,000
7000	Indirect Costs	62,778,880
9000	Owner's Costs, Freight and Contingency	44,710,946
TOTAL		**399,241,580**

Table 21.3: Total CAPEX Summary Per Area

WBS Area	Area Description	Total CAPEX
		US$
1000	Mine	95,394,603
2000	Feed Preparation	12,369,609
3000	Process Plant	157,606,769
4000	Waste Management	28,294,405
5000	Plant Infrastructure	24,401,210
6000	Off-Site Infrastructure	21,420,000
7000	Indirect Costs	62,778,880
9000	Owner's Costs, Freight and Contingency	44,710,946
TOTAL		**446,976,422**

21.2.2.1 Summary of Initial CAPEX

The total initial CAPEX for the RVP was estimated to be **US$399,241,580**, which includes project execution, EPCM and contingency costs. The initial CAPEX is summarised in Table 21.4.

Table 21.4: Initial CAPEX Summary Per Discipline

Description	Initial CAPEX	Contingency	Total CAPEX
	US$	US$	US$
Earthworks	33,791,702	4,055,004	37,846,706
Civil Works	28,999,790	3,453,975	32,453,764
Infrastructure	3,002,717	327,741	3,330,458
Structural Steel	9,348,325	1,121,799	10,470,124
Plate Work	2,343,175	281,181	2,624,356
Machinery and Equipment	82,717,939	4,135,897	86,853,836
Piping	6,686,766	802,412	7,489,178
Valves	702,878	84,345	787,224
Electricals	15,922,310	1,113,092	17,035,402
Instrumentation	6,593,290	527,463	7,120,753
Transport	10,217,332	1,021,733	11,239,065
Electrical and Instrumentation (E&I) Installation	7,237,494	868,499	8,105,994
Structural, mechanical, plate work and piping (SMPP) Installation	36,442,035	4,373,044	40,815,079
Tools and Mobile Equipment	2,407,913	240,791	2,648,704
Total Direct Field Costs	**246,413,665**	**22,406,977**	**268,820,643**
Commissioning Spares	670,309	33,515	703,824
2-Year Operational Spares	5,016,807	331,109	5,347,917
Insurance and Critical Spares	5,166,176	335,920	5,502,095
Vendor Services	1,033,974	103,397	1,137,372

Description	Initial CAPEX	Contingency	Total CAPEX
	US$	US$	US$
First Fills	1,268,338	63,417	1,331,755
Total Indirect Field Costs	**13,155,604**	**867,359**	**14,022,963**
Total Field Cost	**259,569,269**	**23,274,336**	**282,843,605**
Project Management – EPCM	19,944,000	1,993,421	21,937,421
Owner's Costs	5,000,000	Included	5,000,000
Project Roads	8,500,000	Included	8,500,000
Power Supply to Site	15,885,000	Included	15,885,000
Mining Mobile Fleet – Predevelopment	32,566,633	Included	32,566,633
Waste Facility Design and Development	14,113,000	2,537,600	16,650,600
Water Management Design and Development	14,170,064	1,688,256	15,858,320
Total Support Costs	**110,178,697**	**6,219,277**	**116,397,974**
Total Development/Initial CAPEX	**369,747,966**	**29,493,613**	**399,241,580**

21.2.2.2 Summary of Sustaining CAPEX

The total sustaining CAPEX for the RVP was estimated to be **US$47,734,842,** which includes project execution, EPCM and contingency costs. The sustaining CAPEX is summarised in Table 21.5.

Table 21.5: Sustaining CAPEX Summary

Description	CAPEX	Contingency	Total CAPEX
	US$	US$	US$
Mining – Mobile Equipment	47,734,842	Included	47,734,842
Total Sustaining CAPEX	**47,734,842**	**Included**	**47,734,842**

21.2.3 Basis of Estimate, Assumptions and Exclusions

The CAPEX estimate for the RVP has been derived from information collated from the following technical design documents:

- LOM pit production schedule, including stockpiling operations
- LOM processing plan
- Mine haul road designs and layouts
- Mining equipment lists
- Process plant design criteria
- General layouts of the process plant and related infrastructure
- Waste facility and surface water management development schedule and operations
- Process flow diagrams
- Process plant equipment data sheets and lists
- Process plant piping and instrumentation diagrams (P&IDs)
- Process plant line, valve, and instrument lists

- Electrical single-line diagrams and motor lists
- Electrical reticulation routes
- Various discipline material take-offs (MTOs)
- Quotations from vendors on mechanical and/or process equipment
- Quotations from vendors on main construction contracts
- EPCM schedule
- In-house historical databases

The following assumptions were made in the preparation of this estimate:

- The LOM is 16 years.
- There will be a smooth transition between the various project implementation phases.
- Topography, Geotechnical and Materials:
  - The chosen site is suitable for the foundations; there are no specific problems due to excess precipitation or groundwater. Rock excavation will be required during excavation. A geotechnical survey was conducted, and site conditions were confirmed.
  - The piling allowance included in the estimate is based on preliminary geotechnical findings.
  - The earthworks bill of quantities (BOQ) is optimised in terms of cut and fill as a function of the topography. Further topography surveys will be performed during the early stages of the detailed engineering phase.
  - A source of aggregate, adequate for fill/backfill as well as for concrete mix, in sufficient quantity, will be within 2.5 km by road from the centre point of the process area.
  - Waste rock from the mine pit will be adequate for all fill requirements.
  - Excavated material will be non-acid generating.
  - The structural design will not be modified as a result of further topography studies.
  - The structural design will not be modified as a result of further geotechnical studies.

- Construction:
  - The construction work will be executed as a single EPCM contract.
  - The construction schedule for the process plant will run approximately 18 months excluding prior construction of the access roads and mains power supply.
  - There will be proper communication and cooperation by all construction contractors.
  - There will be no shortage of skilled trades workers throughout the entire construction phase, including the early works phase. Hence, there is no provision for salary increases potentially necessary to attract skilled trades workers.
  - Labour considers the remoteness of the RVP, i.e. 50 hours per week will be paid at regular time whilst the first 5 overtime hours per week will be paid at time and one half (× 1.5), and all overtime hours per week thereafter will be paid at double the base wage (× 2).

  - The Construction Contractors' facilities will be located within a maximum of 30 min walking distance from any working point for the whole duration of the RVP implementation.
  - The construction site will be accessible 24 h daily and 7 d weekly with adequate safety supervision.
  - There will be no work disruption resulting from inadequate accommodation and/or catering services.
  - The construction contracts will be of the unit-rate type, cost-plus type or lump-sum/turnkey type; the estimate does not allow for construction contracts of the time and material type.
  - Permanent administration offices will be made available in the early stages of the construction phase.
  - At least 60 % of the concrete will be precast (40 % of the concrete will be cast in place).
  - There will be a workweek of 7 d at 8 h/d (i.e. a single shift, daily, between 07h00 and 17h30, with an unpaid 30 min lunch break) and a rotation schedule of 14 d of work followed by 7 d of rest and relaxation (R&R).
  - Power from the Transelectrica grid will be made available no later than May 2023 so that the 18 months of construction and 3 months of commissioning will not require the use of temporary fuel-powered generators.
  - The inspection of equipment and material upon reception at site will reveal no deficiency; hence, there is no provision for rework or repair.
  - There will be no rework to field erected and installed equipment and material resulting from a quality assurance/quality control (QA/QC) inspection.

- Construction growths were applied to quantities originating from designed/neat quantities to account for the following, as appropriate:

  - Over-excavation (earthworks)
  - Compaction (earthworks)
  - Overlap (siding and roofing)
  - Overpour (concrete)
  - Connections (steel)
  - Routing clash (trays and conduits)

- Design and Measurement:

  - The mine open-pit access consists of only one ramp.
  - Transfer of tailings to the waste facility will be via conveyors, and transfer of waste from the open pits will be via haul trucks and conveyors.
  - Piping was measured from P&IDs and drawings.
  - Instruments were measured from P&IDs.

- The fuel cost is US$0.93/L.
- The power cost is US$0.0734/kWh (from Transelectrica).
- Transportation will be via commercial airlines, i.e. no chartered flights.

The following are excluded from this estimate:

- Taxes and duties (they form part of the financial model)

- Risk provision, including costs pertaining to mitigation plans
- Escalation beyond this NI 43-101 report base date
- Work stoppage resulting from labour dispute
- Work stoppage resulting from community relations dispute
- Any and all scope changes
- Any and all costs beyond commissioning completion
- Delays resulting from the following:
  - Permitting issue
  - Certificate issue
  - Project financing
  - Project approval
  - Agreement with claims owner

21.2.4 Mining CAPEX

The mining scope was derived from a detailed mining plan and based on an Owner operator model whereby the Owner will bear the total mining CAPEX.

In line with standard mining cost principles, the estimated base mining cost includes an incremental cost related to haulage distance to account for increased haulage costs as the depth of mining increases.

The major mining equipment for the RVP is leased equipment operated by the Owner and considers the following:

- Mine development (pre-stripping activities) costs: costs incurred during the pre-production period, including all the mining overhead charges and consumables costs, as well as the costs for ESM's mining personnel
- Mining mobilisation costs
- Mining installation costs (workshop, mine offices)

The mining CAPEX estimate includes an allowance for the following items:

- Topsoil removal
- Clearing and grubbing
- Drill and blast
- Ore mining and haulage to the ROM pad location
- Waste mining and haulage to the waste crusher tip and conveying to the dedicated waste facility for co-mingling with process plant dry tailings
- Haulage roads maintenance
- Mining personnel mobilisation and demobilisation
- Workshop installation and supply of tools and parts
- Procurement, operation, and maintenance of pit dewatering system
- Supply and operation of own equipment and maintenance facilities

The CAPEX for the following facilities as required for the mining operation is included in the following CAPEX sections:

- Maintenance workshops: Infrastructure CAPEX

- Offices: Infrastructure CAPEX
- Fuel depot and dispensing stations: Equipment CAPEX

The supply of explosives and their off-site storage facility costs are included in the OPEX, whereby the Service Supplier will be responsible for their own site establishment, storage, and delivery to site during operations.

The mining CAPEX is further categorised as follows:

- Mining initial CAPEX
- Mining sustaining CAPEX
- Mine closure CAPEX

21.2.4.1 Mining Initial CAPEX

The mining initial CAPEX includes contingency costs and is summarised in Table 21.6.

Table 21.6: Mining Initial CAPEX

Description	Total CAPEX
	US$
Mining – Major Equipment (Leasing Cost)	11,136,643
Mining – Support Equipment (Leasing Cost)	1,774,894
Mining Mobilisation	982,755
Mining Predevelopment (Pre-Strip)	16,706,832
Mining Haul Roads	1,965,510
Total Mining Initial CAPEX	**32,566,633**

The CAPEX associated with the mining operations was calculated as the cost of the first seven months of mining. The CAPEX consists of the Owner's mining site establishment, mining Phase 1 equipment lease capital, and the first seven months of predevelopment capital.

For purposes of the NI 43-101, the leased mining equipment option was selected as a financial basis for the initial and sustaining mining CAPEX (see Table 21.7).

Table 21.7: Mining CAPEX Comparison

Equipment Capital	Total CAPEX (US$)	Pre-Production CAPEX Years −1 and 0 (US$)	LOM Sustaining CAPEX (US$)
Mining Equipment Owning Capital			
Major Equipment	68,876,592	55,683,215	13,193,376
Support Equipment	8,874,470	8,874,470	–
Subtotal – Owned Mining Equipment	**77,751,061**	**64,557,685**	**13,193,376**
Mining Equipment Leasing Capital			
Major Equipment Leasing Cost	80,301,475	35,566,633	47,734,842
Support Equipment Leasing Cost	Included	Included	Included
Subtotal – Leased Mining Equipment	**80,301,475**	**35,566,633**	**47,734,842**

The mining equipment cash flow comparison between owning and leasing is shown in Figure 21.1

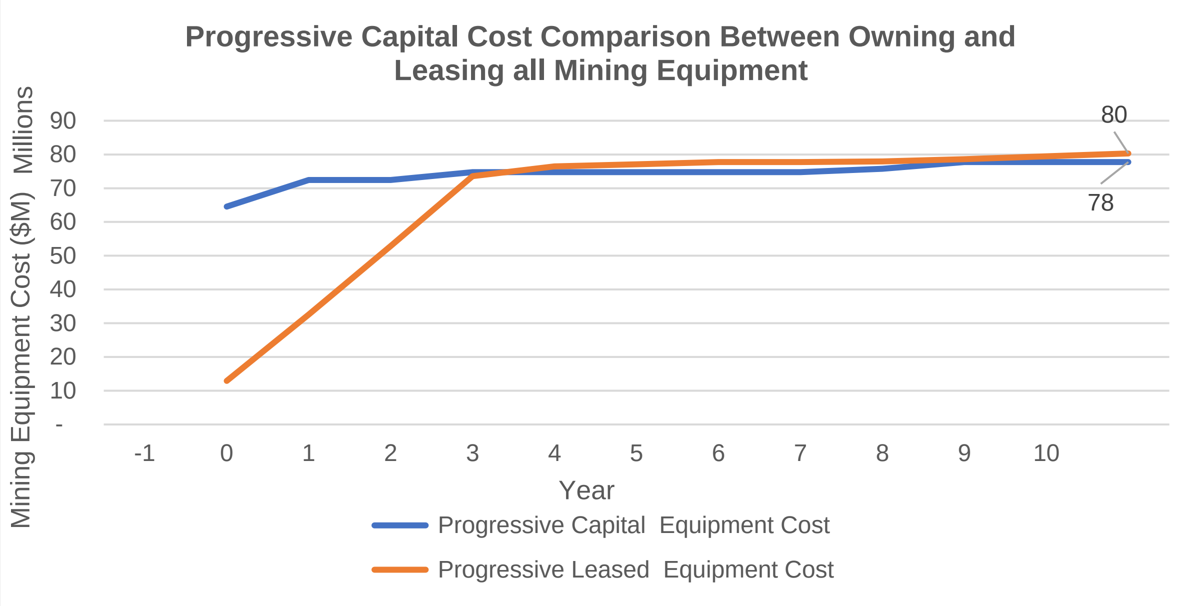


Figure 21.1: Mining Equipment Cash Flow Comparison between Owning and Leasing

21.2.4.2 Mining Sustaining CAPEX

The mining sustaining CAPEX includes contingency costs and is summarised in Table 21.8.

Table 21.8: Mining Sustaining CAPEX

Description	Total CAPEX
	US$
Mining – Major Equipment (Leasing Cost)	47,734,842
Mining – Support Equipment (Leasing Cost)	Included
Total Mining Sustaining CAPEX	**47,734,842**

21.2.4.3 Mine Closure CAPEX

The mine closure CAPEX includes contingency costs and is summarised in Table 21.9.

Table 21.9: Mine Closure CAPEX

Description	Total CAPEX
	US$
Salvage of Mining Equipment	−20,000,000
Mine-Wide Rehabilitation and Closure	20,000,000
Total Mine Closure Costs	**0**

The post-closure costs and passive care for the surface water management systems are included in the water management OPEX.

21.2.5 Process Plant, Mining and Waste Infrastructure CAPEX

The process plant, mining, and waste infrastructure CAPEX is summarised in Table 21.2.

The process plant and mining infrastructure CAPEX is subdivided into various categories:

- Main construction contracts, e.g. earthworks, civils, and SMPP and E&I
- Supply-only contracts, e.g. structural steel, plate work, machinery and equipment, piping, valves, instrumentation, and electrical equipment that will be procured by the EPCM Contractor on behalf of ESM and free-issued to the respective main contractors for erection/installation
- Supply and install contracts, e.g. prefabricated buildings

21.2.6 Main Contracts

21.2.6.1 Earthworks Contract

The process plant and peripheral infrastructure earthworks main contract scope was based on a detailed BOQ derived from earthwork terrace drawings and associated cut and fill long sections and quantities. The BOQ for the process plant earthworks was prepared in accordance with CESMM 4, “Civil Engineering Standard Method of Measurement”, Fourth Edition.

The earthworks enquiry was issued to suitable contractors in Romania and the surrounding countries and included preliminary and general (P&G) cost lists and contractual conditions against which tenderers quoted. Fully inclusive wet rates and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient. Security fencing for the on- and off-site infrastructure will be erected by the contractor.

The earthworks quantities are summarised in Table 21.10.

Table 21.10: Earthworks Quantities

Earthworks	Unit	Quantity
ROM Terrace Earthworks		
Clearing and Stripping of Site	m^2	39,690
Removal of Soil and Excavations	m^3	210,692
Fill and Soil Improvements	m^3	452,901
Fencing	m	336
Process Plant Terrace Earthworks		
Clearing and Stripping of Site	m^2	187,530
Removal of Soil and Excavations	m^3	661,916
Fill and Soil Improvements	m^3	523,839

Earthworks	Unit	Quantity
Fencing	m	18,447
MIA Terrace Earthworks		
Clearing and Stripping of Site	m^2	51,030
Removal of Soil and Excavations	m^3	445,043
Fill and Soil Improvements	m^3	400,968
Fencing	m	830
Wastewater Treatment Plant (WWTP) and Pump Station Terrace Earthworks		
Clearing and Stripping of Site	m^2	11,760
Removal of Soil and Excavations	m^3	63,092
Fill and Soil Improvements	m^3	65,856
Fencing	m	294
Overland Conveyors Earthworks		
Clearing and Stripping of Site	m^2	59,379
Removal of Soil and Excavations	m^3	342,266
Fill and Soil Improvements	m^3	26,082
Merișor Village Potable Water Tank Terrace Earthworks		
Clearing and Stripping of Site	m^2	8,190
Removal of Soil and Excavations	m^3	39,504
Fill and Soil Improvements	m^3	21,861
Fencing	m	126
Rovina Village Potable Water Tank Terrace Earthworks		
Clearing and Stripping of Site	m^2	9,450
Removal of Soil and Excavations	m^3	46,182
Fill and Soil Improvements	m^3	25,641
Fencing	m	126

21.2.6.2 Civil Works Contract

The process plant and peripheral infrastructure civil works main contract scope was based on a detailed BOQ derived from civil outline drawings, mechanical general arrangement drawings and the site block plan. The BOQ for the process plant civil works was prepared in accordance with CESMM 4.

The scope of work for the process plant on- and off-site infrastructure civil works includes the following:

- Reinforced concrete foundations for the support of mechanical equipment, structural steelwork, and plate work
- Reinforced concrete surface beds and bund walls with trenches and sumps to contain spillages within the terraced areas

- Site storm water drainage including a network of v-drains
- Reinforced concrete foundations and surface beds for the following:
    - Infrastructure prefabricated buildings
    - Infrastructure pre-engineered steel buildings (workshops and warehouses)
    - WWTP
    - Raw water pumping station
    - Containerised fire pump stations
    - Containerised potable water treatment plant
- Building works and architectural finishes for the brickwork portions contained within the workshops and warehouses
- Reinforced concrete foundations for the containerised motor MCCs, process plant control and server rooms
- Brickwork for transformer bay buildings
- Sewerage reticulation for the process plant and MIA, including all trenching as required for the electrical, potable water and fire water reticulation

All civil materials of construction are included in the Civil Contractor's scope of supply (cement, reinforcement, mesh, hold-down bolts, bricks, etc.).

The civil works enquiry was issued to suitable contractors currently based in Romania and the surrounding countries and included P&G cost lists and contractual conditions against which tenderers quoted. Fully inclusive wet rates and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient. The civil works quantities are summarised in Table 21.11.

Table 21.11: Civil Works Quantities

Civil Works	Unit	Quantity
Process Plant and On-Site Infrastructure Civil Works		
Excavation for Restricted Foundations	m^3	50,985
Import and Compact Backfill	m^3	27,886
Mass Concrete	m^3	473
30 MPa Concrete Structures	m^3	23,011
20 MPa Concrete and Blinding Layers	m^3	2,362
Reinforcement Steel	t	2,258
Mesh	m^2	87,226
Brickwork	m^2	1,357
Peripheral (MIA) and Off-Site Infrastructure Civil Works		
Excavation for Restricted Foundations	m^3	13,138
Import and Compact Backfill	m^3	5,896
Mass Concrete	m^3	285
30 MPa Concrete Structures	m^3	9,430
20 MPa Concrete and Blinding Layers	m^3	270

Civil Works	Unit	Quantity
Reinforcement Steel	t	1,005
Mesh	m^2	15,797
Brickwork	m^2	250

21.2.6.3 Structural, Mechanical, Platework and Piping Works Contract

The process plant and peripheral infrastructure SMPP works main contract scope was based on a detailed BOQ derived from general arrangement drawings, P&IDs and details of the free-issue mechanical equipment.

The SMPP works enquiry was issued to suitable contractors currently based in Romania and the surrounding countries. The enquiry included detailed P&G cost lists and contractual conditions against which tenderers quoted. Fully inclusive wet rates and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient.

The SMPP works quantities are summarised in Table 21.12

Table 21.12: SMPP Quantities

SMPP Works	Unit	Quantity
Process Plant Steelwork		
Light Steelwork – 0 to 25 kg/m	t	848
Medium Steelwork – 25 kg/m to 60 kg/m	t	1,260
Heavy Steelwork – above 60 kg/m	t	736
Circular Hollow Section Steelwork	t	94
Conveyor Steelwork	t	769
Process Plant and Conveyors Plate Work		
Plate Work – Chutes	t	320
Plate Work – Tanks and Bins	t	1,114
Plate Work – Flotation Cells	t	130
Process Plant and Conveyors Bought-Out Items		
Open Grid Flooring	m^2	4,490
Handrailing	m	6,462
Sheeting – Flat	m^2	21,069
Sheeting – Curved	m^2	2,250
Liners	t	80
Piping		
Overland Piping	m	12,614
Process Plant Rack Piping	m	6,473
All Areas Piping	m	19,636

SMPP Works	Unit	Quantity
Infrastructure Steel and Cladded Buildings		
Light Steelwork – 0 to 25 kg/m	t	170
Medium Steelwork – 25 kg/m to 60 kg/m	t	118
Heavy Steelwork – above 60 kg/m	t	80
Cold Rolled – Steelwork	t	20
Sheeting	m²	12,423

21.2.6.4 E&I Works Contract

The process plant and peripheral infrastructure E&I works main contract scope was based on a detailed BOQ derived from process plant and infrastructure layout drawings, process plant general arrangement drawings, electrical single-line diagrams, mechanical equipment lists and motor lists.

The E&I works enquiry was issued to suitable contractors familiar with construction projects in Romania and the surrounding countries. The enquiry included P&G cost lists and contractual conditions against which tenderers quoted. Fully inclusive wet rates and plant/labour histograms were received from the respective tenderers, and these were compared and adjudicated accordingly. The contractor will provide accommodation for their own workforce and will be completely self-sufficient.

All of the major contractors will be responsible for their own supply of construction power, as the main power switchyard will only be commissioned during the process plant and MIA commissioning phase.

21.2.7 Mechanical Equipment

The mechanical equipment quantities were derived from the equipment lists and process flowsheets. The mechanical scope of work for the project is to supply the equipment as detailed in the mechanical equipment list, mechanical data sheets and mechanical drawings.

Enquiries were prepared, inclusive of equipment data sheets, and were then sent to equipment vendors/suppliers. The quotations received were commercially and technically adjudicated.

21.2.8 Structural Steel

Quantities were established based on MTOs derived from plant general arrangement drawings produced during the DFS. Unit rates for supply and fabrication were obtained from fabricators and were applied to the MTOs.

The respective rates were applied to the bill of materials for the following equipment:

- Structural steelwork
- Plate work
- Liners
- Grating and flooring

- Handrailing
- Sheeting

21.2.9 Plate Work

The plate work scope of work entails shop detailing, supply, manufacturing, inspection, and corrosion protection of plate work in the form of tanks as per the tank schedule. The unit rates for supply and fabrication were obtained from fabricators and were applied to the tank schedule to estimate the plate work cost.

21.2.10 Piping and Valves

The piping and valves scope of work entails the supply, manufacturing, inspection, and corrosion protection of the piping and valves as per the BOQs. The unit rates for supply and fabrication were obtained from fabricators and were applied to estimate the piping and valves supply costs.

21.2.11 Infrastructure and Buildings

The CAPEX of the infrastructure and buildings includes the following:

- Prefabricated infrastructure buildings
- Pre-engineered steel and cladded infrastructure buildings
- Furniture

21.2.11.1 Prefabricated Infrastructure Buildings

The following infrastructure buildings will be supplied as prefabricated buildings:

- On-Site Process Plant Infrastructure:
  - Change house building
  - Office building
  - Gatehouse building
  - Weighbridge control room
  - Process plant control room
  - Metallurgical and assay laboratory
- Off-Site Infrastructure:
  - Change house building (MIA)
  - Office buildings (MIA)
  - Gatehouse building (MIA)
  - Fuel dispensing control room (MIA)
  - Ablution (ROM Pad)
  - Ablution (WWTP)

The prefabricated building package includes the supply and site installation of the building, including all internal electrical reticulation and fittings, all internal water reticulation, plumbing, sanitary fittings, extraction, and air conditioning.

The Prefabricated Building Contractor has allowed for the relevant P&G costs for the installation of the buildings. The contractor will provide accommodation for their own workforce and will be completely self-sufficient.

A supply and installation enquiry package was issued to the market for prefabricated buildings. The vendor was selected based on both technical and commercial adjudications.

The CAPEX for the site clearance and fencing for the infrastructure buildings is included in the earthworks contract.

The CAPEX for the concrete foundations, floor slabs, sewer reticulation and drainage systems is included in the civil works contract.

The CAPEX for the electrical switchgear and MCCs is included in the electrical CAPEX.

The electrical supply reticulation to the respective infrastructure buildings is included in the Electrical, Control and Instrumentation (E, C&I) contract.

21.2.11.2 Pre-Engineered Steel and Cladded Infrastructure Buildings

The following pre-engineered steel and cladded infrastructure buildings will be supplied:

- On-Site Process Plant Infrastructure:
  - Workshop
  - Warehouse
  - Reagents Store
- Off-Site Infrastructure:
  - Workshops (MIA)
  - Warehouse (MIA)

The pre-engineered steel building package includes the design, fabrication, and supply of the steel buildings.

A design, fabrication and supply enquiry package was issued to the market for pre-engineered steel buildings. The vendor was selected based on both technical and commercial adjudications.

The CAPEX for the site clearance and fencing for the infrastructure buildings is included in the earthworks contract.

The CAPEX for the concrete foundations, floor slabs, sewer reticulation and drainage systems is included in the civil works contract.

The site installation of the steel and cladded buildings is included in the SMPP CAPEX.

The CAPEX for the electrical switchgear and MCCs and all internal electrical reticulation, lighting and fittings is included in the electrical CAPEX.

The electrical supply reticulation to the respective infrastructure buildings is included in the E, C&I contract.

21.2.11.3 Furniture

A supply enquiry package was issued to the market for furniture for both the on- and off-site infrastructure buildings and includes desks, chairs, cupboards, store shelving and the like. The vendor was selected based on commercial adjudication.

The costs for the furniture are included in the infrastructure CAPEX.

21.2.12 Electrical Equipment

The process plant, mining and peripheral infrastructure electrical equipment detailed BOQ was derived from layout drawings, process plant general arrangement drawings, electrical single-line diagrams, mechanical equipment lists and motor lists.

The electrical equipment includes the following:

- MV switchgear
- Step-down transformers
- MCCs
- LV and MV cables
- Cable racking, luminaires, and earthing
- Power factor correction units
- Backup power system

Electrical equipment supply enquiries were prepared, inclusive of equipment data sheets, and sent to approved equipment vendors/suppliers. The quotations received were commercially and technically adjudicated.

21.2.13 Piping and Valves

The pipe MTOs were based on pipe sizes derived from the P&IDs and line transpositions that were done using the plant layout to route the lines. MTOs for all carbon steel, stainless steel, and high-density polyethylene (HDPE) piping were forwarded to vendors for pricing. The lengths of overland piping were determined using the overall site layout and geography plans of the area.

A valve list compiled from the P&IDs, detailing each manual valve type, was submitted to vendors for pricing. The actuated valve costs were included in the instrumentation cost estimate. The quotations received were technically and commercially adjudicated, and preferred vendors were selected. The scope of work relating to piping and valves is indicated in the P&IDs.

21.2.14 Process Control (Control and Instrumentation (C&I))

The process plant includes the implementation of a process automation system (PAS). The PAS comprises a SCADA system, PLCs, and instrumentation. The SCADA and PLC equipment will be located in the plant control room and the equipment room located adjacent to the plant control room.

A security system, including CCTV cameras and access control to site, will also be provided.

The communications system consisting of an office local area network (LAN), satellite link for Internet and email, telephone, and radio systems will be provided by ESM.

The PLC and SCADA costs were based on a typical plant configuration with full plant control from a central control room. Provision was also made for a sequel server for constant data logging and trending.

Instrumentation costs were based on instrument and valve lists. The instrumentation BOQ was developed from data derived from the P&IDs, as well as the instrument list and the instrumentation drawings.

Dedicated remote input/output (I/O) panels, located in the specific plant areas, are utilised to connect the field instruments to the PLC. Digital instruments are wired to the remote I/O panel via multipair cables. Analogue instruments are connected to the remote I/O panels via a Profibus PA network.

21.2.15 Engineering, Procurement and Construction Management (EPCM)

Engineering, project management and drawing office man-hours are based on the estimated number of man-hours required to complete the detailed design of the project. Unit rates for man-hours represent actual rates currently being charged on similar projects.

Site construction management is based on a highly skilled team of engineers and site staff who will supervise the construction crew's activities. This part of the estimate assumes that construction will be subcontracted to earthworks, civil works, SMPP and E&I construction companies. This, however, requires a higher level of supervision on the part of the EPCM Contractor and Owner's representative.

21.2.16 First Fills

The first-fill costs for the SAG, ball and regrind mills were developed from first principles. These were defined as those costs incurred prior to commissioning in preparing the circuit to accept ore. These costs included the addition of steel balls (various sizes) to the SAG and ball mills and ceramic media to the regrind mill to design charge levels.

The cost for the first fills as required for the balance of the mechanical equipment are included in the respective machinery and equipment CAPEX.

The first-fill cost for the reagents and diesel for the mining fleet are included in the first three months' OPEX allowance.

The mills first-fill costs for the project are summarised in Table 21.13.

Table 21.13: Mills First-Fill Cost Summary

Description	CAPEX (US$)	Contingency (US$)	Total CAPEX (US$)
SAG Mill (Forged Steel)	308,111	15,405	323,516
Ball Mill (Cast Steel)	920,115	46,006	966,121
Regrind Mill (Ceramic Media)	40,112	2,006	42,118
First Fill Totals	**1,268,338**	**63,417**	**1,331,755**

21.2.17 Contingency

An average contingency of 8 % (see Table 21.2) has been allowed for to cover items that are included in the scope of work, but that cannot be adequately defined at this stage due to the level of engineering conducted during the DFS and a subsequent lack of accurate detailed design and procurement information.

The average contingency of 8 % was derived mathematically and is affected by the design and procurement confidence contingency values that were attributed to each of the respective capital cost categories, i.e.

- Earth and Civil Work: Estimated – assigned 12 %
- Structural Steel and Plate Work: Estimated – assigned 12 %
- Mechanical Equipment: Fixed and firm quotations – assigned 5 %
- Piping and Valves: Estimated – assigned 12 %
- Electrical and Instrumentation: Fixed and firm quotations – assigned 8 %
- First Fills: Fixed and firm quotations – assigned 5 %
- E, C& I and SMPP Installation: Estimated – assigned 12 %
- Commissioning Spares: Fixed and firm quotations – assigned 5 %
- Two-Year and Strategic Spares: Fixed and firm quotations – assigned 7 %
- The balance of the cost categories: Assigned 10 % based on level of engineering

21.2.18 Vendor Services

The cost for vendor services includes all the items where the presence of the vendor is required during the construction phase in order for guarantees to be honoured. It also includes items where construction supervision is required, particularly for the installation of the large and/or critical equipment items. The costs are based on actual quotes obtained from the respective vendors.

21.2.19 Freight

The freight costs for the project are based on a percentage of the various discipline costs as summarised in Table 21.14.

Table 21.14: Freight Cost Summary

Description	CAPEX (US$)	Contingency (US$)	Total CAPEX (US$)
Structural Steelwork	747,866	74,787	822,653
Plate Work (Tanks, Chutes, Bins, etc.)	187,454	18,745	206,199
Machinery and Equipment	6,617,435	661,744	7,279,179
Piping	534,941	53,494	588,435
Valves	56,230	5,623	61,853
Electrical	1,113,092	111,309	1,224,401
Instrumentation	527,463	52,746	580,210
Infrastructure Buildings – MIA	175,048	17,505	192,553
Infrastructure Buildings – Process Plant	65,169	6,517	71,686
Freight Totals	**10,217,332**	**1,021,733**	**11,239,065**

21.2.20 Power Supply and Infrastructure

The 110/20 kV switchyard and underground transmission power line were estimated by ESM through the employment of a local Romanian contractor based on their experience in electric power supply systems and, in particular, experience with the connection conditions of the Romanian power grid code.

The required infrastructure for connection to the Romanian power grid will be constructed under a full turnkey solution including the furnishing of all labour, material and services for the design, supply, manufacture, erection, cold and hot commissioning, performance testing, and operation handover to the Romanian grid operator.

Site preparation of the required terrace area for the 110/20 kV switchyard to be built by others is accounted for in the earthworks CAPEX.

21.2.21 Fuel Storage and Dispensing Depot

The diesel fuel storage facilities will hold 142,000 L total capacity.

A supply enquiry package was issued to the market for the fuel storage tanks, interconnecting pipework and fuel dispensing equipment. The vendor was selected based on both technical and commercial adjudications.

The CAPEX for the site clearance and fencing for the terrace for the diesel fuel farm and mining fleet refuelling station is included in the earthworks contract.

The CAPEX for the concrete foundations, floor slabs, catch pits and drainage systems is included in the civil works contract.

The CAPEX for the dispensing station control room prefabricated building, furniture, fixtures, internal electrical reticulation and electrical fittings, and the necessary fire protection foam systems is included in the respective supply contracts.

The CAPEX for the electrical switchgear and MCCs is included in the electrical CAPEX.

The electrical supply reticulation to the fuel depot is included in the E, C&I contract.

Erection of the fuel storage tanks and dispensing systems, including the raw and potable water supply reticulation to the fuel depot, is included in the SMPP contract.

21.2.22 Spares

Spares were allowed for in the CAPEX. The spares costs were obtained from vendor quotations. Three categories of spares were considered and included in the initial CAPEX:

- Commissioning spares
- Two-year operating spares
- Insurance and critical spares

21.2.23 Insurances

The EPCM Contractor will be responsible for the necessary insurance related to workmen's compensation for their supervisory personnel on site.

The EPCM Contractor's subcontractors for the SMPP installation, electrical services installation, and process control installation will be responsible for the workmen's compensation insurance cover for their personnel in their respective engagements.

All risks insurance cover for materials and equipment on site during the execution phase is included in the CAPEX.

Third-party insurance and maintenance of vehicles supplied for use by the EPCM Contractor form part of the Owner's costs.

Professional Indemnity (PI) Insurance cost allowances as may be required are part of the Owner's costs.

21.2.24 Waste Management Facility and Water Management Costs

The waste management facility and water management CAPEX estimates include an initial design and construction development allowance for the following:

- Cleaner tailings area
- Co-mingled rougher tailings and waste rock area
- Starter buttress as required for waste management
- Catchment dams, ponds, diversion tunnels and channels including portals as required for surface water management

These developments will include the following:

- Clearing and grubbing
- Topsoil removal and stockpiling
- Common excavation and footprint foundation preparation
- Rock excavation and blasting
- Filling and compaction including underdrain rockfill, filter gravel, filter sand and liner beddings
- Trimming of embankment surfaces
- Supply and installation of geosynthetic and clay liners
- Drainage for water and road crossings

The CAPEX associated with the development of the waste management facility and water management has been determined based on a set of MTOs developed for the WMF and SWM features and a set of unit rates based on similar project experience, client references and local Romanian vendors for the earthworks and liner requirements. Water management tunnelling requirements have been determined based on quantities measured by DRA Mining and rates applied from an in-house database.

The estimated CAPEX associated with the development of the waste management facility and water management is summarised in Table 21.15.

Table 21.15: Waste Management Facility and Water Management CAPEX

Description	CAPEX	Contingency	Total CAPEX
	US$	US$	US$
Waste Facilities Development	12,825,000	2,280,000	15,105,000
Tailings Handling Development	1,288,000	257,600	1,545,600
Waste Management Facility CAPEX	**14,113,000**	**2,537,600**	**16,650,600**
Surface Water Management and Catchment Dam Development	3,487,500	620,000	4,107,500
Surface Water Management – Tunnels and Portals Development	10,682,564	1,068,256	11,750,820
Surface Water Management CAPEX	**14,170,064**	**1,688,256**	**15,858,320**
Total Waste and Water Management CAPEX	**28,283,064**	**4,225,856**	**32,508,920**

21.2.25 Other Supporting Infrastructure and Equipment Costs

The proposed infrastructure will support the mining and plant operations and includes the following:

- Raw water management and supply
- Pit dewatering
- Access and haul roads
- Plant and operational support vehicles

21.2.25.1 Raw Water Management and Supply

The raw water management and supply philosophy was derived from the process plant and mining operational requirements as detailed in the process plant and mining water balances, respectively.

Raw water will be supplied to the plant and mining operations from the main surface water catchment dam, which is connected and replenished via tunnels and channels to outlying contact water collection ponds.

Contact water from the main surface water catchment dam that is used for raw water replenishment purposes and/or released into the environment will first be treated at a 300 m^3/h WWTP located alongside the main surface water catchment dam.

The pumping and piping BOQs associated with these systems were compiled by measuring the relevant piping routes on the overall site plot plan, in conjunction with the project P&IDs. These BOQs (along with the associated pump systems and WWTP) were subsequently issued to the respective vendors via a formal enquiry to obtain rates for the fabrication and supply of the materials/equipment. The CAPEX for the raw water pumping, WWTP and piping is included in the process plant mechanical equipment, piping, and valves CAPEX.

Installation/erection for the raw water supply pumping, piping and WWTP is included in the SMPP contract.

The CAPEX for the development of the main surface water catchment dam and peripheral infrastructure is included in the surface water management CAPEX.

The CAPEX for the site clearance and fencing for the raw water pumping station and WWTP is included in the earthworks contract.

The CAPEX for the concrete foundations, floor slabs, catch pits, sewer reticulation and drainage systems is included in the civil works contract.

The CAPEX for the ablution facility prefabricated building, fixtures, internal sewer reticulation and fittings is included in the infrastructure CAPEX.

The CAPEX for the electrical switchgear and MCCs is included in the electrical CAPEX.

The electrical supply reticulation to the raw water pumping station and WWTP is included in the E, C&I contract.

21.2.25.2 Pit Dewatering

The pumping requirements to dewater the pits from rainfall and groundwater inflows were estimated following completion of the mine and geohydrology designs. The pit dewatering systems from the Colnic and Rovina pits will feature dedicated dewatering transfer lines to the nearest contact water diversion channels feeding into the main surface water catchment dam.

The pit dewatering mechanical equipment and all the associated piping BOQs were issued to the relevant vendors to obtain quotations for the manufacture and supply of each package and these costs are included in the mining initial CAPEX for the Colnic pit dewatering equipment and mining sustaining CAPEX for the Rovina pit dewatering equipment.

The CAPEX for the development of the contact water diversion channels and tunnels, including the main surface water catchment dam, is included in the waste management facility CAPEX.

The CAPEX for the electrical switchgear and MCCs is included in the electrical CAPEX.

The electrical supply reticulation to the pit dewatering stations is included in the E, C&I contract.

21.2.25.3 Access and Haul Roads

The BOQs for the main Rovina site access road, the Merișor village access road, the waste management facility perimeter road, and the mine haul roads have been prepared using the topographical drawings, plant location, waste management facility location and mine planning designs as a basis for measurement.

Tender documents were drawn up for the design, procurement and construction management of the access roads and sent to suitable Romanian contractors for pricing. The quotations received were commercially and technically adjudicated. The development costs for the access roads are included in the roads CAPEX.

Haul road development costs were derived from in-house database pricing and are included in the mining initial CAPEX.

The estimated CAPEX associated with the development of the access and haul roads is summarised in Table 21.16.

Table 21.16: Access and Haul Roads CAPEX

Description	CAPEX	Contingency	Total CAPEX
	US$	US$	US$
Main Rovina Site Access Road (3,880 m)	2,965,000	Included	2,965,000
Merișor Village Access Road (2,880 m)	2,285,000	Included	2,285,000
Waste Management Facility Perimeter Road (6,500 m)	3,250,000	Included	3,250,000
Access Roads CAPEX	**8,500,000**	**Included**	**8,500,000**
Mining Haul Roads	1,965,510	Included	1,965,510
Mining Haul Roads CAPEX	**1,965,510**	**Included**	**1,965,510**
Access and Haul Roads Total Capital	**10,465,510**	**Included**	**10,465,510**

21.2.25.4 Plant Support and Operational Vehicles

The CAPEX for the process plant workshop tools, light support vehicles and the plant operational vehicles is based on the selected vendor's quotation, which was commercially and technically adjudicated, and is included in the tools and mobile equipment CAPEX.

The process plant light support and operational vehicles are summarised in Table 21.17.

Table 21.17: Plant Support and Operational Vehicles

Description	Quantity
Light Plant Vehicles	12
Mobile Crane 80 t	1
Mobile Crane 20 t	2
Forklift	3
Skid-steer Loader	3
Front-End Loader	2
Tractor and Trailor	2
Lorry 10 t	2

21.2.26 Owner's Pre-Production Costs

The Owner's pre-production costs are based on costs that will be incurred from the start of the project implementation phase up to the commissioning and handover to plant operations.

The Owner's pre-production costs comprise the following:

- General and administration salaries, including the Owner's project team; the health, safety and environmental (HSE) department; finance; procurement; and human resources
- Mining department labour costs prior to commencement of pre-stripping

- Plant and laboratory labour costs prior to commencement of plant commissioning
- Costs associated with the administration of an off-site office
- Training package implementation and contractor engagement
- Vehicle running, insurance and maintenance costs
- Other administrative support costs
- Insurance costs
- Mine licence costs and reclamation bonds

21.2.26.1 Pre-Production Labour

The pre-production labour cost for the 18-month construction period includes the following:

- Pre-production labour salaries
- Dedicated vehicle costs (diesel and maintenance)
- Recruitment costs

21.2.26.2 Other Pre-Production Costs

The other pre-production costs for the 18-month construction period include the following, which were based on the general and administration costs:

- Facilities maintenance
- Off-site offices and travel costs
- Supplies and spare parts
- Security
- Other administration costs
- Environmental and social costs

21.2.27 Working Capital

The working CAPEX was defined as those fixed and variable costs incurred by the mine from commissioning to the point where the mine is cash flow positive and the revenue from concentrate sales can pay for the mine's operational costs.

The working CAPEX has been calculated from first principles, estimating the ramp-up period (period for plant to reach design production capacity) of two months. In this calculation, the following costs were considered:

- Operating costs for the whole operation, i.e. mining, process plant, and waste management facility
- General and administration costs
- Mining and process plant assay costs
- Stockholding costs

The working CAPEX value is detailed in Section 22.

21.3 OPERATING COSTS

21.3.1 Summary of OPEX

The purpose of this OPEX estimate is to provide operating costs, and the associated general and administrative (G&A) costs, to an accuracy of +15 % −10 % that can be utilised for the economic analysis of the RVP.

The project's annual OPEX estimate for the LOM consists of the following:

- Mining OPEX estimated by DRA
- Process plant OPEX estimated by SENET
- WMF and SWM OPEX estimated by KCB
- Site G&A OPEX estimated by ESM

The overall LOM OPEX for the RVP is summarised in Table 21.18 with the cost distribution shown in Figure 21.2.

Table 21.18: Overall RVP LOM OPEX Summary

Description	Cost		Cost Distribution
	US$/t Feed	US$ Million	%
Mining	6.23	743.8	44
Process Plant, Assay	7.34	876.1	51
WMF and SWM	0.44	52.2	3
G&A	0.29	34.6	2.0
TOTAL	**14.30**	1,706.7	**100**

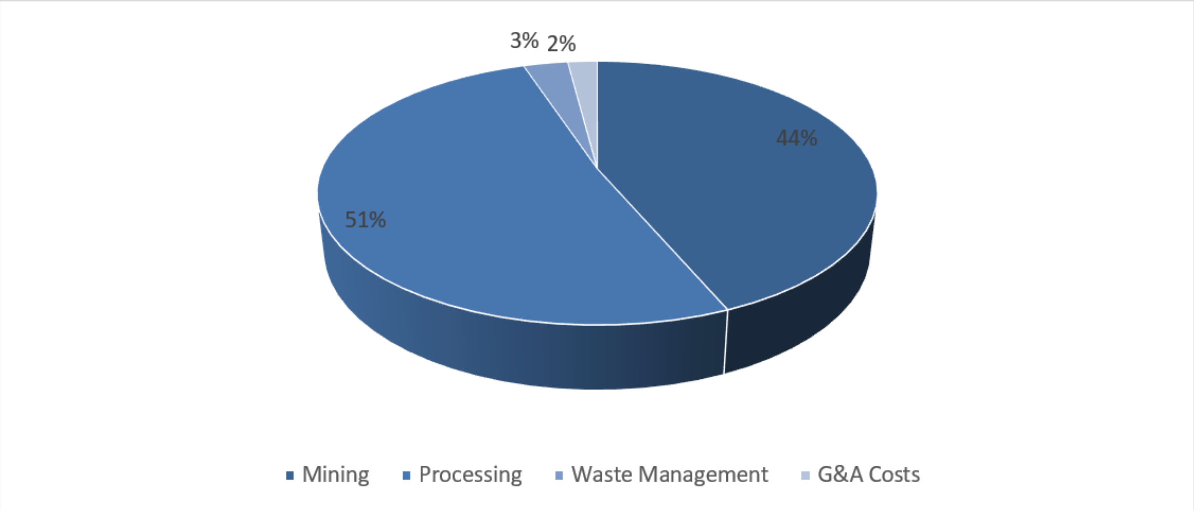


Figure 21.2: OPEX Distribution

The annual process plant OPEX for the LOM for the project is summarised in Table 21.19. Details of the OPEX are shown in the sections that follow.

Royalties have not been included in these tables and are addressed in Section 22 (Economic Analysis).

Table 21.19: LOM OPEX Summary

Description	Unit	LOM	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033	FY 2034
			Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Ore Processed												
Tonnage Processed	t	119,373,886	2,440,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000	7,200,000
Au Feed Grade												
Au Grade – Average	g/t	0.44	0.71	0.72	0.64	0.81	0.63	0.60	0.54	0.50	0.52	0.36
Au Recovered												
Recovered Au – Total	oz	1,348,771	44,642	133,810	117,720	149,464	117,083	110,395	99,855	92,949	95,566	65,335
Cu Feed Grade												
Cu Grade – Average	%	0.1695	0.14	0.11	0.10	0.13	0.09	0.10	0.12	0.10	0.07	0.32
Cu Recovered												
Recovered Cu – Total	lb	407,588,786	7,036,142	15,250,805	14,720,235	18,412,228	12,185,281	13,966,957	17,565,979	14,487,691	9,824,193	46,900,069
OPEX												
Mining	US$	743,764,850	47,827,119	53,154,592	53,068,245	52,258,419	50,591,970	50,906,726	45,582,592	34,113,072	25,691,063	47,827,119
Process Plant, Assay	US$	876,064,629	21,390,119	52,383,570	52,383,570	52,383,570	52,383,570	52,383,570	52,383,570	52,383,570	52,383,570	21,390,119
Waste Management	US$	52,245,000	3,789,000	3,690,500	4,543,000	4,135,000	2,586,500	2,278,500	3,306,500	3,841,000	2,626,000	3,789,000
G&A	US$	34,624,234	1,077,906	2,059,966	2,059,966	2,059,966	2,059,966	2,059,966	2,059,966	2,059,966	2,059,966	1,077,906
Total	**US$**	**1,706,698,714**	**74,084,144**	**111,288,628**	**112,054,781**	**110,836,955**	**107,622,006**	**107,628,762**	**103,332,629**	**92,397,608**	**82,760,599**	**106,917,061**
Total OPEX (per tonne feed processed)	US$/t feed	14.30	30.36	15.46	15.56	15.39	14.95	14.95	14.35	12.83	11.49	14.85
Total OPEX (per Au equivalent ounce produced)	US$/oz Au eq.	769.98	1,242.56	669.29	751.74	587.48	752.47	768.06	752.86	746.38	710.50	647.25
Total OPEX (per Cu equivalent pound produced)	US$/lb Cu Eq.	1.64	2.65	1.42	1.60	1.25	1.60	1.64	1.60	1.59	1.51	1.38

21.3.2 Mining OPEX

The basis of the Definitive Feasibility Study (“DFS”) mining cost estimate is a comprehensive RFBP process that was conducted involving most of the major earthmoving OEMs. The final CAPEX and OPEX reflect the equipment selected, which was optimal in terms of costs, size, flexibility, operability, and suitability to handle the expected conditions that are anticipated in the two planned open-pit mining operations that currently make up the RVP. All price estimates were obtained between September 2021 and January 2021. These DFS OPEX cost were based on mine scheduling of the Colnic and Rovina pits and their associated infrastructure with their planned waste storage facility locations. To accurately project the future OPEX, the monthly haul distance for the production faces to the materials respective tipping points was calculated on both a monthly and annualised basis so that the total production fleet’s operating hours could be accurately estimated. The average operating unit costs per mining function were calculated by applying rates to each year’s materials BOQ for the full LOM and then determining what the average unit rates over the entire LOM would be by taking all the mining-related costs into consideration.

The mining cost estimate is based on a power cost of US$0.0734/kWh and diesel cost of US$0.93/L.

21.3.2.1 Mining Labour

The estimated maximum annual mining operating and maintenance labour cost was estimated at US$5,614,900. The cost was derived from first principles where the actual labour complement required to meet the production requirements of all mining areas and resultant maintenance function was identified, and the required number of personnel and their levels were established. The complement derived was then benchmarked against other operations of similar size and complexity. The maximum mining operations labour cost portion of US$3,214,600 is accounted for in the equipment OPEX in the financial modelling.

The operating and maintenance labour cost was broken down as follows:

- Senior Personnel (office based):
    - Basic remuneration
    - Yearly bonus
    - Fringe benefits
- Operations Personnel (on shift):
    - Basic remuneration
    - Fringe benefits
    - Overtime

The labour schedule and renumeration rates were developed by ESM in consultation with SAMAX and input from SENET/DRA.

The categories selected for the various positions are described in Table 21.20.

Table 21.20: Mining Labour Categories

Employee Category	Schedule
Staff	Office
Hourly	Shift
Operations	Shift

Table 21.21 shows the labour cost summary. The detailed breakdown is given in Table 21.22.

Table 21.21: Mining Labour Cost Summary

Mining Position Description	Total Number of Employees	Total Cost	Total Cost
		US$/a	US$/t Mill Feed
Subtotal Mining Management	44	1,301,100	0.18
Subtotal Mining Maintenance	32	1,099,200	0.15
Subtotal Mining Operation*	160	3,214,600	0.45
Total Cost (Steady State)	**236**	**5,614,900**	**0.78**
* Mining operators' costs are included in the equipment costs.			

Table 21.22: Detailed Maximum Mining Labour Cost Breakdown

Position Description	Employee	Total Employees	Rate/Employee	Total Cost/Employee	Total Cost
	Category	Number	Basic US$	US$ Benefits Included	US$/a
Mine Maintenance					
Maintenance General Foreman	Staff	1	29,670	43,000	43,000
Maintenance Shift Foremen	Staff	4	21,735	31,500	126,000
Maintenance Planner/Contract Administrator	Staff	**2**	16,698	24,200	48,400
Clerk/Secretary	Staff	1	9,039	13,100	13,100
Subtotal		**8**			**230,500**
Mine Operations					
Mine Operations/Technical Superintendent	Staff	1	67,896	98,400	98,400
Mine Operations General Foreman	Staff	1	30,429	44,100	44,100
Mine Shift Foreman	Staff	4	21,735	31,500	126,000
Junior Shift Foreman	Staff	4	19,596	28,400	113,600
Trainer	Staff	1	20,286	29,400	29,400
Road Crew/Services Foreman	Staff	1	21,735	31,500	31,500
Clerk/Secretary		**1**	9,039	13,100	13,100
Subtotal		**13**			**456,100**
Mine Engineering					
Chief Engineer	Staff	1	27,531	39,900	39,900
Senior Engineer	Staff	1	21,735	31,500	31,500
Open-Pit Planning Engineer	Staff	2	20,286	29,400	58,800
Geotechnical Engineer	Staff	1	20,286	29,400	29,400
Blasting Engineer	Staff	1	20,286	29,400	29,400
Blasting/Geotechnical Technician	Staff	1	17,388	25,200	25,200
Surveyor/Mining Technician	Staff	4	16,008	23,200	92,800

Position Description	Employee Category	Total Employees Number	Rate/Employee Basic US$	Total Cost/Employee US$ Benefits Included	Total Cost US$/a
Surveyor/Mine Technician Helper	Staff	4	15,111	21,900	87,600
Subtotal	**Staff**	**15**			**394,600**
Geology					
Chief Geologist	Staff	**1**	25,392	36,800	36,800
Senior Geologist	Staff	1	21,735	31,500	31,500
Grade Control Geologist/Modeller	Staff	2	20,286	29,400	58,800
Sampling/Geology Technician	Staff	4	16,008	23,200	92,800
Subtotal	**Staff**	**8**			**219,900**
Total Mine Staff	**Staff**	**44**			**1,301,100**
Mine Operations					
Driller	Operations	12	13,731	19,900	238,800
Blaster	Operations	2	13,731	19,900	39,800
Blaster’s Helper	Operations	Contractor	12,075	17,500	
Loader Operator	Operations	8	14,628	21,200	169,600
Hydraulic Shovel Operator	Operations	12	14,628	21,200	254,400
Haul Truck Driver	Operations	60	13,731	19,900	1,194,000
Conveyor Operator	Operations	28	13,731	19,900	557,200
Dozer Operator	Operations	16	13,869	20,100	321,600
Tyre Dozer Operator	Operations	4	13,869	20,100	80,400
Grader Operator	Operations	3	13,869	20,100	60,300
Water Truck Driver	Operations	8	13,731	19,900	159,200
Compactor Driver	Operations	6	13,731	19,900	119,400
Utility Excavator Operator	Operations	1	13,731	19,900	19,900
Total Operation Staff	**Operations**	**160**			**3,214,600**

Position Description	Employee	Total Employees	Rate/Employee	Total Cost/Employee	Total Cost
	Category	Number	Basic US$	US$ Benefits Included	US$/a
Mine General					
Road/Pump Crew	Hourly	2	12,765	18,500	37,000
General Mine Labourer	Hourly	8	12,075	17,500	140,000
Trainee	Hourly	4	10,833	15,700	62,800
Light-Duty Mechanic	Hourly	2	14,628	21,200	42,400
Tyre Man	Hourly	4	14,628	21,200	84,800
Service Truck Driver	Hourly	4	12,075	17,500	70,000
Subtotal	**Hourly**	**24**			**437,000**
Mine Maintenance					
Heavy-Duty Mechanic	Hourly	11	14,628	21,200	233,200
Welder	Hourly	11	14,628	21,200	233,200
Electrician	Hourly	4	14,628	21,200	84,800
Apprentice	Hourly	6	12,765	18,500	111,000
Subtotal	Hourly	**32**			**662,200**
Total Mine Staff	**Hourly**	**56**			**1,218,600**

21.3.2.2 Mining Production and Support Equipment

The annual production and support equipment OPEX varies over the FVP LOM depending on total material movement, strip ratio and waste storage facility construction location. The OPEX was determined based on original equipment manufacturers unit OPEX, utilisation, and fuel costs. The fuel cost was obtained from ESM, and the supply cost was obtained from major fuel suppliers. The detailed breakdown is given in Table 21.23.

Table 21.23: Mining Production and Support Equipment OPEX

Description	Unit	Average Diesel Consumption	Tyres/ Tracks	Repairs/ Maintenance	Ground Engaging Tools	Oil/ Lubes	Fuel/ Electrical Cost	Total OPEX
Hydraulic Excavator 290 t – 16 m³	US$/h	95.00	23.40	172.00	15.00	25.00	88.35	323.75
Rigid Dump Truck (100 t)	US$/h	74.00	21.27	41.09	10.00	18.00	68.82	159.18
Drill Rig Tracked (58 t)	US$/h	54.00	19.75	40.50	42.10	3.20	50.22	155.77
Dozer (630 HP)	US$/h	65.00	38.00	43.50	20.00	11.00	60.45	172.95
Wheel Loader (Bucket 10 m³)	US$/h	85.00	22.00	50.59	53.00	10.00	79.05	214.64
Grader (227HP)	US$/h	13.20	14.00	13.08	5.00	9.00	12.28	53.36
Road Compactor (15 t)	US$/h	7.40	3.76	3.92	3.00	8.00	6.88	25.56
Water Truck (30 kL)	US$/h	26.00	20.00	46.20	0.00	5.00	24.18	95.38
Wheel Dozer (Straight Blade 6.67 m³)	US$/h	42.00	9.00	28.60	6.00	8.00	39.06	90.66
Fuel/Service Truck	US$/h	26.00	2.08	26.68	0.00	5.00	24.18	57.94
Tyre Handler	US$/h	20.00	2.96	20.88	0.00	2.00	18.60	44.44
Hydraulic Excavator (70 t Bucket 3 m³)	US$/h	24.00	2.50	17.50	5.00	5.00	22.32	52.32
150 t Low Bed	US$/h	15.00	2.50	6.50	0.00	1.00	13.95	23.95
Electrician's Truck	US$/h	12.00	2.50	6.50	0.00	1.00	11.16	21.16
Light Flatbed Truck	US$/h	12.00	2.50	6.50	0.00	1.00	11.16	21.16
Mobile Crane	US$/h	15.00	2.50	6.50	0.00	1.00	13.95	23.95
Pumper's Truck	US$/h	12.00	2.00	7.00	0.00	1.00	11.16	21.16
Pump	US$/h	12.00	0.55	18.45	0.00	1.00	11.16	31.16
Bus (60-seater)	US$/h	15.00	2.00	6.50	0.00	1.50	13.95	23.95
Foreman's Vehicle	US$/h	10.00	1.50	2.50	0.00	1.00	9.30	14.30
Prime Mover Crew Vehicle	US$/h	10.00	1.50	2.50	0.00	1.00	9.30	14.30
Supervision Vehicle	US$/h	10.00	1.50	2.50	0.00	1.00	9.30	14.30
Minibus	US$/h	10.00	1.50	2.50	0.00	1.00	9.30	14.30
Light Plant	US$/h	5.00	1.50	2.50	0.00	1.00	4.65	9.65
Waste Crushing and Conveying and Stacking	US$/h	N/A	N/A	434.96	N/A	Included	336.96	771.92

The average mining cost over the life of the project is US$1.96 per total tonne. This cost includes the mining unit costs per area in Table 21.24.

Table 21.24: Rovina Valley LOM Mining Costs

Mining OPEX per Cost Area	LOM Total (US$ Thousand)	LOM Average OPEX (US$/total t)
General Mine and Engineering	45,086	0.12
Drilling	33,537	0.09
Blasting	219,724	0.58
Loading – Excavators	76,463	0.20
Loading – Rehandle	17,394	0.05
Waste Haulage	95,832	0.25
Ore Haulage	51,142	0.13
Crushing and Conveying	58,517	0.15
Waste Dozing (Track Dozer)	28,853	0.08
Tyre Dozer	3,462	0.01
Grader	3,576	0.01
Water Truck	7,888	0.02
Compactor	2,842	0.01
Support Equipment	23,209	0.06
Mining Electricity	76,237	0.20
Total Mining Operations	**743,765**	**1.96**

The total mining cost per annum in US$ millions is depicted in Figure 21.3.

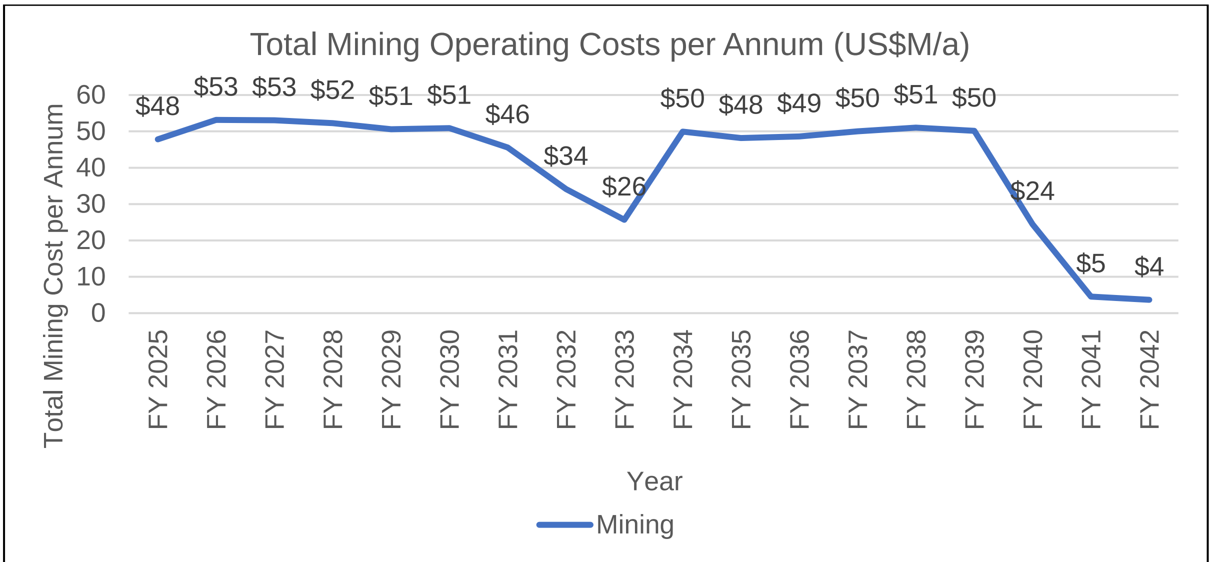


Figure 21.3: Mining OPEX per Annum

The total mining cost breakdown per cost area and the resultant operating cost per total tonne mined are depicted in Figure 21.4 and the mining LOM OPEX distribution in Figure 21.5.

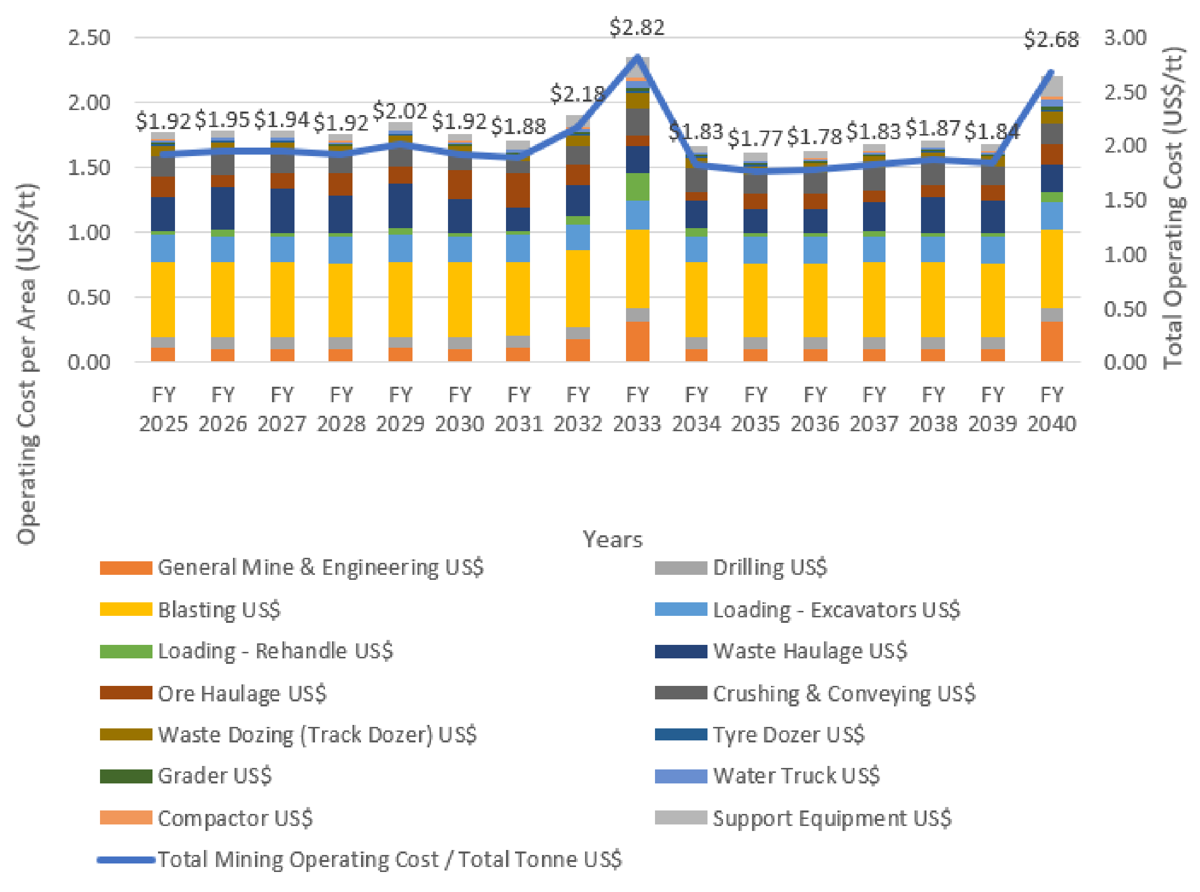


Figure 21.4: Mining OPEX (US$ per Total Tonne)

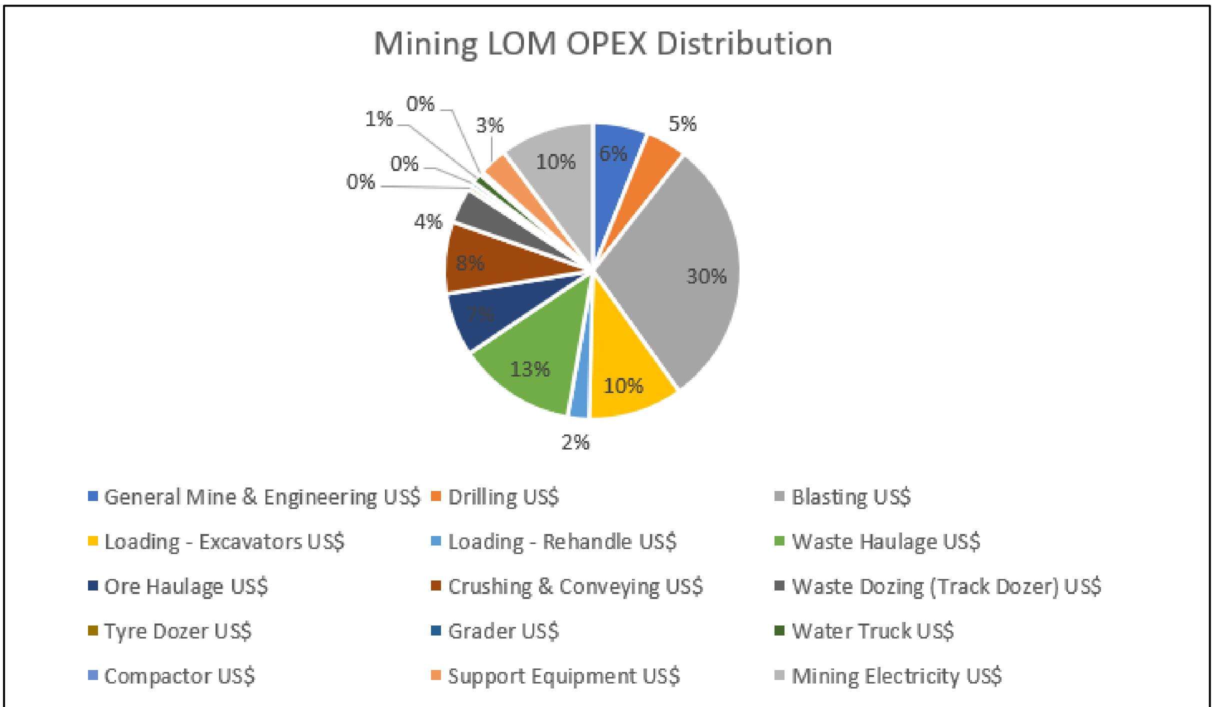


Figure 21.5: Mining LOM OPEX Distribution

The mining OPEX per milled ore tonne is shown in Figure 21.6.

Figure 21.6: Mining Unit Cost per Tonne Milled per Annum

It should be noted that the total cost per tonne peak in Years 2032 and 2033 is smoothed by the Rovina high-grade stockpiles that will be processed at that time.

21.3.3 Process Plant OPEX

21.3.3.1 Basis of Estimate

The process plant OPEX was compiled from a variety of sources, notably:

- First principles, where applicable
- Supplier quotations on reagents and consumables
- SENET's in-house experience and database where applicable
- Client input

The following are the main cost elements of the process plant:

- Reagents and consumables
- Power
- Process plant operating and maintenance labour
- Maintenance parts and supplies
- Plant vehicles
- Assay

21.3.3.2 Process Plant OPEX Summary

The overall LOM process plant OPEX is summarised in Table 21.25, and the distribution of the cost is shown in Figure 21.7. A yearly breakdown of the process plant OPEX is provided in Table 21.26.

Table 21.25: Process Plant OPEX Summary

Description	LOM Cost	Cost
	US$	US$/t feed
Labour	29,655,458	0.25
Power	388,555,307	3.25
Consumables	395,773,860	3.32
Maintenance Supplies and Plant Vehicles	42,186,565	0.35
Assay Costs	17,664,000	0.134
TOTAL	**876,078,012**	**7.34**

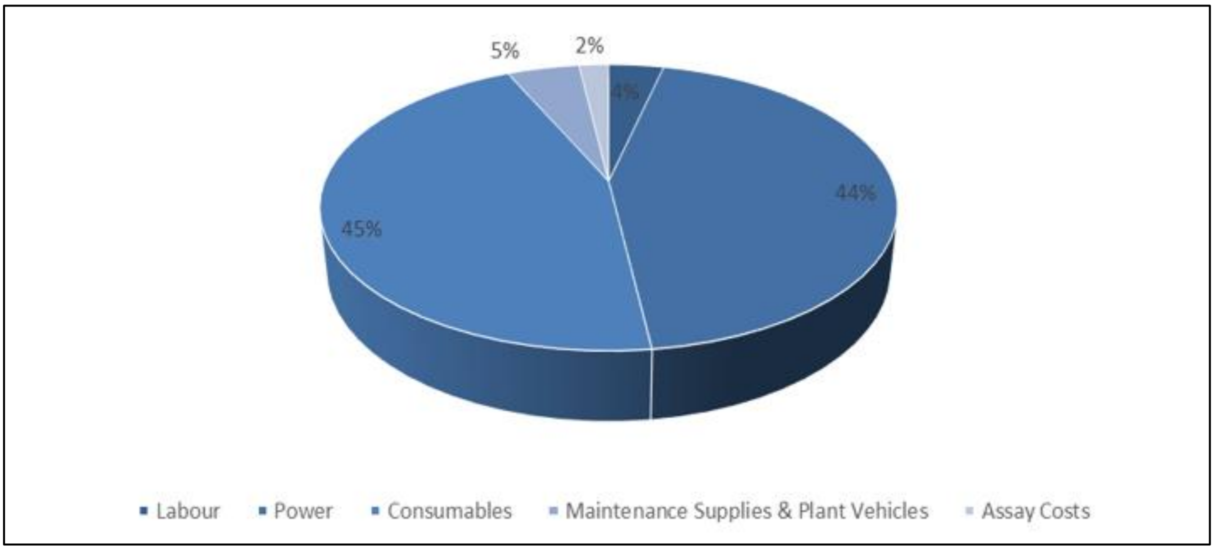


Figure 21.7: Plant OPEX Distribution

Table 21.26: Process Plant LOM OPEX

Description	Unit	LOM	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033	FY 2034
			Year 0	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9
Labour	US$ Million	32.20	1.79	1.79	1.79	1.79	1.79	1.79	1.79	1.79	1.79	1.79
Fixed Power	US$ Million	101.20	2.11	6.10	6.10	6.10	6.10	6.10	6.10	6.10	6.10	6.10
Maintenance Supplies and Plant Vehicles	US$ Million	42.19	2.34	2.34	2.34	2.34	2.34	2.34	2.34	2.34	2.34	2.34
Assay	US$ Million	17.37	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96	0.96
Total Fixed Cost	**US$ Million**	**192.9**	**7.21**	**11.20**	**11.20**	**11.20**	**11.20**	**11.20**	**11.20**	**11.20**	**11.20**	**11.20**
Consumables	US$ Million	139.7	3.09	8.39	8.39	8.39	8.39	8.39	8.39	8.39	8.39	8.39
Reagents	US$ Million	256.1	5.24	15.46	15.46	15.46	15.46	15.46	15.46	15.46	15.46	15.46
Variable Power	US$ Million	287.4	5.87	17.33	17.33	17.33	17.33	17.33	17.33	17.33	17.33	17.33
Total Variable Cost	US$ Million	683.1	14.21	41.19	41.19	41.19	41.19	41.19	41.19	41.19	41.19	41.19
Total LOM Cost (Fixed and Variable)	US$/a	876.078	21.42	52.38	52.38	52.38	52.38	52.38	52.38	52.38	52.38	52.38
Total LOM Cost (Fixed and Variable)	US$/t feed	**7.34**	**8.78**	**7.28**	**7.28**	**7.28**	**7.28**	**7.28**	**7.28**	**7.28**	**7.28**	**7.28**

21.3.3.3 Reagents and Consumables

The reagents and consumables costs were calculated by using vendor supply costs together with the consumptions of the respective reagents or consumables based upon test work results. The reagents and consumables supplied costs are shown in Table 21.27.

Table 21.27: Reagents and Consumables Supplied Costs

Description	Unit	Cost
PRIMARY CRUSHER CONCAVES		
Concave (1st Row)	US$/set	68,223
Concave (2nd Row)	US$/set	54,573
Concave (3rd Row)	US$/set	110,887
Concave (4th Row)	US$/set	109,457
PRIMARY CRUSHER MANTLE		
Primary Crusher	US$/set	188,841
Head Nut	US$/piece	64,872
Locking Pin	US$/piece	1,284
PEBBLE CRUSHER		
Concave	US$/set	7,465
Bottom Shell Lining Kit	US$/set	17,943
Mantle	US$/set	8,054
MILL LINERS, GRINDING MEDIA AND REAGENTS		
SAG Mill Liner Cost per Set – Steel	US$/set	1,735,284
SAG Mill Grinding Media – 125 mm	US$/t	1,003
Ball Mill Liner Cost per Set – Steel	US$/set	910,842
Ball Mill Liner Cost per Set – Rubber	US$/set	1,227,200
Ball Mill Grinding Media – 50 mm	US$/t	1,143
Concentrate Regrind Mill – IsaMill	US$/a	214,230
IsaMill Grinding Media – 3.5 mm	US$/t	2,507
Collector – PAX	US$/t	2,030
Promoter – Aerophine A3418A	US$/t	14,440
Frother – Aerofroth 65	US$/t	4,780
Lime – Quicklime	US$/t	128.67
Flocculant (SNF AN905 SH)	US$/t	2,640
Potable Water Treatment Plant Chemicals	US$/a	2,953
Wastewater Treatment Plant Chemicals	US$/a	1,200,000

The reagents and consumables LOM consumptions and OPEX are provided in Table 21.28 and Table 21.29, respectively.

Table 21.28: Reagents and Consumables Consumptions

Description	LOM (t)	FY 2025 Year 0 (t/a)	FY 2026 Year 1 (t/a)	FY 2027 Year 2 (t/a)	FY 2028 Year 3 (t/a)	FY 2029 Year 4 (t/a)	FY 2030 Year 5 (t/a)	FY 2031 Year 6 (t/a)	FY 2032 Year 7 (t/a)	FY 2033 Year 8 (t/a)	FY 2034 Year 9 (t/a)
Gyro Crusher Liners and Mantle	622	12.7	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5	37.5
Pebble Crusher Liners and Mantle	338	6.9	20.4	20.4	20.4	20.4	20.4	20.4	20.4	20.4	20.4
SAG Mill Liners	13,609	278	821	821	821	821	821	821	821	821	821
Ball Mill Liners	14,444	295	871	871	871	871	871	871	871	871	871
SAG Mill Grinding Media – 125 mm	68,998	1,410	4,162	4,162	4,162	4,162	4,162	4,162	4,162	4,162	4,162
Ball Mill Grinding Media – 50 mm	111,376	2,277	6,718	6,718	6,718	6,718	6,718	6,718	6,718	6,718	6,718
IsaMill Grinding Media – 3.5 mm	1,894	38.7	114	114	114	114	114	114	114	114	114
Promoter	2,388	48.80	144	144	144	144	144	144	144	144	144
Lime	54,912	1,122	3,312	3,312	3,312	3,312	3,312	3,312	3,312	3,312	3,312
Frother	2,388	48.8	144	144	144	144	144	144	144	144	144
Flocculant	3,799	77.65	229	229	229	229	229	229	229	229	229
Collector (PAX)	1,791	31.72	93.60	93.60	93.60	93.60	93.60	93.60	93.60	93.60	93.60
Sodium Hypochlorite	52.0	1.02	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00	3.00
Hydrochloric Acid	0.4	0.008	0.024	0.024	0.024	0.024	0.024	0.024	0.024	0.024	0.024

Table 21.29: LOM Summary of Reagents and Consumables OPEX

Description	LOM (US$)	FY 2025 Year 0 (US$/a)	FY 2026 Year 1 (US$/a)	FY 2027 Year 2 (US$/a)	FY 2028 Year 3 (US$/a)	FY 2029 Year 4 (US$/a)	FY 2030 Year 5 (US$/a)	FY 2031 Year 6 (US$/a)	FY 2032 Year 7 (US$/a)	FY 2033 Year 8 (US$/a)	FY 2034 Year 9 (US$/a)
Gyro Crusher Liners and Mantle	8,869,808	181,298.7	534,979.8	534,979.8	534,979.8	534,979.8	534,979.8	534,979.8	534,979.8	534,979.8	534,979.8
Pebble Crusher Liners and Mantle	2,328,458	47,593.6	140,440.3	140,440.3	140,440.3	140,440.3	140,440.3	140,440.3	140,440.3	140,440.3	140,440.3
SAG Mill Liners	48,669,804	999,166.3	2,948,359.5	2,948,359.5	2,948,359.5	2,948,359.5	2,948,359.5	2,948,359.5	2,948,359.5	2,948,359.5	2,922,496.7
Ball Mill Liners	50,776,815.3	1,135,860.5	3,351,719.6	3,351,719.6	3,351,719.6	3,351,719.6	3,351,719.6	3,351,719.6	3,351,719.6	3,351,719.6	2,770,016.2
IsaMill	3,551,878	72,600	214,231	214,231	214,231	214,231	214,231	214,231	214,231	214,231	214,231
SAG Mill Grinding Media – 125 mm	69,205,101	1,414,551	4,174,085	4,174,085	4,174,085	4,174,085	4,174,085	4,174,085	4,174,085	4,174,085	4,174,085
Ball Mill Grinding Media – 50 mm	127,302,580	2,602,062	7,678,217	7,678,217	7,678,217	7,678,217	7,678,217	7,678,217	7,678,217	7,678,217	7,678,217
IsaMill Grinding Media – 3.5 mm	4,748,204	97,053	286,387	286,387	286,387	286,387	286,387	286,387	286,387	286,387	286,387
Promoter	23,436,256	479,037	1,413,551	1,413,551	1,413,551	1,413,551	1,413,551	1,413,551	1,413,551	1,413,551	1,413,551
Lime	7,065,525	144,419	426,155	426,155	426,155	426,155	426,155	426,155	426,155	426,155	426,155
Frother	11,412,144	233,264	688,320	688,320	688,320	688,320	688,320	688,320	688,320	688,320	688,320
Flocculant	10,028,767	204,988	604,882	604,882	604,882	604,882	604,882	604,882	604,882	604,882	604,882
PAX	3,150,277	64,392	190,008	190,008	190,008	190,008	190,008	190,008	190,008	190,008	190,008
Potable Water Treatment Plant	48,963.3	1,001	2,953.2	2,953.2	2,953.2	2,953.2	2,953.2	2,953.2	2,953.2	2,953.2	2,953.2
Wastewater Treatment Plant	19,895,648	406,667	1,200,000	1,200,000	1,200,000	1,200,000	1,200,000	1,200,000	1,200,000	1,200,000	1,200,000
Total	**392,216,551**	**8,880,226**	**23,855,274**	**23,855,274**	**23,855,274**	**23,855,274**	**23,855,274**	**23,855,274**	**23,855,274**	**23,855,274**	**23,247,708**

21.3.3.3.1 Crusher Liners

The primary crusher and pebble liner costs were obtained from vendor information by estimating the number of liner changes per annum using the abrasion indices obtained from metallurgical tests and the expected liner life for a given throughput. Quotations for the crusher liners, including the weights of the liners, were obtained from a vendor. The estimated delivered costs (using the customs clearance, port handling and transport costs to site obtained from ESM) were calculated and used together with the liner supply cost.

21.3.3.3.2 Mill Liners

The SAG and ball mill liner costs were based on estimating the liner consumption by using the abrasion index results obtained from test work. OMC used the test work data to simulate the expected wear rates. This was further cross-referenced by wear rates from other operating mines using the same type of liners and grinding media size based on SENET's experience. The current pricing for a set of liners for the SAG and ball mills was obtained from a liner supply vendor and used in the cost estimate based on the number of liner changes per annum for a given throughput. IsaMill costs per year were obtained from the supplier. The delivered costs were estimated using the customs clearance, port handling and transport costs to site obtained from ESM.

21.3.3.3.3 Mill Grinding Media

The grinding media costs were obtained by estimating the consumption using the standard method abrasion index results that were obtained from laboratory tests. OMC used the test work data to estimate the expected mill grinding media consumptions. The grinding media consumption for the IsaMill was estimated by the supplier. In addition, the mill throughputs and quotations for 125 mm and 50 mm balls and 3.5 mm ceramic media were obtained from suppliers. Quotations obtained from grinding media suppliers, together with the consumption, were then used to estimate the grinding media costs.

21.3.3.3.4 Quicklime

Hydrated lime consumption in flotation was estimated from test work. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the quicklime costs.

21.3.3.3.5 Flocculant

Flocculant consumption for tails and concentrate thickening was estimated from test work. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the flocculant costs.

21.3.3.3.6 Collector

Collector (PAX) consumption in flotation circuits was estimated from test work. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the collector costs.

21.3.3.3.7 Frother

Frother consumption in flotation circuits was estimated from test work. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the frother costs.

21.3.3.3.8 Promoter

Promoter consumption in flotation circuits was estimated from test work. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the promoter costs.

21.3.3.3.9 Sodium Hypochlorite

Sodium hypochlorite consumption was based on information from the water treatment plant suppler. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the sodium hypochlorite costs.

21.3.3.3.10 Hydrochloric Acid

The hydrochloric acid (11 %) consumption associated with water treatment filter washing was based on information from the water treatment plant suppler. Quotations obtained from reagent suppliers, together with the consumption, were then used to estimate the hydrochloric acid costs.

21.3.3.3.11 Wastewater Treatment Plant

The costs associated with the wastewater treatment plant were obtained from SENET's in-house estimate based on a similar-sized plant.

21.3.3.4 Power

The power consumption can be categorised into two forms: fixed and variable power. The average continuous fixed power consumption was determined by taking into account the installed power rating of each of the equipment in the plant and infrastructure, excluding standbys and the projected running times. The fixed power draw includes the absorbed operating loads associated with the process plant equipment as detailed in the mechanical equipment list and on-site infrastructure, including the following buildings:

- Plant Offices
- Assay Laboratory
- Administration Building
- Weighbridge
- Control Room
- Gatehouse
- Warehouse
- Reagents Stores
- Ball Storage and Bunker
- Workshop
- Sewage Treatment Plant

- Change House
- First-Aid Building
- Wastewater Treatment Plant
- Potable Water Treatment Plant

The variable power was estimated using the gross specific energy reported by OMC. This mill-specific energy was used with the plant throughput to calculate the annual power usage (kilowatt hours per annum). IsaMill specific energy was estimated by the mill supplier.

The power draw and power costs for the RVP plant and infrastructure were determined using the power consumption basis detailed above and the unit energy cost of US$0.0734/kWh.

Table 21.30 and Table 21.31 show the power draw summaries for fixed and variable power, respectively.

Table 21.30: Power Draw Summary – Fixed Power

Process Area	Process Area Description	Installed Power	Operating Power	Power Consumption
		kW	kW	kWh/a
2100	Primary Crushing	1,461	966	5,847,890
3100	Ore Stockpiling and Reclamation	320	181	1,133,679
3200	Milling	2,922	1,257	8,284,176
3200	Pebble Crushing	241	127	860,256
3300	Flotation Feed Conditioning	484	219	1,615,096
3300	Rougher Flotation	2,642	1,161	9,099,265
3300	Scavenger Flotation	1,706	878	6,929,915
3300	Concentrate Regrind Milling	295	104	769,441
3400	Cleaner Flotation Feed Conditioning	169	73	396,869
3400	Cleaner 1 Flotation	498	262	2,044,350
3400	Cleaner 2 Flotation	234	128	980,960
3400	Cleaner 3 Flotation	209	130	996,000
3400	Online Analyser	105	75	598,800
3500	Concentrate Thickening	354	152	1,115,873
3500	Concentrate Filtration	531	336	7,329,242
3500	Concentrate Bagging	24	15	26,689
3600	Rougher Tails Thickening	2,567	1,396	10,839,550
3600	Rougher Tails Filtration	3,537	1,966	5,081,365
3600	Tails Conveying and Stacking	487	202	1,312,320
3600	Cleaner Tails Thickening	1,153	550	4,193,712
3600	Cleaner Tails Filtration	933	358	1,042,710
3600	Cleaner Tails Conveying and Stacking	245	77	616,000
3700	Lime	203	76	545,607
3700	Flocculant	164	46	329,526
3700	Frother	11	6	16,947
3700	Collector	13	6	13,000

Process Area	Process Area Description	Installed Power	Operating Power	Power Consumption
		kW	kW	kWh/a
3700	Promoter	8	4	458
3800	Air Services	1,602	670	4,681,080
3800	Raw Water	220	90	718,400
3800	Process Water	2,100	762	4,976,000
3800	Potable Water	30	16	126,800
3800	Gland Water	59	22	176,800
3800	Fire Water	200	164	116,220
5600	Wastewater Treatment Plant	355	231	230,750
5700	Sewage Treatment	11	9	51,986
	Total Fixed Power	**26,091**	**12,712**	**83,097,734**

Table 21.31: Power Draw Summary – Variable Power

Process Area	Process Area Description	Specific Energy	Operating Power	Power Consumption
		kWh/t	kW	kWh/a
2100	Primary Crusher	450	333	2,054,943
3200	SAG Mill	16,000	13,392	107,136,000
3200	Ball Mill	16,000	13,727	109,816,644
3200	Pebble Crusher	500	340	2,720,000
3300	Concentrate Regrind Mill	3,000	2,212	17,697,204
	Total	**35,950**	**29,592**	**236,127,587**

21.3.3.5 Process Plant Operating and Maintenance Labour

The estimated annual plant operating and maintenance labour cost was estimated at **US$1,788,660**. The cost was derived from first principles where the actual labour complement for each plant area and maintenance function was identified, and the required number of personnel and their levels were established. The complement derived was then benchmarked against other operations of similar size and complexity.

The operating and maintenance labour cost was broken down as follows:

- Senior Personnel (office based):
  - Basic remuneration
  - Yearly bonus
  - Fringe benefits
- Operations Personnel (on shift):
  - Basic remuneration
  - Fringe benefits
  - Overtime

The labour schedule and renumeration rates were developed by ESM in consultation with SAMAX and with input from SENET.

The categories selected for the various positions are described in Table 21.32.

Table 21.32: Process Plant Labour Categories

Employee Category	Schedule
Staff	Office
Hourly	Shift

Table 21.33 shows the labour cost summary. The detailed breakdown is given in Table 21.34.

Table 21.33: Process Plant Labour Cost Summary

Position Description	Total Number of Employees	Total Cost	Total Cost
		US$/a	US$/t feed
Subtotal Mill Management	3	140,400	0.02
Subtotal Mill Operations	34	663,480	0.09
Subtotal Mill Maintenance	32	599,580	0.08
Subtotal Mill Metallurgy	15	385,200	0.05
Total Cost	**84**	**1,788,660**	**0.25**

Table 21.34: Detailed Process Plant Labour Cost Breakdown

Position Description	Employee Category	Total Number of Employees	Rate/Employee	Total Cost/Employee	Total Cost
			US$/a	US$/a	US$/a
Mill Administration					
Mill Superintendent	Staff	1	72,000	86,400	86,400
Mill Clerk	Staff	2	22,500	27,000	54,000
Subtotal Mill Administration		**3**			**140,400**
Mill Operations					
General Operations Foreman	Staff	1	36,000	43,200	43,200
Shift Foreman	Staff	2	27,000	33,750	67,500
Trainer and Safety Coordinator	Staff	1	22,500	27,000	27,000
Control Room Operator	Hourly	4	18,000	20,700	82,800
Plant Operators	Hourly	10	16,200	18,630	186,300
Plant Helpers	Hourly	12	13,500	15,525	186,300
Mill Labour (including mobile equipment)	Hourly	4	15,300	17,595	70,380
Subtotal Mill Operations		**34**			**663,480**
Mill Maintenance					
General Maintenance Foreman	Staff	1	36,000	43,200	43,200
Mechanical Maintenance Foreman	Staff	1	27,000	32,400	32,400
E&I Foreman	Staff	1	27,000	32,400	32,400
Maintenance Planner	Staff	1	19,800	23,760	23,760
Maintenance Journeyman	Hourly	4	16,200	18,630	74,520
Maintenance Apprentice	Hourly	8	13,500	15,525	124,200
Electrical Journeyman	Hourly	4	16,200	18,630	74,520

Position Description	Employee Category	Total Number of Employees	Rate/Employee	Total Cost/Employee	Total Cost
			US$/a	US$/a	US$/a
Electrical Apprentice	Hourly	4	13,500	15,525	62,100
Instrumentation Technician	Hourly	4	16,200	18,630	74,520
Instrumentation Apprentice	Hourly	4	12,600	14,490	57,960
Subtotal Mill Maintenance		**32**			**599,580**
Mill Metallurgy					
Chief Metallurgist	Staff	1	39,600	47,520	47,520
Senior Metallurgist	Staff	2	31,500	37,800	75,600
Junior Metallurgist	Staff	4	19,800	24,750	99,000
Metallurgical Technician	Staff	4	16,200	20,250	81,000
Environmental Coordinator	Staff	2	18,000	21,600	43,200
Environmental Technician	Staff	2	16,200	19,440	38,880
Subtotal Mill Metallurgy	Staff	**15**			**385,200**
Total Process Manpower		**84**			**1,788,660**

21.3.3.6 Maintenance Parts and Supplies

The plant maintenance parts and supplies annual costs for the RVP were estimated at **US$2,343,698.** Plant maintenance and supplies costs refer to the costs of operating spares, lubricants, and other maintenance-related consumables for the plant. An average annual cost was calculated using the maintenance cost factors as shown in Table 21.35 for the various commodities. The plant maintenance, parts and supplies costs are summarised in Table 21.36.

Table 21.35: Plant Maintenance Cost Factors

Description	Maintenance Factor (%)
Mechanical Equipment CAPEX Base Case	2.5
Piping and Valves CAPEX	2.5

Table 21.36: Plant Maintenance, Parts and Supplies OPEX

Description	Unit	Quantity
Machinery and Equipment		
Mechanical Equipment CAPEX	US$	82,717,939
Maintenance Cost Factor	%	2.5
Total Annual Cost	US$	2,067,948
Piping and Valves		
Piping and Valves CAPEX	US$	7,389,644
Maintenance Cost Factor	%	2.5
Total Annual Cost	US$	184,741
Maintenance Parts and Supplies Cost	**US$/a**	**2,343,698**
Maintenance Parts and Supplies Cost	**US$/t feed**	**0.31**

21.3.3.7 Plant Vehicles

The annual plant vehicle OPEX was estimated at **US$91,009**. The OPEX was determined based on unit operating costs, utilisation and fuel costs. The fuel cost was obtained from ESM, and the unit costs were obtained from suppliers. The detailed breakdown is given in Table 21.37.

Table 21.37: Plant Vehicle OPEX

Description	Unit	Total Cost (US$/a)
Vehicle		
12 × Plant Vehicles Toyota Hilux Double cab 4x4 3.0 diesel	US$	26,234
Plant Mobile Equipment		
1 × Mobile Crane 80 t Zoomlion ZTC700V552	US$	4,255

Description	Unit	Total Cost (US$/a)
2 × Mobile Crane 20 t ZJCM RT30 rough terrain crane Zoomlion RT35	US$	5,685
2 × Manitou Forklifts MX50-2	US$	3,970
1 × Manitou Forklifts MTX1440	US$	2,912
3 × Bobcat 226B3 Skid Steer	US$	13,099
2 × Front-End Loaders CAT 966L	US$	27,707
2 × Tractor and Trailer Case IH JX 80 4wd and12 m trailer	US$	2,298
2 × 10 t Lorry Tata LPT 2523 ex2 6x4 with Palfinger Crane	US$	2,256
1 × Mobile Rock breaker Excavator 330D2l + H130ES Hammer	US$	2,593
Maintenance and Fuel Cost	**US$/a**	**91,009**
Maintenance and Fuel Cost	**US$/t feed**	**0.01**

21.3.3.8 Assay

The annual assay OPEX was obtained from a contractor's quotation and was estimated at **US$964,800**.

The assay costs consist of the mining/grade control and process plant assay requirements. The process plant assay OPEX was determined by identifying the samples to be collected, the frequency of the sampling, and the type of analysis required. The number of metal accounting samples was based on a shift cycle of three 8-hour shifts per day. The metallurgical control samples would be collected as and when required. An allowance of the number per annum has been made to account for these.

The number of grade control samples was estimated based on the mining plan, to identify how many samples and where in the pit they would be collected. These sample numbers and required tests were submitted to analytical laboratory contractors for quotations.

The costs for the plant and mining assay requirements are summarised in Table 21.38.

Table 21.38: Assay OPEX Summary

Description	Cost	Cost
	US$/month	US$/a
Labour	23,742	284,904
Consumables	20,000	240,000
Equipment Amortisation	23,239	278,868
Staff Travel Costs	583	6,996
Repairs and Maintenance	3,075	36,900
Quality Costs	517	6,204
Miscellaneous	9,244	110,928
Total Cost	**80,400**	**964,800**

21.3.4 WMF and SWM OPEX

The OPEX associated with the WMF and SWM has been estimated at US$52.245 million over the LOM, based on a set of MTOs developed for the WMF and SWM features. A set of unit rates was developed for the line-items included in the MTO, based on similar project experience, client references, and quotes from local suppliers in Romania. This estimate includes annual ground preparation and underdrain construction for WMF expansion including related earthworks and liner costs, as well as costs related to annual waste placement, spreading, and compaction. The estimate also includes earthworks costs related to construction and expansion of the SWM facilities through the LOM, after start-up.

Other OPEX related to the WMF and SWM, such as inspection, maintenance, surveillance, progressive rehabilitation and closure, are included elsewhere in the DFS cost estimate

A summary LOM of the WMF and SWM OPEX is shown in Table 21.39.

Table 21.39: LOM WMF and SWM OPEX Summary

Description	Unit	LOM (US$)	FY 2025 Year 0 (US$/a)	FY 2026 Year 1 (US$/a)	FY 2027 Year 2 (US$/a)	FY 2028 Year 3 (US$/a)	FY 2029 Year 4 (US$/a)	FY 2030 Year 5 (US$/a)	FY 2031 Year 6 (US$/a)	FY 2032 Year 7 (US$/a)	FY 2033 Year 8 (US$/a)	FY 2034 Year 9 (US$/a)
SWM	US$/a	2,866,000	225,000	212,500	250,000	125,000	112,500	112,500	212,500	225,000	50,000	37,875
WMF	US$/a	49,379,000	3,564,000	3,478,000	4,293,000	4,010,000	2,474,000	2,166,000	3,094,000	3,616,000	2,576,000	2,495,000
Total SWM and WMF Cost	**US$/a**	**52,245,000**	**3,789,000**	**3,690,500**	**4,543,000**	**4,135,000**	**2,586,500**	**2,278,500**	**3,306,500**	**3,841,000**	**2,626,000**	**2,532,875**

21.3.5 G&A Costs

The G&A annual OPEX was estimated at **US$1,964,120**. The costs were determined by ESM from first principles and by using information from SAMAX's in-house database for similar projects from the same locality.

A summary LOM of the G&A OPEX is shown in Table 21.40.

Table 21.40: LOM G&A Cost Summary

Description	Unit	LOM (US$)	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033	FY 2034
			Year 0 (US$/a)	Year 1 (US$/a)	Year 2 (US$/a)	Year 3 (US$/a)	Year 4 (US$/a)	Year 5 (US$/a)	Year 6 (US$/a)	Year 7 (US$/a)	Year 8 (US$/a)	Year 9 (US$/a)
Salaries and Wages	US$/a	27,293,492	533,452	1,574,120	1,574,120	1,574,120	1,574,120	1,574,120	1,574,120	1,574,120	1,574,120	1,574,120
Material and Services	US$/a	6,762,167	132,167	390,000	390,000	390,000	390,000	390,000	390,000	390,000	390,000	390,000
Total G&A Cost	**US$/a**	34,055,658	665,618.4	1,964,120	1,964,120	1,964,120	1,964,120	1,964,120	1,964,120	1,964,120	1,964,120	1,964,120

21.3.5.1 G&A Labour

The labour cost for G&A personnel was estimated using the same principles and labour rates as outlined in the process plant labour section. The G&A labour cost summary is shown in Table 21.41, and the detailed breakdown is given in Table 21.42.

Table 21.41: G&A Labour Cost Summary

Description	Total Number of Employees	Cost	Total Cost
		US$/month	US$/a
General Management	5	76,667	920,000
Finance and Materials Management	11	30,274	363,285
Human Resources Management	8	17,078	204,930
Environmental	3	7,158	85,908
Total	**27**	**131,177**	**1,574,120**

Table 21.42: Detailed G&A Labour Cost Breakdown

Position Description	Total Number of Employees	Rate/Employee	Total Cost/Employee	Total Cost
		US$/a	US$/a	US$/a
MANAGEMENT				
CEO (Corporate)	1	275,000	316,250	316,250
General Manager (Corporate)	1	250,000	287,500	287,500
Deputy General Manager (Corporate)	1	140,000	161,000	161,000
Assistant General Manager	1	85,000	97,750	97,750
General Manager Secretary (Corporate)	1	50,000	57,500	57,500
Subtotal	**5**			
FINANCE AND MATERIALS MANAGEMENT				
Manager – Finance and Administration	1	45,000	51,750	51,750
Senior Administrative Assistant	1	27,000	31,050	31,050
Senior Accountant – Supervisor	1	36,000	41,400	41,400
Accountant	1	27,000	31,050	31,050
Accounting Clerk	1	18,000	20,700	20,700
Manager – Purchasing and Logistics	1	45,000	51,750	51,750
Materials Management Superintendent	1	27,000	31,050	31,050
Senior Logistics Officer	1	27,000	31,050	31,050
Senior Purchasing Officer	1	27,000	31,050	31,050
IT Coordinator	1	22,500	25,875	25,875
IT Assistant	1	14,400	16,560	16,560
Subtotal	**11**			

Position Description	Total Number of Employees	Rate/Employee	Total Cost/Employee	Total Cost
		US$/a	US$/a	US$/a
HUMAN RESOURCES				
Manager Human Resources	1	45,000	51,750	51,750
Human Resources Officer	1	27,000	31,050	31,050
Community Relations Coordinator	1	22,500	25,875	25,875
Manager Health, Safety and Training	1	22,500	25,875	25,875
Senior Officer – Health, Safety and Training	1	18,000	20,700	20,700
Training Foreman	1	16,200	18,630	18,630
Nurse	1	14,400	16,560	16,560
Assistant Nurse	1	12,600	14,490	14,490
Subtotal	**8**			
ENVIRONMENTAL				
Environment Manager	1	36,000	41,400	41,400
Environment Officer	1	22,500	25,875	25,875
Environment Technician	1	16,200	18,630	18,630
Subtotal	**3**			
TOTAL	**27**			**1,574,120**

21.3.5.2 Material and Services

The material and services costs were estimated to be **US$390,000** per annum by ESM in consultation with SAMAX.

The costs are summarised in Table 21.43. The detailed breakdown is given in Table 21.44.

Table 21.43: Material and Services

Description	Cost	Cost
	US$/month	US$/a
Management	17,750	213,000
Administration and Accounting	3,000	36,000
Purchasing and Warehouses	1,000	12,000
Human Resources	10,750	129,000
Total	**32,500**	**390,000**

Table 21.44: Detailed Material and Services Cost Breakdown

Description	Cost	Cost
	US$/month	US$/a
MANAGEMENT		
Security Agency	1,500	18,000
Mining Leases	4,167	50,000
Municipal Taxes	2,083	25,000
Site Insurance	4,167	50,000
Mining Association Fees	833	10,000
Travel Expenses	–	–
Telecommunications	1,000	12,000
Janitoring	1,500	18,000
Employee Transportation to Site	2,500	30,000
Office supplies and Miscellaneous Costs	–	–
Subtotal		**213,000**
ADMINISTRATION AND ACCOUNTING		
Transport of Goods	1,000	12,000
IT Maintenance and Supplies	2,000	24,000
Subtotal		**36,000**
HUMAN RESOURCES		
Recruiting	1,000	12,000
Training	1,000	12,000
Safety – Equipment and Supplies	1,500	18,000
Medical and First Aid	750	9,000
Employee Relations	1,000	12,000
Community Relations	2,000	24,000
Employee Transportation (On Site)	1,000	12,000
Legal	1,000	12,000
Contingency	1,500	18,000
Subtotal		**129,000**
TOTAL		**390,000**

22 ECONOMIC ANALYSIS

22.1 EVALUATION PRINCIPLES AND METHOD

This economic evaluation of the RVP was prepared by SENET for ESM as part of the NI 43-101 Technical Report. The NI 43-101 Technical Report presents an independent financial valuation of the Mineral Resources and Mineral Reserve of the RVP, based on the technical, economic, and financial findings of the DFS as documented in this report. The QP neither holds any interest in ESM nor has any association with ESM. The QP will not receive any benefit should the valuation assignment and report result in a transaction. The valuation is restricted to the Mineral Asset itself and does not provide a valuation of any corporate securities or liabilities which may be held by ESM.

The valuation methodology applied is the widely accepted income approach, which is the preferred method where there is a sufficient body of technical and economic studies and/or historical operational information to forecast production, sales, CAPEX and OPEX for the asset. This allows the determination of the present value of asset cash flows to ascertain the estimated asset value. This method is commonly used for development properties, production properties and economically viable dormant properties.

The valuation was done by building a financial model which projected the future income of the RVP based on the forecast metal prices and metal production over the project life and subtracted from the income all CAPEX (establishment and sustaining), OPEX, selling expenses, working capital requirements, royalties, and taxes to give the free cash flow (FCF) of the project. The FCF assumes a 100 % equity investment and excludes financing cash flows. This FCF was discounted at an appropriate discount rate to give the net present value (NPV) of the project.

22.2 ASSUMPTIONS

22.2.1 Timeframe and Valuation Date

The model runs from the construction start date of 1 June 2023 to the end of the LOM on 31 May 2042. Year 2043 is included to allow for the unwinding of working capital. The model financial years (FYs) are numbered according to the year of the last month, for example, the first model year, from 1 June 2023 to 31 May 2024, is known as “FY 2024”.

The model is based on the project development and LOM schedule produced by SENET and the detailed CAPEX and OPEX budgets and supporting information produced in the DFS.

The model is based on the United States dollar at the valuation date of 31 December 2020.

22.2.2 Inflation and Discount Rates

The financial model was done in real United States dollars as at 31 December 2020, and inflation was not applied.

ESM is listed on the Toronto Stock Exchange (TSX) and extensive financial data is available. Additional corporate financial information is also available on SEDAR.com. However, the current weighted average cost of capital for the company is not seen as a suitable discount rate for the RVP as the capital structure of ESM will change substantially when funds are raised for the project. A real, post-tax discount rate of 5 % was applied, which considers the current low costs of funding (US Fed rate 0.25 %, US LIBOR 12 month 0.28 %), the low-risk profile of the project due to its location in the European Union, the advanced level of the DFS conducted, the relatively near-term production schedule, and gold as the principal revenue metal in the first half of the LOM. This discount rate is commonly applied to projects currently. The project NPV is, however, presented for a range of discount rates.

22.2.3 Gold and Copper Prices

Section 19 provides a detailed discussion of the gold and copper prices applied in the economic analysis. The prices applied are in real 2020 dollars, set at US$1,550/oz for gold and US$3.30/lb for copper through the LOM. These prices are slightly higher than the US$1,500/oz for gold and US$3.00/lb for copper applied in the reserve estimate. The higher prices in the economic assessment reflect higher prevailing market prices than the more conservative prices applied to conduct the mine design.

22.2.4 Taxation and Royalties

Taxation and mineral royalty assumptions were sourced from the EY Worldwide Corporate Tax Guide for 2020 and relevant Romanian government websites. The assumptions applied were confirmed by ESM.

The corporate taxation assumptions applied in the economic analysis are shown in Table 22.1.

Table 22.1: Corporate Taxation Assumptions

Mining Corporate Tax Rate	Unit	% Taxable Income	Notes
Corporate Income Tax	%	16.00	
Dividend Withholding Tax	% dividends	5.00	
Tax Depreciation – Year 1	% /a	50.00	Straight line thereafter

The royalty rates applied are 6 % of the gold revenue and 5 % of the copper revenue.

22.2.5 Working Capital

The working capital assumptions applied in the economic analysis are receivables (debtors) of 30 days of sales and payables (creditors) of 30 days of costs.

Stockpile balances are not maintained in the model, and production costs related to stockpiled ore are expensed in the year they occur.

22.3 PRODUCTION

Production forecasts in the model are based on the technical work contained in Sections 16 and 17.

22.3.1 Mining

The scheduled waste and ore tonnages for the Colnic pit are shown in Figure 22.1 and Table 22.2. The Colnic pit is mined at a steady state of approximately 27 Mt/a until FY 2030, thereafter tailing off rapidly to 6.6 Mt by its final year, FY 2032. In this period, the Colnic pit produces 111 Mt of waste and 74 Mt of ore; 56 Mt of the ore is high grade and 17 Mt low grade.

The distinction between high-grade and low-grade ore is made on the basis of gold grade. For the Colnic pit, the split is variable over time: for the first 2.5 years, the minimum grade for high-grade ore is 0.45 g/t Au, for the next 2.5 years, it is 0.40 g/t Au, and for the remainder of the life of Colnic pit, it is 0.28 g/t Au. For the Rovina pit, the minimum grade for high-grade ore is a constant 0.28 g/t Au. The minimum grade for low-grade ore for both pits is 0.20 g/t Au.

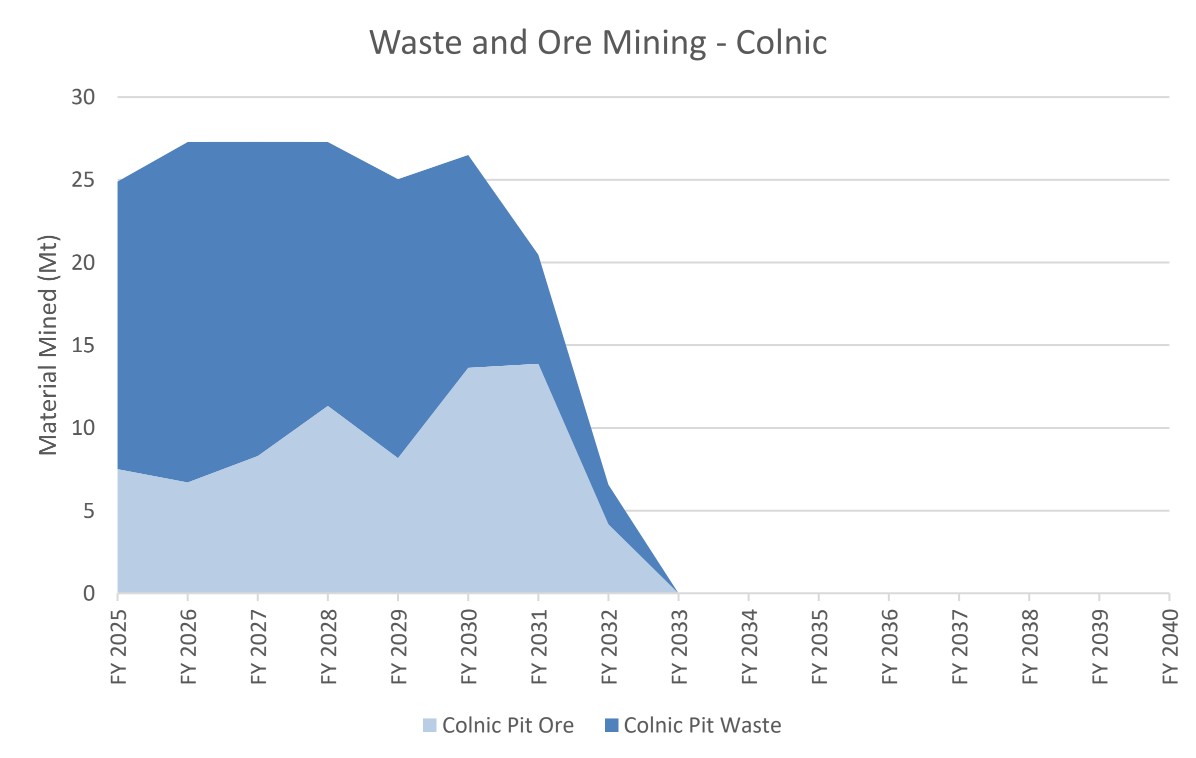


Figure 22.1: Mining at the Colnic Pit

Table 22.2: Colnic Mining Volumes

Volume (Mt)	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032
Colnic Pit Waste	17.4	20.6	19.0	16.0	16.9	12.9	6.6	2.4
Colnic Pit Ore	7.5	6.7	8.3	11.3	8.2	13.6	13.9	4.2
Total Colnic Pit	**24.9**	**27.3**	**27.3**	**27.3**	**25.0**	**26.5**	**20.5**	**6.6**

The same data for the Rovina pit is shown in Figure 22.2 and Table 22.3. Mining commences in FY 2031 at the Rovina pit as the Colnic pit is being exhausted. Initially, the Rovina pit is mined at a lower rate as sufficient high-grade ore has been stockpiled at the Colnic pit to feed the plant at its full capacity. From FY 2034 to FY 2039, the Rovina pit is mined at a steady state of approximately 27 Mt/a, with only 9 Mt in FY 2040. The Rovina pit produces 135 Mt of waste and 60 Mt of ore; 36 Mt of the ore is high grade and 23 Mt low grade.

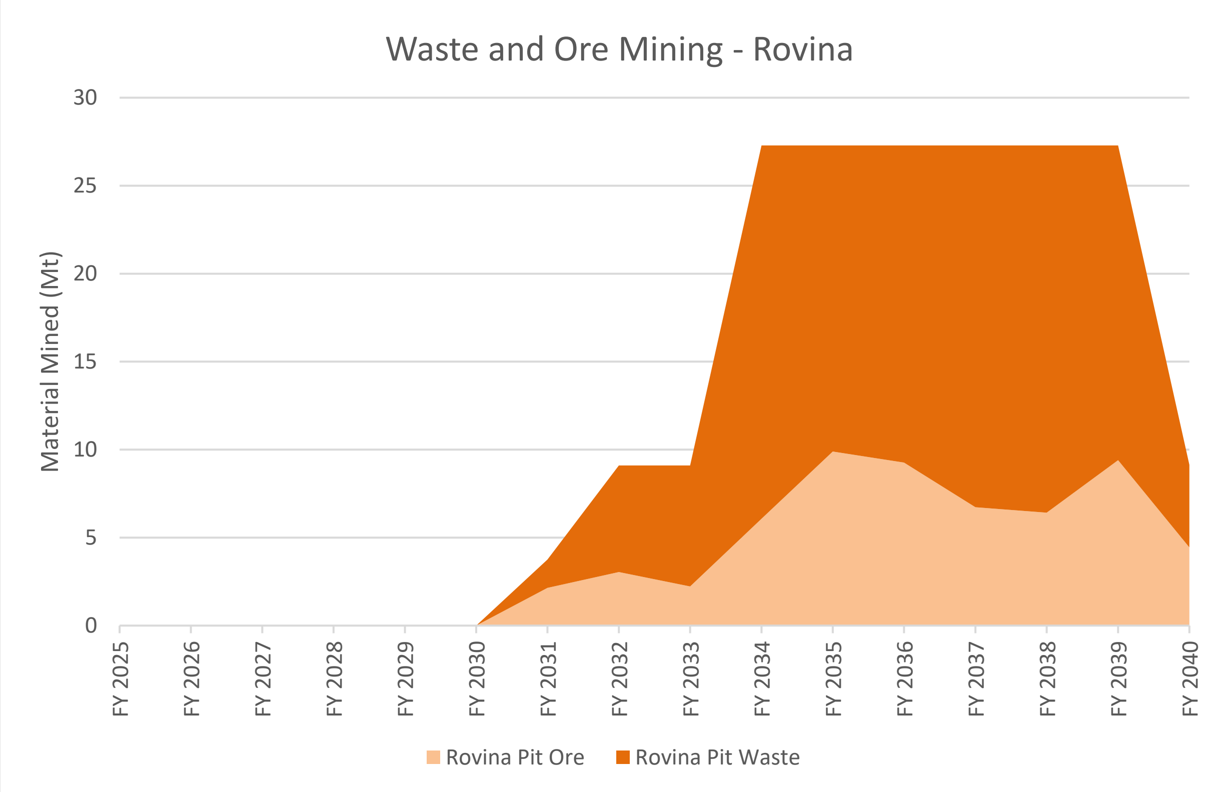


Figure 22.2: Mining at the Rovina Pit

Table 22.3: Rovina Mining Volumes

Volume (Mt)	FY 2031	FY 2032	FY 2033	FY 2034	FY 2035	FY 2036	FY 2037	FY 2038	FY 2039	FY 2040
Rovina Pit Waste	1.6	6.1	6.9	21.2	17.4	18.0	20.6	20.9	17.9	4.7
Rovina Pit Ore	2.1	3.0	2.2	6.1	9.9	9.3	6.7	6.4	9.4	4.4
Total Rovina Pit	**3.8**	**9.1**	**9.1**	**27.3**	**27.3**	**27.3**	**27.3**	**27.3**	**27.3**	**9.1**

The combined mining schedule for the operation is shown in Figure 22.3. The Colnic pit has a more favourable stripping ratio than the Rovina pit, dropping to less than 1.0 in FY 2030 and FY 2031. This allows ore to be stockpiled, and mining can be done at lower levels from FY 2031–2033 while there is sufficient high-grade ore on the stockpiles to feed the plant at full capacity.

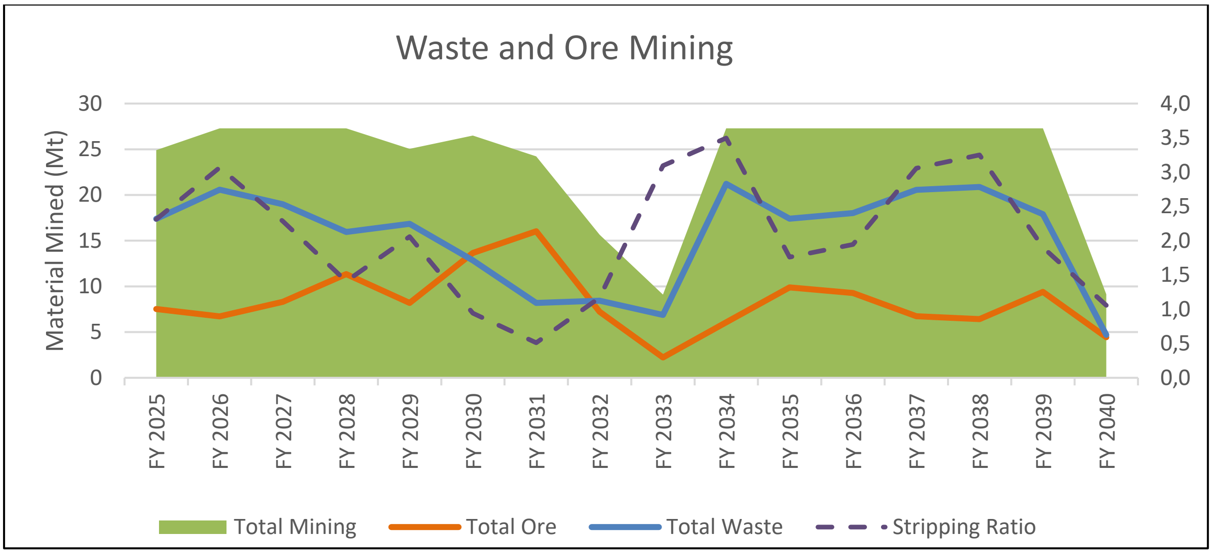


Figure 22.3: Combined Mining Schedule

Table 22.4: Combined Mining Volumes

Item	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032
Total Waste (Mt)	17.4	20.6	19.0	16.0	16.9	12.9	8.2	8.4
Total Ore (Mt)	7.5	6.7	8.3	11.3	8.2	13.6	16.0	7.2
Total Mining (Mt)	**24.9**	**27.3**	**27.3**	**27.3**	**25.0**	**26.5**	**24.2**	**15.7**
Stripping Ratio	*2.3*	*3.1*	*2.3*	*1.4*	*2.1*	*0.9*	*0.5*	*1.2*
Item	**FY 2033**	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**
Total Waste (Mt)	6.9	21.2	17.4	18.0	20.6	20.9	17.9	4.7
Total Ore (Mt)	2.2	6.1	9.9	9.3	6.7	6.4	9.4	4.4
Total Mining (Mt)	**9.1**	**27.3**	**27.3**	**27.3**	**27.3**	**27.3**	**27.3**	**9.1**
Stripping Ratio	*3.1*	*3.5*	*1.8*	*1.9*	*3.1*	*3.3*	*1.9*	*1.1*

22.3.2 Plant Feed and Grades

Plant feed is managed to ensure that the full capacity of 7.2 Mt/a is utilised. High-grade ore is prioritised, with low-grade ore only being processed if there is insufficient high-grade ore available from either ROM production or the stockpiles. Figure 22.4 and Table 22.5 show the plant feed tonnage and grades for FY 2025–2042.

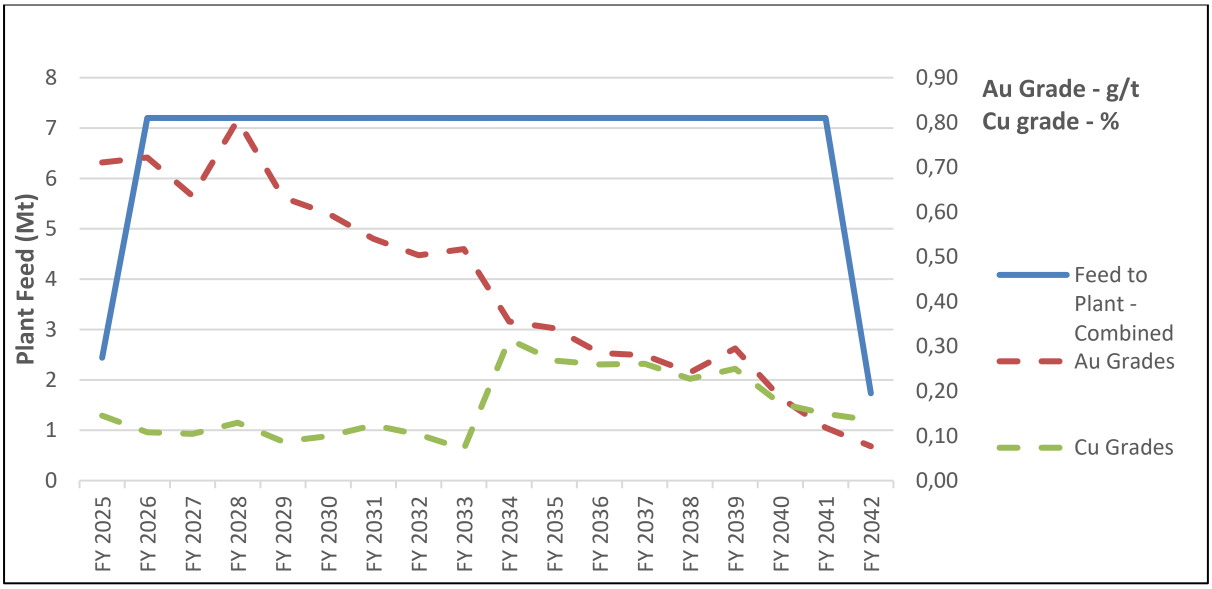


Figure 22.4: Plant Feed Tonnages and Grades

Table 22.5: Plant Feed Tonnages and Grades

Item	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033
Feed to Plant (Mt)	2.4	7.2	7.2	7.2	7.2	7.2	7.2	7.2	7.2
Au Grade (g/t)	0.71	0.72	0.64	0.81	0.63	0.60	0.54	0.50	0.52
Cu Grade (%)	0.14	0.11	0.10	0.13	0.09	0.10	0.12	0.10	0.07
Item	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**	**FY 2041**	**FY 2042**
Feed to Plant (Mt)	7.2	7.2	7.2	7.2	7.2	7.2	7.2	7.2	1.7
Au Grade (g/t)	0.36	0.34	0.29	0.28	0.24	0.30	0.19	0.12	0.08
Cu Grade (%)	0.32	0.27	0.26	0.26	0.23	0.25	0.17	0.15	0.13

The plant is fed ore from only the Colnic pit up to FY 2032, 6.9 Mt, from the Colnic pit and 0.3 Mt from the Rovina pit in FY 2033, and ore from only the Rovina pit from FY 2034–2042. A total of 119 Mt of ore is processed over the LOM.

The difference between the orebodies is clearly shown in the graph by the step changes in gold and copper grades from FY 2033–2034. The Colnic ore has higher gold grades and lower copper grades than the Rovina ore. The average plant feed grades at the Colnic pit are 0.62 g/t Au and 0.10 % Cu, while for the Rovina pit they are 0.26 g/t Au and 0.24 % Cu.

22.3.3 Concentrate Production and Metal Sold

Production from the concentrator plant is based on the yields discussed in Section 17 and the plant feed data above.

The average percentage of gold in the ore recovered to concentrate is 79.7 % (80.0 % for Colnic, 78.9 % for Rovina), while for copper the average is 91.4 % (88.8 % for Colnic, 92.5 % for Rovina).

The amount of concentrate produced, and the gold and copper grades of the concentrate are shown in Figure 22.5 and Table 22.6. Concentrate tonnage is driven by feed grades and is relatively higher for the Rovina ore. The Colnic ore produces an average 55 kt (dry mass) of concentrate per year for a total of 459 kt. The Rovina ore produces an average 75 kt (dry mass) of concentrate per year for a total of 690 kt.

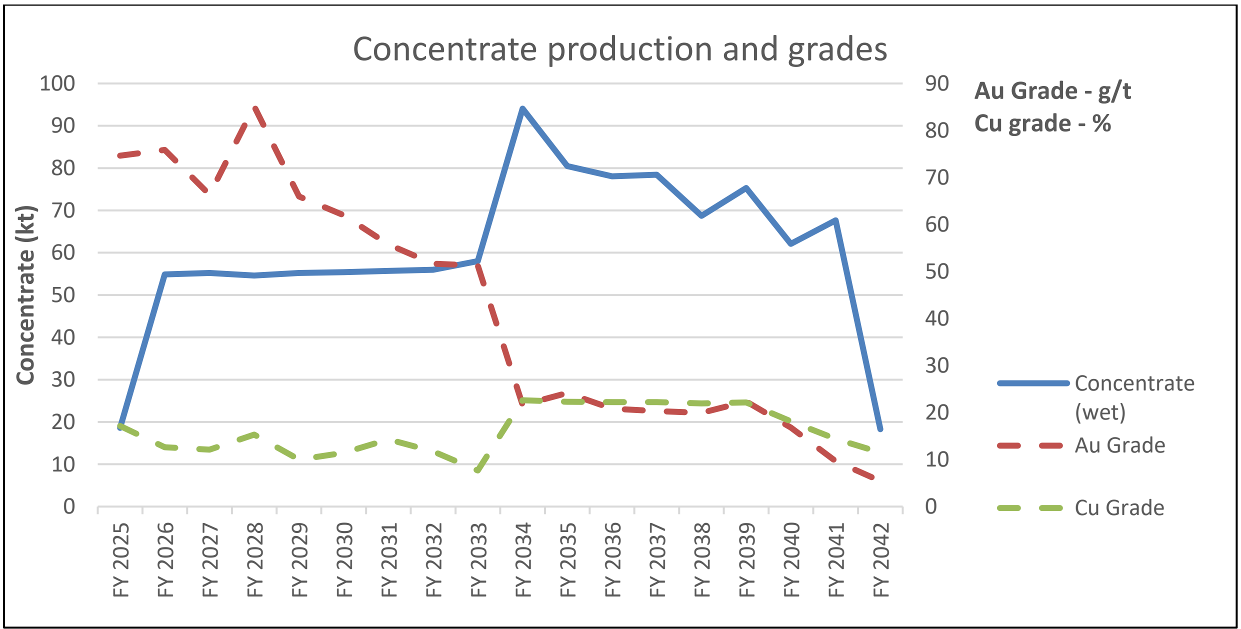


Figure 22.5: Concentrator Output

Table 22.6: Concentrator Output Tonnage and Grades

Item	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033
Concentrate (kt)	20.4	60.3	60.7	60.0	60.7	60.9	61.2	61.5	63.7
Au Grade (g/t)	74.6	75.9	66.3	85.1	66.0	62.0	55.8	51.7	51.2
Cu Grade (%)	17.2	12.6	12.1	15.3	10.0	11.4	14.3	11.7	7.7
Item	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**	**FY 2041**	**FY 2042**
Concentrate (kt)	103.4	88.4	85.8	86.2	75.5	82.7	68.3	74.3	20.1
Au Grade (g/t)	21.6	24.2	20.8	20.3	20.0	22.3	16.8	9.7	5.4
Cu Grade (%)	22.6	22.3	22.2	22.2	22.0	22.2	18.1	14.4	11.5

The concentrate is grossed up to a wet mass such that the moisture content is 9 %. This wet mass of concentrate is transported to the smelter at Bor in Serbia. Smelter payability is assumed at 97.5 % of the contained metal for gold and 96.5 % of the contained metal for copper. Due to the projected quality of the concentrate, no penalties are expected. The resulting metal sales are shown in Figure 22.6 and Table 22.7. Over the LOM, 1.3 Moz of gold is sold, with 932 koz from Colnic (71 % of the project total) and 393 koz from Rovina (29 % of the project total). Copper sales over the LOM total 393 Mlb, with 117 Mlb from Colnic (30 % of the project total) and 125 Mlb from Rovina (70 % of the project total). Converting the copper produced to equivalent gold using the gold price of US$1,550/oz and the copper price of US$3.30/lb, gives 0.9 Moz of gold equivalent, for a total for the project of 2.2 Moz Au gold equivalent.

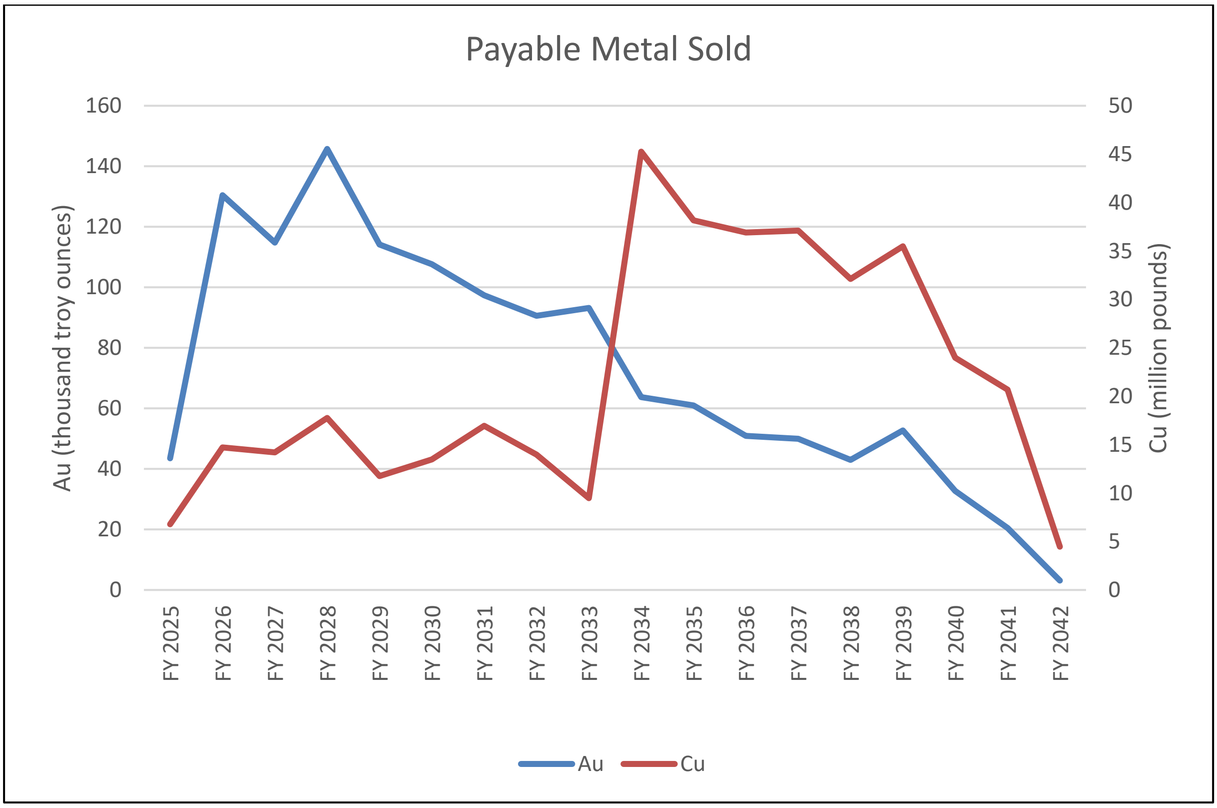


Figure 22.6: Metal Sales Volumes

Table 22.7: Payable Metal Sold

Metal	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033
Gold (koz)	43.5	130.5	114.8	145.7	114.2	107.6	97.4	90.6	93.2
Copper (Mlb)	6.8	14.7	14.2	17.8	11.8	13.5	17.0	14.0	9.5
Metal	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**	**FY 2041**	**FY 2042**
Gold (koz)	63.7	61.0	50.9	49.9	43.0	52.7	32.7	20.5	3.1
Copper (Mlb)	45.3	38.2	36.9	37.1	32.1	35.5	24.0	20.7	4.5

22.4 FINANCIAL RESULTS AND ANALYSIS

22.4.1 Revenue

The gross revenue (before royalties and concentrate transport and treatment costs) totals US$3.34 billion (real 2020 dollars) over the LOM as shown in Figure 22.7 and Table 22.8. Gold contributes US$2.04 billion (61 %), and copper contributes US$1.30 billion (39 %) to the project revenue. The annual average revenue in the steady-state years (FY 2026–2039) is US$214 million, ranging from a high of US$284 million in FY 2028 to a low of US$172 million in FY 2038.

During the Colnic pit operations (up to FY 2033), gold contributes 79 % to the total revenue, while during the Rovina pit operations (from FY 2034), copper is a greater contributor to revenue than gold, providing 61 % of the revenue.

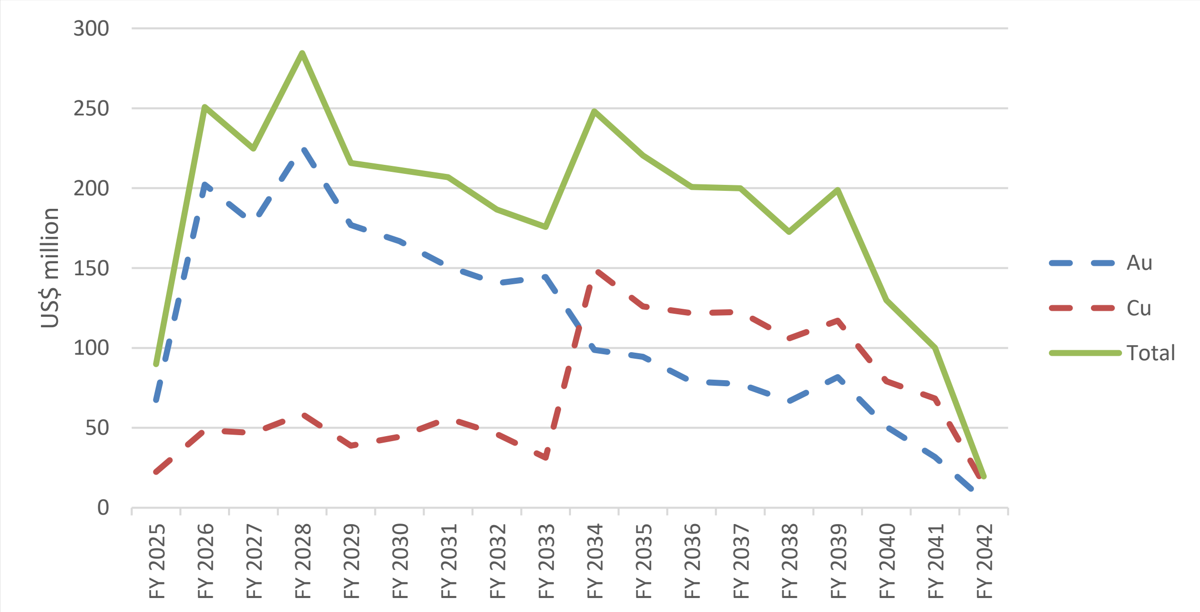


Figure 22.7: Gross Revenue

Table 22.8: Gross Revenue

Revenue (US$ million)	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033
Gold	67.5	202.2	177.9	225.9	176.9	166.8	150.9	140.5	144.4
Copper	22.4	48.6	46.9	58.6	38.8	44.5	55.9	46.1	31.3
Total	**89.9**	**250.8**	**224.8**	**284.5**	**215.7**	**211.3**	**206.8**	**186.6**	**175.7**
Revenue (US$ million)	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**	**FY 2041**	**FY 2042**
Gold	98.7	94.5	78.9	77.4	66.6	81.7	50.7	31.8	4.8
Copper	149.4	125.9	121.8	122.5	106.0	117.1	79.1	68.3	14.8
Total	**248.1**	**220.4**	**200.8**	**199.9**	**172.6**	**198.8**	**129.9**	**100.1**	**19.6**

Selling costs include royalties and concentrate transport and treatment costs. The selling costs are shown in Figure 22.8 and Table 22.9. Royalties are calculated at the rates discussed in Section 22.2.4, and product transport costs are US$41.00/t concentrate for the round trip to Bor. Refining charges are US$6.00/oz of payable gold and US$132.28/t of payable copper. The overall selling costs total US$331 million of the LOM, with a steady-state annual average of US$20.8 million.

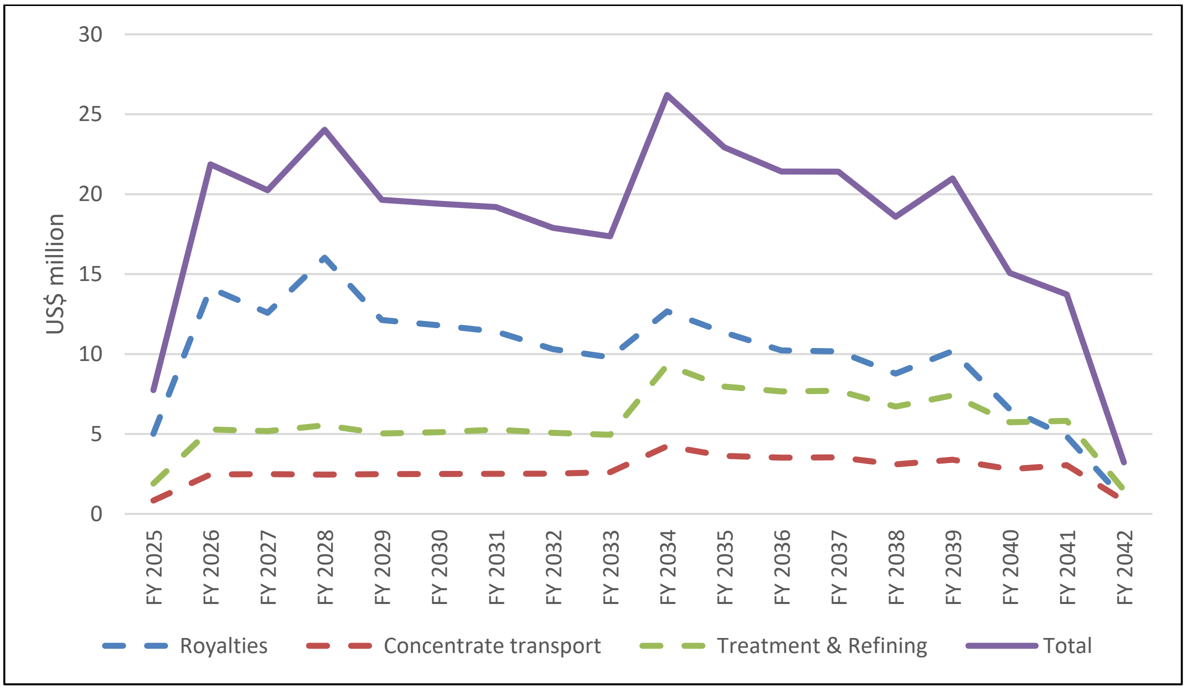


Figure 22.8: Selling Costs

Table 22.9: Selling Costs

Costs (US$ million)	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033
Royalties	5.0	14.1	12.6	16.0	12.1	11.8	11.4	10.3	9.8
Concentrate Transport	0.8	2.5	2.5	2.5	2.5	2.5	2.5	2.5	2.6
Treatment and Refining	1.9	5.3	5.2	5.5	5.0	5.1	5.3	5.1	5.0
Total	**7.7**	**21.9**	**20.2**	**24.0**	**19.6**	**19.4**	**19.2**	**17.9**	**17.4**
Costs (US$ million)	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**	**FY 2041**	**FY 2042**
Royalties	12.7	11.3	10.2	10.2	8.8	10.2	6.5	4.9	0.9
Concentrate Transport	4.2	3.6	3.5	3.5	3.1	3.4	2.8	3.0	0.8
Treatment and Refining	9.3	8.0	7.7	7.7	6.7	7.4	5.7	5.8	1.5
Total	**26.2**	**22.9**	**21.4**	**21.4**	**18.6**	**21.0**	**15.1**	**13.7**	**3.2**

The resulting net revenue is compared to the gross revenue in Figure 22.9. The net revenue totals US$3.01 billion over the LOM, which is 90.1 % of the gross revenue. Selling costs thus amount to 9.9 % of the gross revenue. The annual average net revenue in the steady-state years (FY 2026–2039) is US$193 million, ranging from US$260 million in FY 2028 to US$154 million in FY 2038.

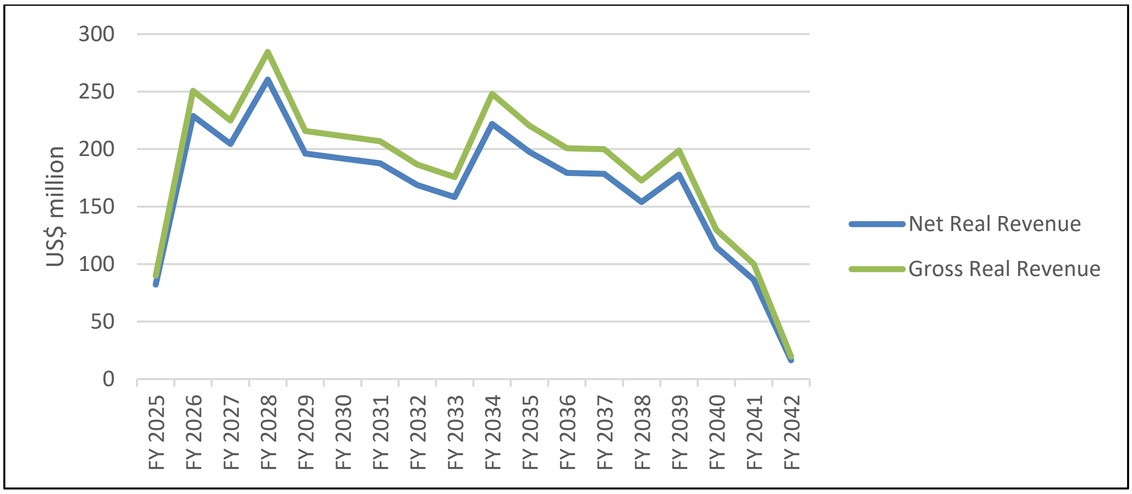


Figure 22.9: Gross and Net Revenue

22.4.2 Capital Expenditure

CAPEX is discussed in detail in Section 21.2. In the financial model, 66.7 % of the establishment capital is assumed to be spent in FY 2024 and 33.3 % in FY 2025. Sustaining capital, primarily related to mobile equipment, is scheduled according to operational requirements. The initial CAPEX is US$399 million and sustaining capital is US$48 million. The CAPEX in each year is shown in Figure 22.10.

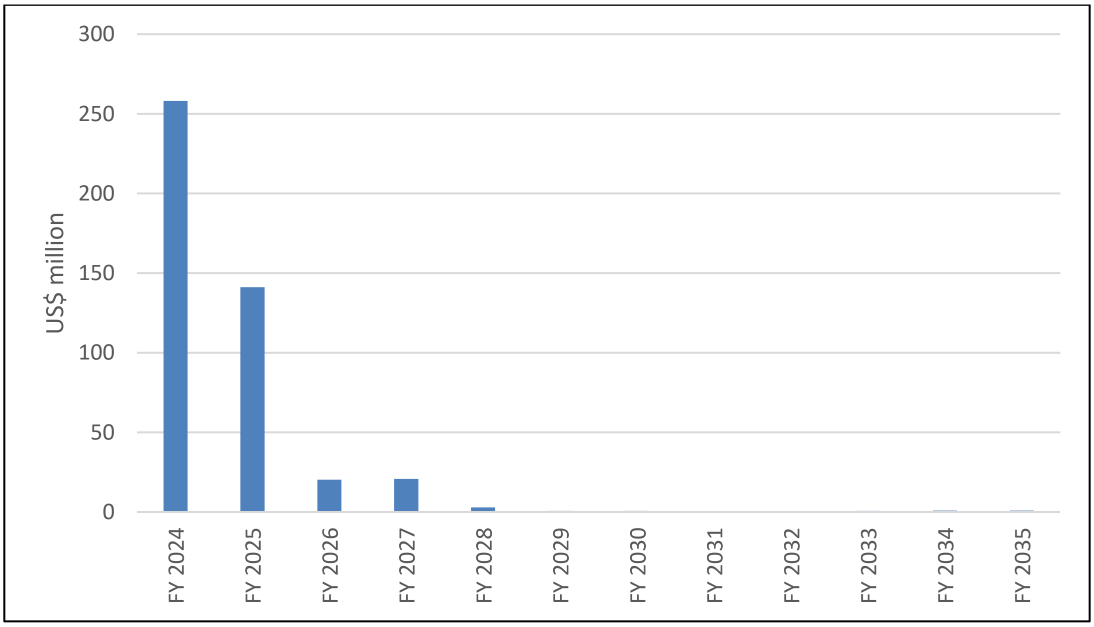


Figure 22.10: Annual CAPEX

22.4.3 Operating Costs

The OPEX assumptions applied in the financial analysis are discussed in detail in Section 21.3. The resulting annual OPEX is shown in Figure 22.11 and Table 22.10.

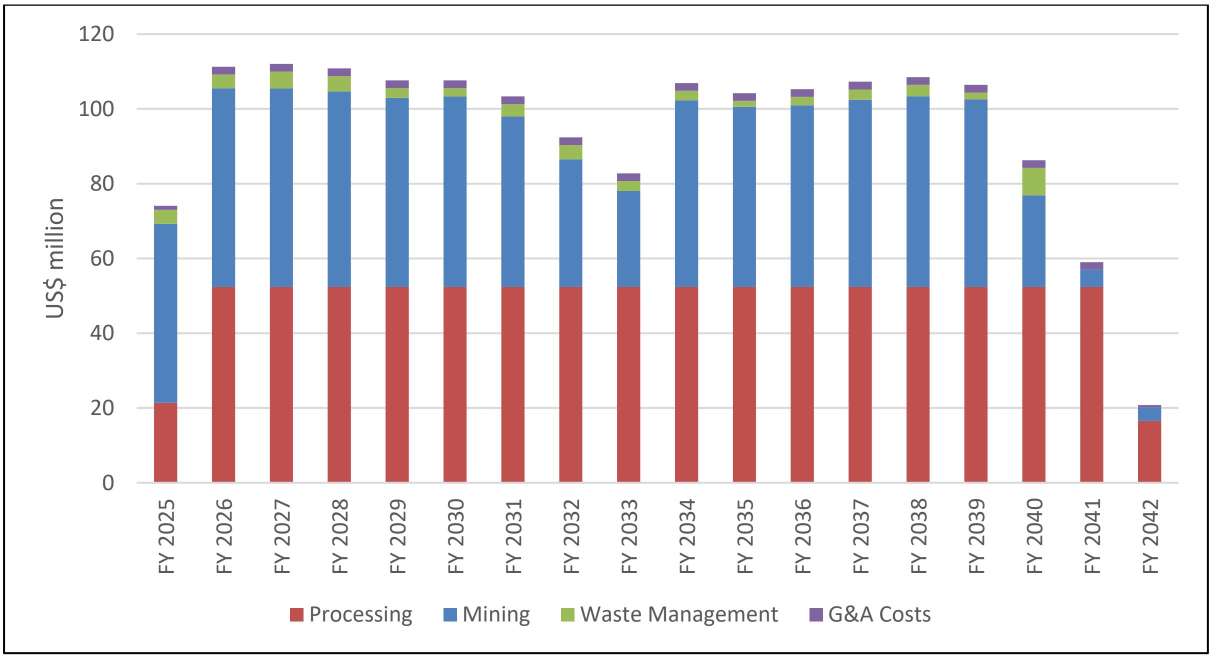


Figure 22.11: Annual OPEX

Table 22.10: Annual OPEX

OPEX (US$ million)	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032	FY 2033
Mining	47.8	53.2	53.1	52.3	50.6	50.9	45.6	34.1	25.7
Processing	21.4	52.4	52.4	52.4	52.4	52.4	52.4	52.4	52.4
Waste Management	3.8	3.7	4.5	4.1	2.6	2.3	3.3	3.8	2.6
G&A	1.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
Total	**74.1**	**111.3**	**112.1**	**110.8**	**107.6**	**107.6**	**103.3**	**92.4**	**82.8**
OPEX (US$ million)	**FY 2034**	**FY 2035**	**FY 2036**	**FY 2037**	**FY 2038**	**FY 2039**	**FY 2040**	**FY 2041**	**FY 2042**
Mining	49.9	48.2	48.6	50.0	51.0	50.1	24.5	4.6	3.7
Processing	52.4	52.4	52.4	52.4	52.4	52.4	52.4	52.4	16.5
Waste Management	2.5	1.6	2.3	2.8	3.0	1.8	7.4	0.0	0.0
G&A	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	0.6
Total	**106.9**	**104.2**	**105.3**	**107.3**	**108.5**	**106.4**	**86.3**	**59.0**	**20.8**

The LOM total OPEX, summarised in Table 22.11, is US$1,707 million, which equates to US$770/oz gold equivalent. Processing costs are the greatest contributor to OPEX, comprising 51.3 %. Mining costs contribute 43.6 %, and waste management and G&A costs combined contribute 5.1 %.

Table 22.11: OPEX Summary

OPEX	LOM Total (US$ million)	Steady-State Average (US$ million/year)	Unit Cost (US$/t ore treated)	Unit Cost (US$/oz Au eq.)	LOM Percentage Contribution (%)
Mining	743.8	47.4	6.23	335.55	43.6
Processing	876.1	52.4	7.34	395.24	51.3
Waste Management	52.2	2.9	0.44	23.57	3.1
G&A	34.6	2.1	0.29	15.62	2.0
Total	**1,706.7**	**104.8**	**14.30**	**769.98**	**100.0**

22.4.4 Project Cash Flow and Valuation

The resulting project cash flows are shown in Figure 22.12 and Table 22.12 and Table 22.13. The first two years (FY 2024–2025) are negative due to the establishment capital expenditure. Cash flow is positive thereafter as full operations commence. The project breaks even on a cash basis in FY 2029, 4.8 years after the start of production. The NPV before tax is US$447 million with an internal rate of return (IRR) of 21.3 %, and the NPV after tax is US$359 million with an IRR of 19.2 %.

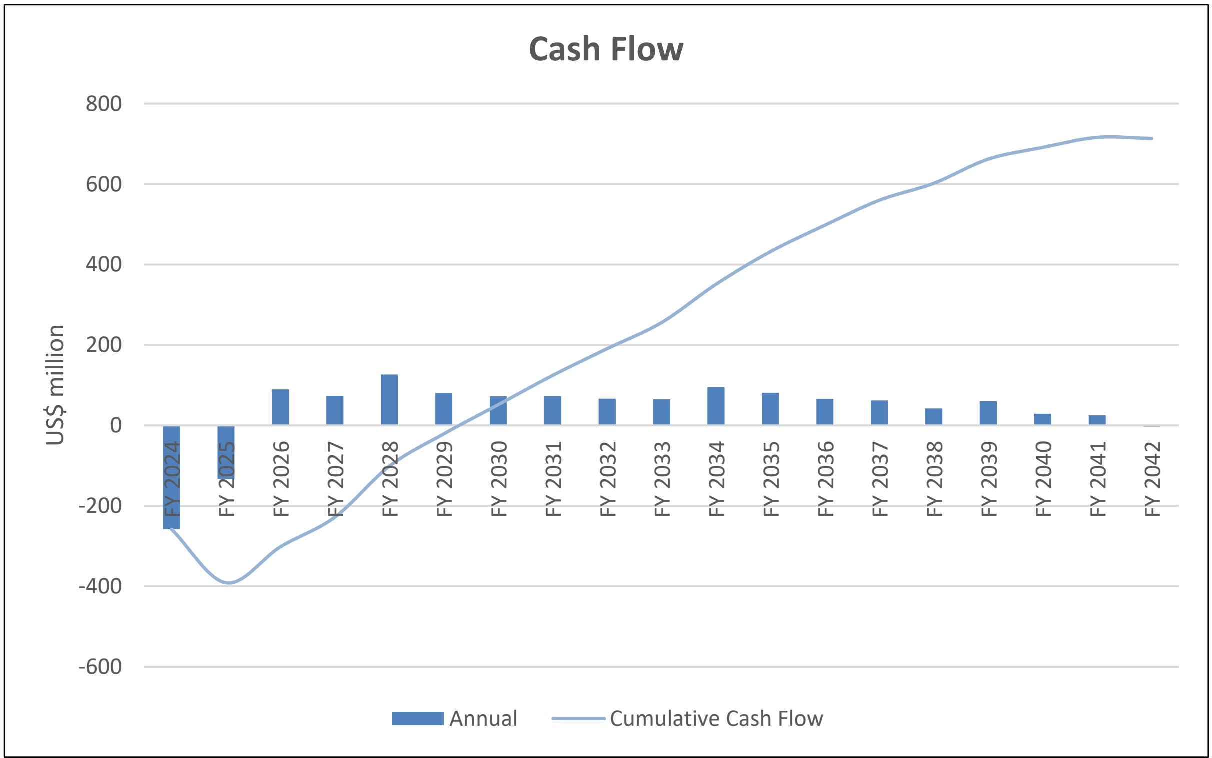


Figure 22.12: Project Cash Flow

Table 22.12: Project LOM Cash Flow (FY 2024–2032)

Cash Flow	Unit	LOM Total	FY 2024	FY 2025	FY 2026	FY 2027	FY 2028	FY 2029	FY 2030	FY 2031	FY 2032
Total Mining	Mt	**380.1**	0.0	24.9	27.3	27.3	27.3	25.0	26.5	24.2	15.7
Ore Produced	Mt	**133.4**	0.0	7.5	6.7	8.3	11.3	8.2	13.6	16.0	7.2
Plant Feed	Mt	**119.4**	0.0	2.4	7.2	7.2	7.2	7.2	7.2	7.2	7.2
Concentrate Produced	kt	**1,194.1**	0.0	20.4	60.3	60.7	60.0	60.7	60.9	61.2	61.5
Gold Produced	koz	**1,315**	0.0	43.5	130.5	114.8	145.7	114.2	107.6	97.4	90.6
Copper Produced	Mlb	**117**	0.0	6.8	14.7	14.2	17.8	11.8	13.5	17.0	14.0
Gross Revenue	**US$ million**	**3,336**	**0.0**	**89.9**	**250.8**	**224.8**	**284.5**	**215.7**	**211.3**	**206.8**	**186.6**
Selling Costs											
Concentrate Transport Costs	US$ million	**−49**	0.0	−0.8	−2.5	−2.5	−2.5	−2.5	−2.5	−2.5	−2.5
Treatment and Refining Charges	US$ million	**−103**	0.0	−1.9	−5.3	−5.2	−5.5	−5.0	−5.1	−5.3	−5.1
Royalties	US$ million	**−179**	0.0	−5.0	−14.1	−12.6	−16.0	−12.1	−11.8	−11.4	−10.3
Net Revenue	**US$ million**	**3,005**	**0.0**	**82.1**	**228.9**	**204.5**	**260.5**	**196.1**	**191.9**	**187.6**	**168.7**
Operating Costs											
Mining	US$ million	**−744**	0.0	−47.8	−53.2	−53.1	−52.3	−50.6	−50.9	−45.6	−34.1
Processing	US$ million	**−876**	0.0	−21.4	−52.4	−52.4	−52.4	−52.4	−52.4	−52.4	−52.4
Waste Management	US$ million	**−52**	0.0	−3.8	−3.7	−4.5	−4.1	−2.6	−2.3	−3.3	−3.8
G&A	US$ million	**−35**	0.0	−1.1	−2.1	−2.1	−2.1	−2.1	−2.1	−2.1	−2.1
Earnings before interest, tax and depreciation	**US$ million**	**1,299**	**0.0**	**8.0**	**117.6**	**92.5**	**149.6**	**88.5**	**84.3**	**84.3**	**76.3**
Capital Expenditure	US$ million	**−447**	−258.1	−141.1	−20.3	−20.7	−2.9	−0.6	−0.6	0.0	−0.2
Working Capital Movement	US$ million	**0**	0.0	0.0	−7.8	1.9	−4.4	4.7	0.3	0.0	0.6
Project Cash Flow Before Tax	**US$ million**	**852**	**−258.1**	**−133.1**	**89.5**	**73.7**	**142.4**	**92.5**	**84.0**	**84.3**	**76.7**
Corporate Income Tax	US$ million	**−139**	0.0	0.0	0.0	0.0	−15.8	−12.1	−11.4	−11.5	−10.2
Project Cash Flow After Tax	**US$ million**	**713**	**−258.1**	**−133.1**	**89.5**	**73.7**	**126.6**	**80.4**	**72.5**	**72.8**	**66.5**

Table 22.13: Project LOM Cash Flow (FY 2033–2042)

Cash Flow	Unit	FY 2033	FY 2034	FY 2035	FY 2036	FY 2037	FY 2038	FY 2039	FY 2040	FY 2041	FY 2042
Total mining	Mt	9.1	27.3	27.3	27.3	27.3	27.3	27.3	9.1	0.0	0.0
Ore produced	Mt	2.2	6.1	9.9	9.3	6.7	6.4	9.4	4.4	0.0	0.0
Plant Feed	Mt	7.2	7.2	7.2	7.2	7.2	7.2	7.2	7.2	7.2	1.7
Concentrate Produced	kt	63.7	103.4	88.4	85.8	86.2	75.5	82.7	68.3	74.3	20.1
Gold Produced	koz	93.2	63.7	61.0	50.9	49.9	43.0	52.7	32.7	20.5	3.1
Copper Produced	Mlb	9.5	45.3	38.2	36.9	37.1	32.1	35.5	24.0	20.7	4.5
Gross Revenue	**US$ million**	**175.7**	**248.1**	**220.4**	**200.8**	**199.9**	**172.6**	**198.8**	**129.9**	**100.1**	**19.6**
Selling Costs											
Concentrate Transport Costs	US$ million	−2.6	−4.2	−3.6	−3.5	−3.5	−3.1	−3.4	−2.8	−3.0	−0.8
Treatment and Refining Charges	US$ million	−5.0	−9.3	−8.0	−7.7	−7.7	−6.7	−7.4	−5.7	−5.8	−1.5
Royalties	US$ million	−9.8	−12.7	−11.3	−10.2	−10.2	−8.8	−10.2	−6.5	−4.9	−0.9
Net Revenue	**US$ million**	**158.4**	**221.9**	**197.5**	**179.3**	**178.5**	**154.0**	**177.9**	**114.8**	**86.4**	**16.4**
Operating Costs											
Mining	US$ million	−25.7	−49.9	−48.2	−48.6	−50.0	−51.0	−50.1	−24.5	−4.6	−3.7
Processing	US$ million	−52.4	−52.4	−52.4	−52.4	−52.4	−52.4	−52.4	−52.4	−52.4	−16.5
Waste Management	US$ million	−2.6	−2.5	−1.6	−2.3	−2.8	−3.0	−1.8	−7.4	0.0	0.0
G&A	US$ million	−2.1	−2.1	−2.1	−2.1	−2.1	−2.1	−2.1	−2.1	−2.1	−0.6
Earnings before interest, tax and depreciation	**US$ million**	**75.6**	**115.0**	**93.3**	**74.0**	**71.2**	**45.5**	**71.4**	**28.5**	**27.3**	**−4.4**
Capital Expenditure	US$ million	−0.7	−0.8	−0.8	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Working Capital Movement	US$ million	0.0	−2.5	1.5	1.5	0.2	1.9	−1.9	3.0	0.0	1.7
Project Cash Flow Before Tax	**US$ million**	**74.9**	**111.6**	**94.0**	**75.5**	**71.5**	**47.4**	**69.5**	**31.5**	**27.3**	**−2.7**
Corporate Income Tax	US$ million	−10.0	−16.3	−12.8	−9.8	−9.4	−5.3	−9.4	−2.5	−2.4	0.0
Project Cash Flow After Tax	**US$ million**	**64.9**	**95.3**	**81.1**	**65.7**	**62.1**	**42.1**	**60.1**	**29.0**	**25.0**	**−2.7**

22.4.5 Sensitivity Analysis

The sensitivity of the project NPV (at the base discount rate of 5 %) to changes in key variables is shown in Table 22.14 and Figure 22.13.

Table 22.14: NPV Sensitivity

NPV (US$ million)	−20 %	−10 %	0 %	10 %	20 %
Gold Price	134.3	246.9	**359.3**	471.7	583.9
Copper Price	236.5	297.9	**359.3**	420.7	482.1
CAPEX	433.9	396.6	**359.3**	322.0	284.6
OPEX	554.4	456.9	**359.3**	261.7	164.0
Discount Rate	414.8	386.3	**359.3**	333.8	309.6
Treatment costs and refining charges	369.4	364.3	**359.3**	354.3	349.3
Transport	364.1	361.7	**359.3**	356.9	354.5
Diesel	412.2	385.7	**359.3**	332.9	306.5
Electricity	373.2	366.3	**359.3**	352.3	345.4

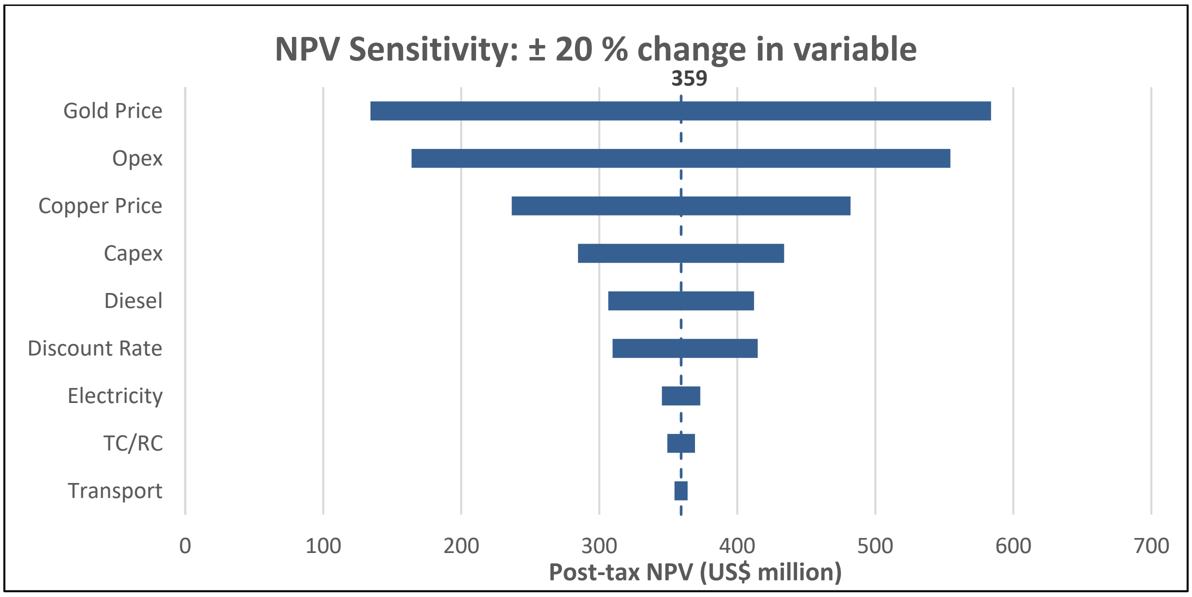


Figure 22.13: NPV Sensitivity

Figure 22.13 shows that the project NPV is most sensitive to the gold price. A 20 % reduction in the gold price to US$1,240/oz reduces the project value to US$134 million. The project is, however, very robust in the face of changing gold prices, with a 32 % reduction to US$1,050/oz required to reduce the NPV to 0. The project is less sensitive to copper price reductions, requiring a 57 % reduction to US$1.41/lb to reduce the NPV to 0. OPEX is the second most significant sensitivity driver. An increase in OPEX of 37 % will bring the NPV to 0.

The sensitivity of the NPV to the discount rate is shown in Table 22.15. The NPV is robust in the face of changes in the discount rate, remaining well positive even if the discount rate is doubled to 10 %.

Table 22.15: NPV Sensitivity to Discount Rate

Discount Rate	2.5 %	5.0 %	7.5 %	10.0 %
NPV (US$ million)	510.6	**359.3**	244.4	156.0

The sensitivity of the project IRR to changes in key variables is shown in Table 22.16 and Figure 22.14.

Table 22.16: IRR Sensitivity

IRR (%)	−20 %	−10 %	0 %	10 %	20 %
Gold Price	11.7	15.5	**19.2**	22.8	26.3
Copper Price	16.1	17.7	**19.2**	20.6	21.9
CAPEX	24.5	21.6	**19.2**	17.2	15.4
OPEX	24.6	21.9	**19.2**	16.3	13.2
TC/RC	19.5	19.3	**19.2**	19.1	18.9
Transport	19.3	19.3	**19.2**	19.1	19.1
Diesel	20.6	19.9	**19.2**	18.5	17.7
Electricity	19.6	19.4	**19.2**	19.0	18.8

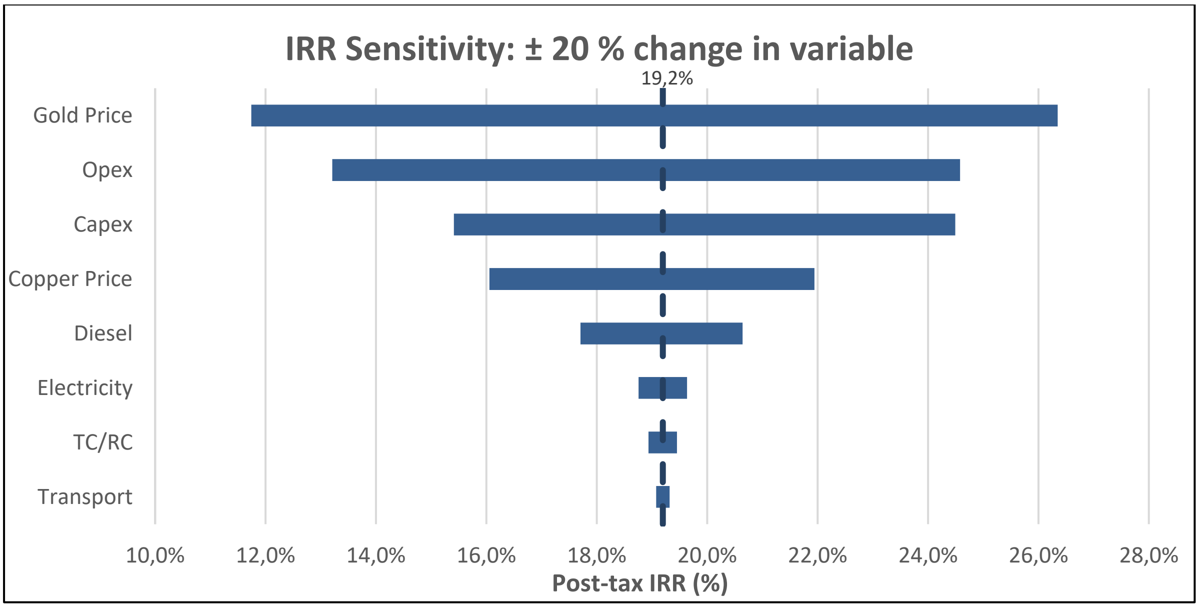


Figure 22.14: IRR Sensitivity

Gold price is once again the most sensitive driver, with a 20 % reduction reducing the IRR by 7.5 % to 11.7 %. The copper price is the fourth most significant driver of the IRR, with a 20 % reduction reducing the IRR by 3.1 % to 16.1 %. OPEX is again second with a 20 % increase reducing the IRR by 6.0 % to 13.2 %.

22.5 CONCLUSION

The financial analysis of the RVP was done by building a discounted cash flow model, using the market price, production, CAPEX and OPEX data from the DFS. The project has a strongly positive post-tax NPV of US$359 million with an IRR of 19.2 %. The project value is robust in the face of changes to key input variables and can withstand a reduction of up to 32 % in the gold price (to US$1,050/oz) before the NPV becomes negative.

23 ADJACENT PROPERTIES

The information presented in this section is publicly available information. No means of verifying this information was made available to the authors. ESM cautioned that this information is not necessarily indicative of the mineralisation on the RVP.

One deposit within the Golden Quadrilateral in Romania, in the immediate vicinity of the Property, is in an advanced-stage exploration and permitting phase. This is the Certej deposit which is now held by Eldorado Gold (80.5 %) and Minvest (19.5 %), The Certej deposit is located 15 km southeast of Ciresata. In addition, recent drilling completed by Eldorado Gold on its 100 % owned Bolcana-Troita Exploration Licence has resulted in a brownfield discovery and an initial resource estimate. The Bolcana-Troita Exploration Licence area is located 11 km south of the Ciresata deposit.

The historical Sacaramb deposit (also known as Nagyag or Szekeremb) is contained within the European Goldfields' Certej property while the Valea Morii deposit is located 2.5 km west of the Ciresata deposit.

23.1 CERTEJ

Certej is an epithermal gold/silver deposit. Mineralisation occupies a sub-horizontal pipe-like zone, located across, and deformed around, the Baiaga Andesite. It is localized localised by an east–west trending dilational jog, locally offset by later north–south faulting. Styles of mineralisation recognised in the deposit to date include disseminated and breccia-hosted gold-silver, and vein-hosted base metal (± gold-silver) mineralisation. There are three main mineralised zones, Hondol, Baiaga, and Dealul Grozii (Warries et al., 2006).

As of 30 September 2020, Eldorado Gold reported a Proven and Probable Reserve of 44.3 Mt grading 1.69 g/t gold, 11 g/t silver for 2.4 Moz gold, and 15.5 Moz silver (Eldorado Gold website). Eldorado Gold has finalised a feasibility study and is well under way to the final permitting of the project. The mine life is stated at 15 years. SENET has commented that the deposit type and style of mineralisation is different from ESM's RVP. The QP has been unable to verify the information reported by Eldorado Gold or that the information is not necessarily indicative of the mineralisation on the property that is the subject of this Technical Report.

23.2 BOLCANA

The Bolcana deposit is a gold-rich copper porphyry with geologic similarities to ESM's Rovina Valley porphyry deposits. On 26 November 2018, Eldorado Gold announced a maiden resource estimate for Bolcana (https://www.eldoradogold.com/news-and-media/news-releases/press-release-details/2018/Eldorado-Gold-Releases-Updated-Reserve-and-Resource-Statement/default.aspx).

The maiden resource estimate for the Bolcana gold-copper porphyry project in Romania is based on 98 diamond drillholes totalling over 61,995 m completed by Eldorado mainly in 2017/2018, and 17 drillholes totalling 4,609 m and 4,224 m of underground channel samples collected by European Goldfields from 2002 to 2004.

The Bolcana porphyry system includes three shallow mineralised zones (north, central and south) over a strike extent of greater than 1 km, which coalesce at depth into a north-plunging high-grade mineralised core. The highest grades coincide with late-stage gold-rich dikes that are superimposed on an earlier gold-copper porphyry that intrudes broadly coeval breccias and andesitic country rocks. Stockwork veins and disseminations of chalcopyrite and subordinate bornite are hosted in the dikes and associated breccias. Alteration includes potassic assemblages (biotite-feldspar-magnetite) and a shallow pyrite-white mica-clay domain, with a magnetite-albite-chlorite-epidote overprint related to the late-stage dikes.

The Bolcana resource is classified as an Inferred Mineral Resource and is based on an open-pit and an underground component. The open-pit portion, which contains just over half of the inferred resources, is constrained by a conceptual pit design with an average depth of approximately 720 m. All the resources outside this pit shell were available to be classified as underground inferred resources. Inferred open-pit and underground resources total 381 Mt at 0.58 g/t gold and 0.18 % copper, containing 6.5 Moz gold.

Preliminary rougher flotation testing on a suite of nine samples representative of the alteration and mineralogical variability in the deposit achieved recoveries of up to 90 % for copper and 86 % for gold.

23.3 SACARAMB

The historical Sacaramb deposit (also known as Nagyag or Szekeremb) is also contained within the European Goldfields' Certej property. When production ceased in 1935, approximately 85 t of combined silver and gold was reported to have been produced (Patrick and Jackson, 2004).

Epithermal gold-silver telluride mineralisation is developed in Neogene andesite flows and breccias. Gold occurs as disseminations within breccias and in mineralised linear alteration zones associated with fracturing and traditionally described as veins. Bonanza high-grade zones at vein junctions formed pipe-like zones that have been mined to surface. Over 230 individual mineralised veins are known and have been traced along strike for up to 2,000 m, and down dip for 1,000 m (Patrick and Jackson, 2004).

European Goldfields drill-tested the deposit in 2002 and partially sampled seven levels of historical workings as part of an assessment of the open pit potential of the deposit (Patrick and Jackson, 2004). In 2012, Eldorado Gold acquired European Goldfields.

23.4 VALEA MORII

The Valea Morii deposit comprises earlier porphyry copper-gold and later epithermal gold-silver mineralisation associated with a shallow subvolcanic body of andesitic to dioritic composition (Grancea et al., 2001). The top of the porphyry copper system is crosscut by low-sulphidation epithermal veins, with a paragenetic sequence from early veining related to propylitic alteration, followed by a second composite stage, characterised by the emplacement of barren, centimetre-thick, quartz-rich veins followed by millimetre-thick quartz-rich copper-gold mineralised veins. This porphyry-related stockwork-style mineralisation is partly overprinted by quartz ± calcite ± barite epithermal veins, hosting gold-silver mineralisation (Rosetti et al., 1999; Andre-Meyer et al., 2001). Valea Morii is located 2.5 km west of the Ciresata deposit.

24 OTHER RELEVANT DATA AND INFORMATION

24.1 PROJECT SCHEDULE

The project implementation schedule has been compiled to ensure that the engineering, procurement and construction management activities are aligned for successful project execution.

24.1.1 Work Packages

ESM will appoint an EPCM Contractor to execute or manage and oversee the execution (by appointed EPCM/EPC/turnkey subcontractors, who will report to and be managed by the EPCM Contractor) of the following work packages:

- Detailed surveys
- Geotechnical investigations
- Road designs and construction
- Power supply from the main grid
- Raw water supply including catchment dams and tunnels
- Mine industrial area infrastructure
- Process plant construction and on-site infrastructure
- Waste facility predevelopment and drainage
- Mining operations – Colnic pit predevelopment
- Process plant commissioning and ramp-up

The following packages will be owner operated and managed directly by ESM, who will execute the predevelopment requirements and progress these packages into an operational environment:

- Mining operation including pre-strip
- WMF including predevelopment

24.1.2 Project Milestones

The project schedule has been constrained around key dates for permitting provided by ESM. The most significant of these is the construction permit date, which is scheduled for May 2023 and will allow the construction works within the plant's permitted boundary to begin. Therefore, the schedule allows for the detailed design and engineering, along with the development of off-site infrastructure, to be managed so that site works can start immediately on receipt of the construction permit.

The project schedule assumes that there will be a seamless advancement between the various phases of the project evolution. It is also recognised that this is a moderately aggressive schedule and that it will require diligent progress monitoring and coordination of all the parties involved. The project milestones are given in Table 24.1.

Table 24.1: Project Execution Milestones

Project Milestone	Project Month
Commence early works design	Month −13
Commence road designs	Month −13
Commence geotechnical site investigations	Month −10
Commence main power line design	Month −11
Commence raw water supply design	Month −13
Complete process plant access road and Merișor road designs	Month −4
Commence process plant access road and Merișor road construction	Month −4
Complete geotechnical site investigations and report	Month −1
Complete initial process design	Month −6
Place orders for long-lead delivery items (e.g. ball and SAG mills)	Month 1
Commence process plant, mining, and waste facility detailed designs	Month 1
Commence procurement and contracts administration activities	Month 1
Complete main power line design	Month 2
Commence main 36 km buried power line construction	Month 3
Complete process plant, mining, perimeter road and waste facility detailed designs	Month 4
Complete process plant access road and Merișor road construction	Month 20
Commence construction activities, i.e. waste facility including perimeter roads, dam walls and tunnels, mining terracing and MIA interim infrastructure, and process plant infrastructure	Month 14
Complete main 36 km buried power line construction	Month 14
Commence main power line switchyard construction	Month 25
Complete dam walls, water tunnels and water diversion channels construction	Month 20
Complete perimeter road (Colnic to MIA) and waste facility underdrains and pre-strip construction activities	Month 20
Commence MIA and fuel depot construction including Colnic pit predevelopment	Month 14
Complete procurement, manufacture, and equipment deliveries to site	Month 24
Complete process plant, MIA, mining predevelopment and main power line switchyard construction	Month 32
Commence process plant commissioning	Month 33
Complete process plant commissioning	Month 36
Ramp-up to 100 % of nameplate production	Month 37

24.1.3 Long-Lead Equipment

Placing the purchase orders for the long-lead equipment is crucial not only to ensure that the equipment is on site in time to allow for a seamless construction sequence and a successful project execution but also to obtain the certified information from the supply vendors on their equipment to complete the detailed engineering phase of the project. Long-lead equipment has been identified based on a manufacturing time frame of more than 5 months. The longest equipment manufacturing time quoted for the RVP at the time of tendering was 15 months.

The key long-lead equipment for the project is as follows:

- Ball and SAG Mills
- Regrind Mill
- Gyratory and Pebble Crushers
- Apron and Pan Feeders
- Flotation Cells
- Thickeners
- Filter Press Plants
- Cyclone Cluster
- Pumps (Cyclone Feed Pumps and Overland Pumps)
- MV Switchgear
- Mining Fleet

24.1.4 Rain Delay

The rainy season for the geographical region of the project is from April to August. It is important to note that no rain delay has been allowed for in the project execution schedule. The completion of the bulk earthworks and the predevelopment of the WMF could be impacted by the timing of the construction permit being planned for receipt at the start of the rainy season.

The summarised project schedule is shown in Figure 24.1.

SP0829 - Rovina Valley Project
Execution Schedule
Rolled-up Summary

ID	Task Name	Duration	Start	Finish
1	**SP0829 - Rovina Valley Project - Execution Schedule**	**1047 days**	**2021/04/01**	**2025/05/30**
44	Early Works Design & Construction	690 days	2021/04/01	2023/12/29
260	Waste Management Facility Design & Development	815 days	2021/10/06	2024/11/19
296	Mining Design & Development	497 days	2023/01/09	2024/12/13
342	Detail Design (Incl. De-Risking)	253 days	2022/05/04	2023/05/09
1552	Procurement Phase	497 days	2022/05/04	2024/04/25
2479	Construction	557 days	2023/03/01	2024/12/20
3229	Pre-Commissioning	25 days	2024/11/11	2024/12/16
3234	Commissioning	88 days	2025/01/06	2025/04/17
3246	Ramp-up and Optimization	52 days	2025/04/01	2025/05/30

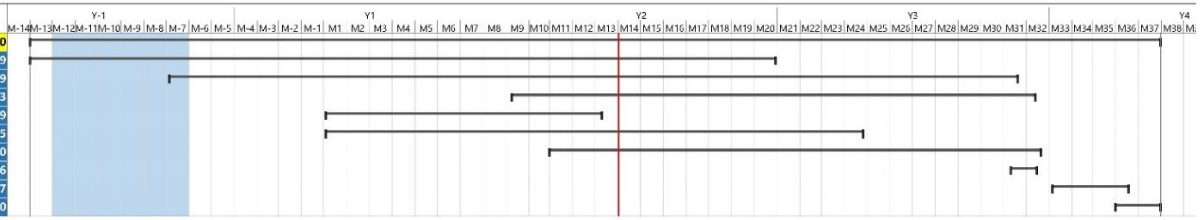


Figure 24.1: Project Schedule Summary

24.2 RISKS AND OPPORTUNITIES

24.2.1 Hazard and Operability Studies

Hazard and operability (HAZOP) studies are used to identify potential hazards in a system and to determine where operability problems could be encountered. The HAZOP process promotes the undertaking of HAZOP 1 and HAZOP 2 studies at the process development stage and process and project definition stage, respectively, of a project. Further HAZOP studies are then undertaken from the project design stage through to operation. In total, there are six different HAZOP stages to progress through over the life of a project from start to full operation.

During the March RVP DFS development, HAZOP 1 and HAZOP 2 studies were performed with an external facilitator, ESM personnel and various consultants.

The key elements of focus during a HAZOP 1 study are as follows:

- Process Description
- Hazard and Operational Experience
- Inherent SHE Protection
- Materials Handling Hazards
- Environmental Statement
- Environmental Objectives
- General Pollution
- Environmental Impact
- Health and Toxicology
- Site Selection and Layout
- Consultation with External Authorities
- SHE Criteria
- Design Guidelines and Codes
- Control Philosophy
- Organisational and Human Requirements
- Emergency Facilities
- Security Aspects
- Further Studies
- Conclusion

The key elements of focus during a HAZOP 2 study are as follows:

- Major and Significant Chemical Hazards
- Physical Hazards
- Process Pollution
- Spills
- Noise
- Visual Impact
- Unavailability
- Security Breach
- Financial Impacts

24.2.2 Project Risk Assessment

The purpose of conducting the project risk assessment for the RVP was to identify, assess and rank the risks that could affect the project during its execution and subsequent operation.

The qualitative risk assessment was performed with an external facilitator to ensure a broad view on the potential project risks. The technical and management disciplines within the project group were represented by the stakeholders' contribution to the risk register development.

The risk register was developed from first principles and evaluated the pre-control probability, frequency, and impact, as well as the post-control probability, frequency, and impact, of each risk.

Control actions were identified and documented for each item. During the assessment of the risks and definition of risk control strategies, the project stakeholders focused on identifying activities that would reduce the impact and/or probability of the risks. To test the expected effectiveness of these control strategies, the pre- and post-control probability and impact of each risk were assessed.

Identified risk weights were characterised as per the 5 × 5 Risk Matrix shown in Table 24.2.

Table 24.2: 5 × 5 Risk Matrix Characterisation Table

Probability/ Risk Level	1 – Insignificant	2 – Minor	3 – Moderate	4 – High	5 – Major
5 – Almost Certain > 90 %	11 (Medium)	16 (Significant)	20 (Significant)	23 (High)	25 (High)
4 – Likely 60 % to 90 %	7 (Medium)	12 (Medium)	17 (Significant)	21 (High)	24 (High)
3 – Possible 40 % to 60 %	4 (Low)	8 (Medium)	13 (Significant)	18 (Significant)	22 (High)
2 – Unlikely 20 % to 40 %	2 (Low)	5 (Low)	9 (Medium)	14 (Significant)	19 (Significant)
1 – Rare < 20 %	1 (Low)	3 (Low)	6 (Medium)	10 (Medium)	15 (Significant)

24.2.3 Risk Assessment Results

A total of 26 risks and 6 opportunities were identified and analysed in this assessment.

24.2.3.1 Pre-Control and Post-Control Weights

Risks and opportunities are listed in Table 24.3 and Table 24.4, respectively, in terms of pre-control and post-control weights. Operational risks can sometimes have significant impacts on the project business case, and the project team should consider modelling sensitivities to these risks in the financial model.

Table 24.3: Risks

No.	Risk Name	Risk Realisation	Pre-Control Risk Weight	Post-Control Risk Weight
R01	Community and NGO Challenge to Mine	Project	25	24
R02	Environmental Permits	Project	13	9
R03	Land Acquisition	Project	13	1
R04	Geology Yield	Operations	6	6
R06	Slope Angles	Operations	13	8
R08	Colnic Pit Water Inflow	Operations	25	10
R09	Mine Operations	Operations	21	13
R10	Consumables Supply	Operations	9	9
R11	Plant Recovery	Operations	13	13
R13	Final Concentrate Moisture Content	Operations	13	13
R15	Moisture	Operations	13	9
R16	Hardness	Operations	18	18
R17	Equipment Selection	Project	9	6
R18	Contamination	Operations	13	8
R19	Catastrophic Failure of Embankment	Operations	15	10
R20	Forest/Vegetation Fire Hazard	Operations	18	18
R21	Pandemics	Project	13	13
R22	Qualified Workforce	Operations	21	18
R23	Project Funding	Project	19	1
R24	Commodity Price, Forex, Escalation	Projects and Operations	9	5
R26	a. CAPEX b. OPEX	Project	6	3
R28	Political Change	Project	18	18
R29	Water Balance	Operations	6	6
R30	Equipment Transport Delays	Project	18	18
R31	Late Design Changes	Project	17	9
R32	Board Approval of Changes	Project	13	8

Table 24.4: Opportunities

No.	Risk Name	Pre-Control Risk Weight	Post-Control Risk Weight
R05	Geology Yield	6	6
R07	Slope Angles	13	13
R12	Plant Recovery	13	13
R14	Final Concentrate Moisture Content	13	13
R25	Commodity Price, Forex, Escalation	9	5
R27	a. CAPEX b. OPEX	6	3

24.2.3.2 Risk Heat Maps for Pre-Control and Post-Control Risks

The heat maps in Table 24.5 and Table 24.6 show the number of risks for each score/weight.

Table 24.5: Pre-Control Risks

Number of Pre-Control risks per category				
		5		1
		4		1
		12	4	
		1	2	
				2

Table 24.6: Post-Control Risks

Number of Post-Control risks per category				
2	2	4	2	
	2	4		
	3	7	5	
				1

24.3 UPSIDE POTENTIAL

A ground magnetic survey, together with soil geochemical sampling, completed by ESM between 2004 and 2006 identified several porphyry targets. One of the key potential targets that is still to be fully tested is the Zdrapti porphyry prospect, shown in Figure 24.2. The Zdrapti prospect lies approximately 2.5 km northwest from the Colnic deposit and approximately 2.0 km southwest from the Rovina deposit. ESM drilled eleven reconnaissance holes totalling 2,671 m to test the sub-cropping potassic alteration. Zdrapti remains a prospect with no current mineral resource estimate.

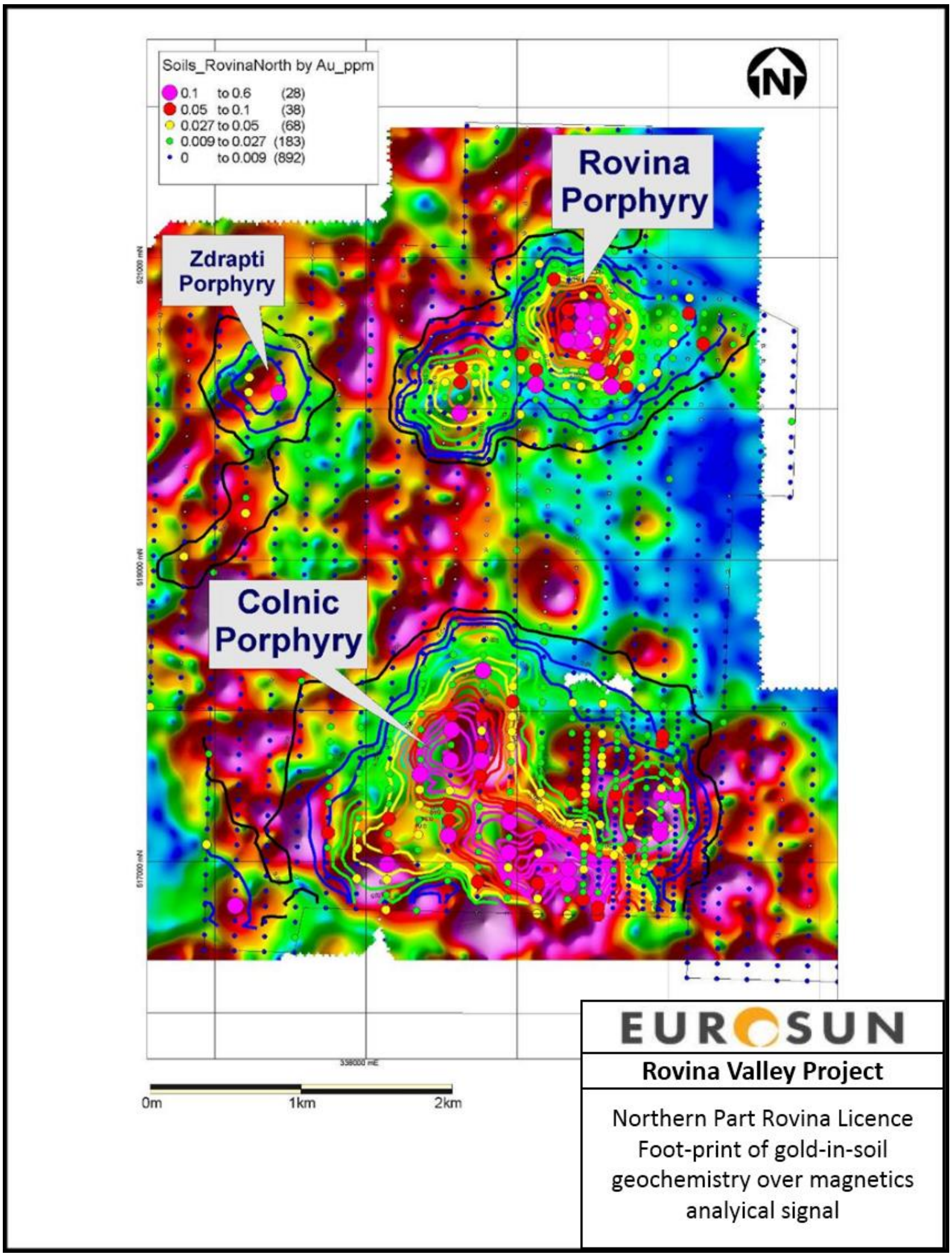


Figure 24.2: Ground Magnetic Survey and Soil Geochemistry in Licence Area

24.3.1 Colnic Pit Upside Potential

The Colnic pit optimisation, design and production schedule used a preliminary BFA of 65°. Post the updated geotechnical investigation, the rock mass strengths showed significant improvement, and an improvement in the BFA has been recommended for the Colnic Pit.

The final summary results for most of the Colnic pit BFA increased from 65° to 70° (see Figure 24.3).

The Rovina Pit design BFA parameters did not change.

For more details, refer to the following:

- Klohn Crippen Berger, 2021. Rovina Valley Project – Feasiblity Study – Geotechnical, Hydrotechnical, and Hydrogeological Report.

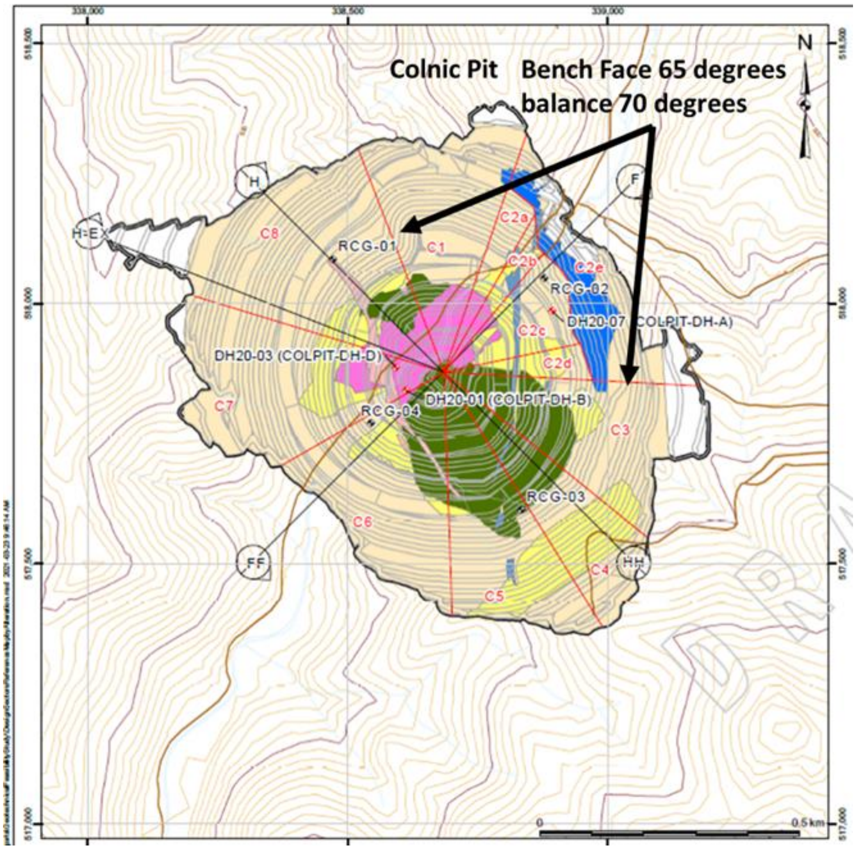


Figure 24.3: Colnic Pit BFA Outcome Summary

25 INTERPRETATION AND CONCLUSIONS

25.1 MINERAL RESOURCE STATEMENT

This March 2021 mineral resource estimate for Rovina and Colnic deposits is updated to reflect current metal prices and updated operating parameters derived during the DFS, and to make the resource current and in conformance with the 2014 CIM Mineral Resource and Mineral Reserve definitions referred to in the NI 43-101, Standards of Disclosure for Mineral Projects. Mr Subramani, BSc Hons (Geology), Pri.Sci.Nat. (400184/06), is the QP for this mineral resource estimate. The mineral resources are constrained to a Lerchs-Grossmann pit shell using different metal equivalent cut-off grades for the Rovina and Colnic deposits. The geological model and mineral resource block models remain unchanged in this current estimate. The mineral resource estimate for Ciresata remains unchanged from February 2019.

Table 25.1 summarises the mineral resource estimates for the Rovina and Colnic deposits, stated above a 0.25 % Cu equivalent grade cut-off for the Rovina deposit, and above a 0.35 g/t Au equivalent grade cut-off for the Colnic deposit. The total Measured mineral resources for the Rovina and Colnic deposits amount to 62.2 Mt grading at 0.49 g/t Au and 0.21 % Cu, containing 0.99 Moz Au and 287 Mlb Cu; with the Au equivalent grading of 0.79 g/t. The total Indicated mineral resources for the Rovina and Colnic deposits amount to an additional 175.6 Mt grading at 0.39 g/t Au and 0.15 % Cu, containing 2.19 Moz Au and 589 Mlb Cu, with the Au equivalent grading of 0.60 g/t.

Table 25.1: 2021 Mineral Resource Estimate – Rovina and Colnic Deposits

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Colnic	Measured	29.1	0.65	0.12	0.61	74	0.81	0.76
	Indicated	97.5	0.49	0.10	1.53	210	0.62	1.96
	Inferred	1.6	0.41	0.09	0.02	3	0.49	0.03
Rovina	Measured	33.1	0.36	0.29	0.38	212	0.77	0.82
	Indicated	78.1	0.26	0.22	0.66	379	0.57	1.44
	Inferred	16.0	0.18	0.19	0.09	66	0.44	0.23
Total	Measured	62.2	0.49	0.21	0.99	287	0.79	1.58
	Indicated	175.6	0.39	0.15	2.19	589	0.60	3.40
	Inferred	17.6	0.20	0.18	0.11	69	0.45	0.26
Grand Total	Measured and Indicated	237.7	0.42	0.17	3.18	875	0.65	4.97

NOTES:

1. Mineral Resources are reported inclusive of Mineral Reserves. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
2. Mineral Resources are contained within conceptual pit shells that are generated using the same economic and technical parameters used for Mineral Reserves but at a gold price of US$1,700/oz and a copper price of US$3.50/lb.
3. The Colnic and Rovina deposits are amenable to open-pit mining and Mineral Resources are pit constrained and tabulated at a base case cut-off grade of 0.35 g/t AuEq for Colnic and 0.25 % CuEq for Rovina.
4. Minor summation differences may occur as a result of rounding.

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,700/oz and a copper price of US$3.50/lb with metallurgical recoveries not taken into account.

The Ciresata underground mineral resource estimate remains unchanged from the 20 February 2019 estimate by AGP. Table 25.2 summarises the mineral resource estimate for Ciresata, stated at above a 0.65 g/t Au equivalent grade cut-off. The Measured mineral resources amount to 28.5 Mt grading at 0.88 g/t Au and 0.16 % Cu, containing 0.81 Moz Au and 102 Mlb Cu, with the Au equivalent grading of 1.13 g/t. The Indicated mineral resources amount to an additional 125.9 Mt grading at 0.74 g/t Au and 0.15 % Cu, containing 3.01 Moz Au and 413 Mlb Cu, with the Au equivalent grading of 0.97 g/t.

Table 25.2: 2019 Mineral Resource Estimate – Ciresata Deposit

Deposit	Resource Classification	Tonnage (Mt)	Au (g/t)	Cu (%)	Au (Moz)	Cu (Mlb)	AuEq* (g/t)	AuEq* (Moz)
Ciresata	Measured	28.5	0.88	0.16	0.81	102	1.13	1.03
	Indicated	125.9	0.74	0.15	3.01	413	0.97	3.92
	Inferred	8.6	0.70	0.14	0.19	26	0.94	0.25
Total	**Measured & Indicated**	**154.4**	**0.77**	**0.15**	**3.82**	**515**	**1.00**	**4.95**

NOTES:

1. The Ciresata deposit is amenable to bulk underground mining and resources are tabulated at a base case 0.65 g/t AuEq.
2. No Mineral Reserves have been defined at the Ciresata deposit. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
3. Minor summation differences may occur as a result of rounding.

* The Au and Cu equivalents were determined by using a long-term gold price of US$1,500/oz and a copper price of US$3.50/lb.

Source: From Table 14-20, AGP PEA NI 43-101 2019 Report (available on SEDAR)

It must be noted that the quantity and grade of Inferred resource reported above are conceptual in nature and are estimated based on limited geological evidence and sampling. Geological evidence is sufficient to imply, but not verify, geological and grade or quality continuity. For these reasons, an Inferred mineral resource has a lower level of confidence than an Indicated mineral resource, and it is reasonably expected that the majority of Inferred mineral resources could be upgraded to an Indicated mineral resource with continued exploration. Mineral resources that are not mineral reserves do not have demonstrated economic viability. Rounding of tonnes as required by reporting guidelines may result in apparent differences between tonnes, grade, and contained metal content.

Changes in the current metal prices and updated operating parameters from the 2012 mineral resource estimate resulted in a shrinkage of the Lerchs-Grossmann mineral resource constraining shell and, therefore, a reduction in the overall mineral resource estimates for the Rovina and Colnic deposits. The total Measured mineral resource tonnage increased by 1.4 %, with the Au and Cu grades remaining the same. The total Indicated mineral resource tonnage decreased by 2.8 %, from 180.7 Mt to 175.6 Mt, with the Au and Cu grades remaining the same. The total Inferred mineral resource tonnage decreased by 10.4 %, from 19.6 Mt to 17.6 Mt, with the Au and Cu grades remaining the same.

CCIC MinRes's conclusions are given below.

25.1.1 Database

While other companies conducted exploration drilling on the property, only the ESM sampling data, collected since 2006, was used in the mineral resource estimate. This ensured that modern assaying techniques and proper QA/QC protocols were in place for the entire drill programme and eliminated any need to rely on historical data. CCIC MinRes has reviewed the methods and procedures used to collect and compile the geological and assaying information for the RVP and found that they met accepted industry standards for an advanced-stage project and are sufficient for the porphyry style of mineralisation.

25.1.2 QAQC

After a review of the QA/QC results for all drilling since 2006, CCIC MinRes concluded that, despite minor insufficiencies of the QA/QC programme, the Au and Cu assays from drillhole sampling are sufficiently precise and accurate for mineral resource estimation purposes.

25.1.3 Geological Models

CCIC MinRes assessed the possible impact of the six additional drillholes (from the ESM-Barrick exploration collaboration) on the mineral resource estimates for the Rovina and Colnic deposits and concluded that there is no risk of overestimation and, therefore, recommended that ESM not update the 2012 geological and mineral resource block models until more holes are added to the resource database.

CCIC MinRes completed a detailed technical audit of the geological and mineral resource block models for Rovina and Colnic and confirmed the robustness of the AGP mineral resource models for these deposits.

25.2 GENERAL PROJECT CONCLUSION

The RVP has undergone a significant amount of exploration, testing and study work to demonstrate what is believed to be a significant resource and reserve. This can be seen with the amount of detail expressed in this report and within the referenced documentation supporting this report. The level of accuracy used herein is sufficient to consider this report to be compliant with the NI 43-101 requirements with its demonstration of the technical feasibility to develop a concentrator at the RVP site that will produce the current reserve of 1,826 koz of gold and 208,231 t of copper in saleable concentrate form over a 16-year LOM.

This report has demonstrated that the RVP ore deposits can be economically mined using the open-pit mining method with waste crushing and conveying and processed through a flotation circuit at a planned annual throughput of 7.2 Mt/a.

There is a risk of a high bench turnover rate in the final four years of the Colnic pit production schedule. This risk will be mitigated in the next optimisation phase based on the new information detailed below:

- The recently completed geotechnical and hydrogeological studies have shown that there is some potential to steepen the pit bench slope angles in the Colnic pit by 5°, which will improve the project economics by reducing the strip ratio and lowering the total mining cost in the early RVP LOM.

- The completed Colnic waste storage facility design has shown that there is sufficient storage capacity for waste rock and processed tailings so that the mining rate at Colnic can be slowed. This will still allow for the waste crushing and conveying infrastructure to be moved from the Colnic pit during the Rovina pit predevelopment phase. This extra Colnic waste storage facility capacity will mitigate operational risk in operational life of both the Colnic and Rovina pits.

26 RECOMMENDATIONS

The following recommendations have been proposed for the further development and execution of the RVP.

26.1 GEOLOGY AND MINE DEVELOPMENT

In preparation for mine development, ESM should consider the following recommendations for operational readiness:

- Conduct pre-production drilling within the mine planning footprint for the first 24 to 36 months of ore mining. This will ensure that all mineral resources mined during this period are converted to Proven mineral reserves, thereby minimising geological risks during this crucial payback period.
- Implement database management software to manage and monitor sampling QA/QC programmes. This system will flag batches that are beyond the threshold limits and allow for immediate remedial action. Also, where there is evidence of positive or negative drifts, the laboratory can be notified to calibrate their equipment more regularly.
- Research the implementation of Leapfrog Implicit modelling of geological and geostatistical domains during grade-control modelling. This will allow for quick and efficient updating of the geological models when turnaround times are crucial.
- Undertake a geostatistical study to determine the optimum drill spacing for grade-control modelling. Optimum drill spacing will assist with time and costs during mine production.
- Research the correlation between the results from a mobile X-ray fluorescence (XRF) scanner and those from a laboratory. This will be useful for grade-control sampling and modelling as follows:

    - If there is a reliable proxy between the Cu and Au grades in the deposit, then the XRF readings for Cu can be used as a proxy for anticipating the Au grades.
    - If a reliable proxy can be established, this will facilitate quick and cost-efficient turnaround times for grade-control assays.

26.2 METALLURGY

The following recommendations are proposed for future metallurgical work.

26.2.1 Flotation Configuration

Dilution cleaner test work has been carried out to determine the performance of the scalper rougher Jameson flotation on the Rovina Valley ores. When available, the results will confirm and validate the rougher circuit configuration and equipment sizing while minimising the capital and operational costs.

An investigation of the mineralogical characteristics of the high zinc content and corresponding flotation reagents schemes is required during operations to effectively depress zinc without decreasing gold recovery.

26.2.2 IsaMill Signature Plots

IsaMill signature plots tested on primary milled composites from the Colnic and Rovina orebodies showed significantly higher specific energy requirements compared to initial estimates used in the DFS. Since the mainstream composite samples are not necessarily representative of the concentrate from ore, for a better understanding of what the true specific energy would be, it is recommended that a further signature plot test on a rougher-scavenger concentrate be investigated.

26.3 MINING

The recently completed geotechnical and hydrogeological studies have shown that there is some potential to steepen the pit bench slope angles in the Colnic pit by 5°, which will improve the project economics by reducing the strip ratio and lowering the total mining cost in the early RVP LOM. This is to be incorporated in design optimisations.

26.4 HAZARD AND OPERABILITY

It is recommended that the further stages of the HAZOP studies be followed as the project progresses through design and development. This would start with the HAZOP 3 and electrical area classification studies.

26.5 ENGINEERING STUDIES

A more detailed field based seismic hazard assessment will need to be carried out in the early stages of detailed design. This is necessary to identify and assess any potentially active faults in the vicinity of the project site. Additional climate studies that include precipitation, snowpack, evaporation monitoring, wind studies and collection of continuous flow and precipitation monitoring for the catchments will refine the design assumptions regarding precipitation, base flows, time of concentration and losses in the local catchments.

26.6 SITE INVESTIGATIONS

Additional geotechnical and hydrogeological research and testing in the following areas will allow final design documents to be submitted for the on-going permitting as well as the construction tender process.

- Foundations at the processing plant site.
- Subsoil conditions along the proposed road alignments.
- The proposed CDF and WMFs, including the location of each proposed dam.
- The proposed tunnel alignments and their portals.

Furthermore, an additional geotechnical investigation at the Rovina Pit and additional geotechnical characterization of the Flysch unit located in the upper NE parts of the Colnic Pit will bolster the designs presented in this report.

26.7 GENERAL

Given the promising results from this March 2021 DFS, SENET would expect ESM to continue to advance the RVP. The next steps are expected to include ongoing permitting activities and culminating in obtaining an operating licence. During this phase it would be prudent to commence with detailed engineering and SENET recommends that ESM start the Front-End Engineering Phase of the project to be ready for project construction once all licences are obtained.

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