RungePincockMinarco

NI 43-101 Technical Report

Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant

District of Panfilo Natera,
Zacatecas State, Mexico

Prepared for:

Xtierra Inc.

Prepared by:

Kevin Tanas, P.Eng.
Esteban Acuña
Richard Parker, BSc, MIMMM, C.Eng., FGS
Malcolm Buck, P.Eng.
Lyn Jones, P.Eng.
Clinton Swemmer, PMP PrEng MSAICE
Jane Spooner, MSc, P.Geo.
Dana Strength, R.G.

Report No: ADV-TO-00011

Date: April 28, 2014

Date and Signature Page

The effective date of this Technical Report, entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" is April 28, 2014. The undersigned have prepared the Technical Report in accordance with National Instrument 43-101 guidelines for Technical Reports.

Kevin Tanas {Signed and Sealed} April 28, 2014

Kevin Tanas, P.Eng. Dated at Toronto, Ontario
Principal Mining Consultant
RungePincockMinarco (Canada) Limited

Esteban Acuña {Signed and Sealed} April 28, 2014

Esteban Acuña Dated at Toronto, Ontario
Senior Geologist
RungePincockMinarco

Richard Parker {Signed and Sealed} April 28, 2014

Richard Parker, BSc, MIMMM, C.Eng., FGS Dated at Edinburgh, Scotland
Consulting Geologist

Malcolm Buck {Signed and Sealed} April 28, 2014

Malcolm Buck, P.Eng. Dated at Toronto, Ontario
Mining Consultant
AMBUCK Investment Corporation

Lyn Jones {Signed and Sealed} April 28, 2014

Lyn Jones, P.Eng. Dated at Toronto, Ontario
Principal Metallurgist
ConsuMet

Clinton Swemmer {Signed and Sealed} April 28, 2014

Clinton Swemmer PMP PrEng (South Africa) MSAICE Dated at Toronto, Ontario
Vice President: Projects
DRA Americas Inc.

Jane Spooner {Signed and Sealed} April 28, 2014

Jane Spooner, MSc, P.Geo. Dated at Toronto, Ontario
Vice President
Micon International Limited

Dana Strength {Signed and Sealed} April 28, 2014

Dana Strength, R.G. Dated at Toronto, Ontario
President
Strength GEC

Certificate of Qualifications

RungePincockMinarco (Canada) Limited

Suite #1007
141 Adelaide Street West
Toronto Ontario M5H 3L5
Canada

April 28, 2014

Certificate of Qualified Person
Kevin Tanas, P.Eng.

I, Kevin Tanas, Principal Mining Consultant, as an author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am employed by, and carried out this assignment for:

   RungePincockMinarco (Canada) Limited
   Suite 1007, 141 Adelaide Street West
   Toronto, Ontario
   M5H 3L5

   tel. (416) 306-3096
   fax (403) 217 4389
   e-mail: ktanas@rpmglobal.com

2. I hold the following academic qualifications:

   B.Sc. (Mining Engineering) Queens University 1999

3. I am a registered Professional Engineer with Professional Engineers Ontario (Licence #100182770).

4. I have practiced as a mining engineer for 15 years since my graduation from university.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. I have been involved in the planning, evaluation and operation of mineral properties for gold, copper, lead, zinc, vanadium, iron, coal and oil sands in Canada, United States, Chile, Brazil and Australia.

6. I visited Xtierra's Bilbao project during the period February 25th to March 1st, 2013 as part of this project review. The property had not previously been visited by me.

7. I am responsible for Sections 1.0 (Summary), 2.0 (Introduction), 3.0 (Reliance on Other Experts), 4.0 (Property Description and Location), 5.0 (Accessibility, Climate, Local Resources, Infrastructure and Physiography), 6.0 (History), 15.0 (Mineral Reserve Estimates), 18.0 (Infrastructure), 21 (Capital and Operating Costs), 22 (Economic Analysis), 23.0 (Adjacent Property), 24.0 (Other Relevant Data and Information), 25.0 (Interpretation and Conclusions), 26.0 (Recommendations) and 27.0 (References) of the Technical Report.

8. I am independent of the parties involved in the transaction for which this Technical Report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated in Toronto, Ontario this 28th day of April, 2014.

"Kevin Tanas" {Signed and Sealed}

____

Kevin Tanas, P.Eng.
Principal Mining Consultant

RungePincockMinarco (U.S.) Limited

165 South Union Blvd
Suite #950
Lakewood, Colorado 80228
United States

April 28, 2014

Certificate of Qualified Person
Esteban Acuña, Geologist.

I, Esteban Acuña, Senior Geology Consultant, as an author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am employed by, and carried out this assignment for:

   RungePincockMinarco (U.S.A) Limited
   165 S Union Blvd, Suite 950,
   Lakewood, Colorado
   80228

   tel. (303) 914-4473
   e-mail: eacuna@rpmglobal.com

2. I hold the following academic qualifications:

   Geologist, University of Concepcion, Chile 1995

3. I am a registered of Comision Calificadora de Competencias en Recursos y Reservas Mineras (member of CRIRSCO) (Register #0161).

4. I have practiced as a resource geologist for 18 years since my graduation from university.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. I have been involved in the geologic modelling, resource estimation and operation of mineral properties for copper, gold, molybdenum, lead, zinc, and iron, in Chile, United States, Canada, Brazil, Peru, Mexico, Pakistan, Zambia, and Botswana.

6. I visited Xtierra's Bilbao project during the period February 25th to March 1st, 2013 as part of this project review. The property had not previously been visited by me.

7. I am responsible for Sections 10.0 (Drilling), 11.0 (Sampling Preparation, Analyses, and Security), 12.0 (Data Verification), and 14.0 (Mineral Resources) of the Technical Report.

8. I am independent of the parties involved in the transaction for which this Technical Report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated in Lakewood, Colorado this 28th day of April, 2014.

Esteban Acuña, Geologist.
Senior Geology Consultant

Comisión Calificadora de
Competencias en Recursos
y Reservas Mineras

CERTIFICATE OF QUALIFIED COMPETENCY

The Chilean **Comisión Calificadora de Competencias en Recursos y Reservas Mineras**[1], certifies that Mr. **Esteban Acuña,** National Id. Nr 10.900.412-K, Geologist, is registered in the Public Registry of Competent Persons in Mining Resources and Reserves, under Nr. 0161, with specialization in **Geology,** and that his competencies and experience as a Competent Person allow to inform and report on mineral deposits up to date.

The Chilean Mining Commission issued this certificate at the request of Mr. Acuña to present:
Preliminary Economic Assessment of the Bilbao Silver-Zinc-Lead Project, 720,000 Tonnes per Year Processing Plant.
Sections 10,11,12, and 14.

Gladys Hernández
Executive Secretary

Santiago, march 4th, 2014
CM – 119 – 032014

Information:

a. **The Certificate of Qualified Competency** proves the validity of the party's competencies to inform or report about a specific matter or subject in the context of mining resources and reserves in accordance with the competencies and experience of a Competent Person.

b. **Law No. 20.235 , Article 18°** : For the preparation of the technical and public reports, the Competent Persons must adhere strictly to the rules, regulations, criteria and procedures established in the Code, and likewise to all other rules of technical character that the Mining Commission enacts using their legal faculties."

c. **Application of CH 20.235 code** and use of this certificate is the sole responsibility of the person concerned , according to the technical criteria and ethical standards set forth in Law No. 20,235.

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[1] The **Comisión Calificadora de Competencias en Recursos y Reservas Mineras** is a member of the Committee for Mineral Reserves International Reporting Standards (CRIRSCO) that groups the organizations of Australia (JORC), Canada (Instrument 43-101), South Africa (SAMREC), U.S.A. (Society of Mining Engineers), Europe (PanEuropean Code), Russia (NAEN), which respond to a common international ruling to inform and report exploration prospects, mining resources and reserves.

SONAMI CONSEJO MINERO COLEGIO DE INGENIEROS DE CHILE A.G.

Luis Thayer Ojeda 166, oficina 706, Providencia Santiago de Chile Teléfonos (562) 22345134 – 22343016

RICHARD PARKER
CONSULTING GEOLOGIST

==

86-8 Trinity Road, Edinburgh EH5 3JU, Scotland
phone 0131 5527142, mobile 07941847576
rickparker@blueyonder.co.uk,

April 28, 2014

Certificate of Qualified Person
Richard Parker, C.Eng.

I, Richard Parker, Consulting Geologist, a co-author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am employed by, and carried out this assignment for:

   Richard Parker, Consulting Geologist
   86-8 Trinity Road,
   Edinburgh EH5 3JU
   Scotland
   UNITED KINGDOM
   tel. +44 131 5527142
   e-mail: rickparker@blueyonder.co.uk

2. I hold the following academic qualifications:

   B.Sc. (Geology), University of Newcastle on Tyne, United Kingdom, 1968

3. I am a Chartered Engineer (registration number 323907) registered with the Engineering Council (UK) and a Professional Member of the Institute of Materials, Minerals and Mining (Member Number 465460.

4. I have practiced as a mineral exploration and mining geologist for 44 years since my graduation from university.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. I have been involved in mineral exploration and mine development throughout my career and have practiced in over 30 countries.

6. I visited Xtierra's Bilbao project during the periods 23rd January to 7th February 2007, 6th January to 16th January 2010 and September 18th to 23rd, 2012.

7. I am responsible for Sections 7.0 (Geological Setting and Mineralisation), 8.0 (Deposit Types), and 9.0 (Exploration) of the Technical Report.

8. I am independent of the parties involved in the transaction for which this Technical Report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated in Edinburgh, UK this 28th day of April, 2014.

R T G Parker

____
Richard Parker, C.Eng.
Consulting Geologist

AMBUCK Associates

CERTIFICATE of QUALIFIED PERSON
MALCOLM BUCK, P.ENG.

I, Malcolm Buck, M.Eng., P. Eng., President, of AMBUCK Associates a subsidiary of AMBUCK Investment Corporation, residing at 164 Castle Crescent, Oakville, Ontario, Canada as an author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am an Associate of RungePincockMinarco (Canada) Limited.

2. I am a graduate of The Technical University of Nova Scotia, with a Bachelor of Engineering in Mining Engineering (1983). I have also obtained a Masters of Engineering, in Mining Engineering (Mineral Economics) from McGill University (1986).

3. I am licensed by the Professional Engineers Ontario (License No. 5881503). In addition, I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

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. My relevant experience is:

    - Practiced my profession continuously since 1983.
    - Extensive and progressively more senior engineering and operational duties at base metals, gold and uranium mining operations and development projects.
    - 20 years' experience performing all types of feasibility studies and due diligence and strategic planning studies for mines and mining companies.

6. I am responsible for Section 16.0 (Mining Methods) of the Technical Report.

7. I visited Xtierra's Bilbao project during the period February 25th to March 1st, 2013 as part of this project review. The property had not previously been visited by me.

8. As of the date of this certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9. I am independent of the issuer applying all of the tests in sect 1.5 of NI 43-101.

10. I have not had prior involvement with the Property that is the subject of this Technical Report.

11. I have read NI 43-101 and Form 43-101F1 and this Technical Report has been prepared in compliance therewith.

Dated in Toronto, Ontario this 28th day of April, 2014.

LICENSED PROFESSIONAL ENGINEER
M. K. BUCK
PROVINCE OF ONTARIO

Malcolm Buck, P.Eng.

AMBUCK Associates, 164 Castle Crescent, Oakville, Ontario, L6J 5H3, CANADA
Telephone: (416) 573-2031

ConsuMet
Mineral Process Consulting

651 Walkerfield Avenue
Peterborough, ON
K9J 4W1
705-768-0461
lyn.jones@consumet.ca

April 28, 2014

I, Lyn Jones of Peterborough, Ontario, as an author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and with an effective date of 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am Principal Metallurgist with ConsuMet Ltd, with a business address at 651 Walkerfield Avenue, Peterborough, ON K9J 4W1.
2. I am a graduate of the University of British Columbia, Metals and Materials Engineering, 1998.
3. I am a member in good standing of the Association of Professional Engineers of Ontario, Registration No. 100067095.
4. I have practiced my profession continuously since graduation.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101 or the Instrument) 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 purpose of NI 43-101.
6. My relevant experience with respect to metallurgy and process design includes 15 years of work experience in flowsheet development, process engineering, and plant operations.
7. I have visited the property from September 28th-30th, 2010.
8. I am responsible for Section 13.0 (Mineral Processing and Metallurgical Testing) and those portions of the Summary, Interpretations and Conclusions, and Recommendations that pertain to that section of the Technical Report.
9. I have had no prior involvement with the mineral properties in question.
10. I have read NI 43-101 and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
11. As of the effective date of the Technical Report, to my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed and dated this 28th day of April 2014, at Peterborough, Ontario.

LICENSED PROFESSIONAL ENGINEER
L. JONES
100067095
28/04/14
PROVINCE OF ONTARIO

Lyn Jones, P.Eng.

April 28, 2014

Certificate of Qualified Person
Clinton Swemmer, PrEng (South Africa)

I, Clinton Swemmer, Vice President: Projects, as an author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am employed by, and carried out this assignment for:

DRA Americas
Suite 300, 44 Victoria Street
Toronto, Ontario
M5C 1Y2

tel. (416) 800-8797 ext. 105
e-mail: clinton.swemmer@DRAglobal.com

2. I hold the following academic qualifications:

B.Eng (Hons) Civil Engineering University of Hertfordshire 1998

3. I am a registered Professional Engineer with Engineering Council South Africa (Registration #20090418) and a member of the Project Management Institute.

4. I have practiced as an engineer in minerals processing industry for over 10 years.

5. I do, by reason of education, experience and professional registration, fulfil the requirements of a Qualified Person as defined in NI 43-101. I have been involved in the planning, evaluation, engineering, procurement, construction, commissioning and / or operation of mineral properties including for gold, platinum, copper, lead, zinc, nickel, graphite, vanadium, iron, coal and uranium in Canada, Tanzania, Guyana, South Africa, Brazil, Mozambique, Zimbabwe and Botswana.

6. I have not visited Xtierra's Bilbao property.

7. I am responsible for Section 17.0 (Recovery Methods) of the Technical Report.

8. I am independent of the parties involved in the transaction for which this Technical Report is required, as defined in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the part of this Technical Report for which I am responsible has been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated in Toronto, Ontario this 28th day of April, 2014.

"Clinton Swemmer, Pr.Eng (South Africa), PMP, MSAICE" - Signed

____

Clinton Swemmer, Pr.Eng (South Africa), PMP, MSAICE.
Vice President: Projects

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ENGINEERING 30 EXCELLENCE

micon INTERNATIONAL LIMITED | mineral industry consultants

CERTIFICATE OF QUALIFIED PERSON
JANE SPOONER, M.Sc., P.Geo.

As a co-author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), I, Jane Spooner, P.Geo., do hereby certify that:

1. I am employed by, and carried out this assignment for
   Micon International Limited
   Suite 900, 390 Bay Street
   Toronto, Ontario
   M5H 2Y2
   tel. (416) 362-5135 fax (416) 362-5763
   e-mail: jspooner@micon-international.com

2. I hold the following academic qualifications:

   B.Sc. (Hons) Geology, University of Manchester, U.K. 1972
   M.Sc. Environmental Resources, University of Salford, U.K. 1973

3. I am a member of the Association of Professional Geoscientists of Ontario (membership number 0990); as well, I am a member in good standing of the Canadian Institute of Mining, Metallurgy and Petroleum.

4. I have worked as a specialist in mineral market analysis for over 35 years.

5. I do, by reason of education, experience and professional registration, fulfill the requirements of a Qualified Person as defined in NI 43-101. My work experience includes the analysis of markets for base and precious metals, industrial and specialty minerals, coal and uranium.

6. I have not visited the project site.

7. I am responsible for the preparation of Section 19 (Market Studies and Contracts) of the Technical Report.

8. I am independent of the parties involved in the transaction for which this Technical Report is required, as described in Section 1.5 of NI 43-101.

9. I have had no prior involvement with the mineral property in question.

10. I have read NI 43-101 and the portions of this report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the section of this Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this report not misleading.

Signing Date: 28 April, 2014

PROFESSIONAL GEOSCIENTIST
JANE SPOONER
PRACTISING MEMBER
0990
ONTARIO

Jane Spooner, M.Sc., P.Geo.
Vice President

SUITE 900 - 390 BAY STREET, TORONTO ONTARIO, CANADA M5H 2Y2
Telephone (1) (416) 362-5135 Fax (1) (416) 362 5763

STRENGTH
GEC

April 28, 2014

Certificate of Qualified Person
Dana A. Strength, R.G.

I, Dana Strength, Environmental and Social Specialist, as an author of this report entitled "Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant" prepared for Xtierra Inc. ("Xtierra"), and effective date 28th April 2014 (the "Technical Report"), do hereby certify that:

1. I am self-employed, and have carried out this assignment on a subcontract basis for RungePincockMinarco (Canada) Limited. My contact information is:

Mr. Dana A. Strength
Principal, Strength GEC, LLC
27 Songbird Lane
Durango, CO 81301
USA

tel. (571) 216-8087
e-mail: dana@strengthgec.com

2. I hold the following academic qualifications:

M.S. (Geology)	Northern Arizona University	1997
B.S. with Honours (Geology)	Indiana University	1994

3. I am a registered Professional Geologist with the State of Arizona (Licence #40819).

4. I have worked as an environmental and social professional for 17 years and have been involved in the evaluation of environmental and social impacts for hundreds of small to large-scale projects, with a particular focus in the mining sector

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 past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.

6. I visited Xtierra's Bilbao project during the period February 25th to March 1st, 2013 as part of this project review. The property had not previously been visited by me.

7. I am responsible for the overall preparation of Section 20 of the Technical Report (Environmental Studies, Permitting and Social or Community Impact)

8. I am independent of the parties involved in the transaction for which this Technical Report is required, as defined in Section 1.5 of NI 43-101.

STRENGTH GEC

9. I have had no prior involvement with the mineral properties in question.

10. I have read NI 43-101 and the portions of this Technical Report for which I am responsible have been prepared in compliance with the instrument.

11. As of the date of this certificate, to the best of my knowledge, information and belief, the portion of the Technical Report for which I am responsible contains all scientific and technical information that is required to be disclosed to make this Technical Report not misleading.

Dated in Durango, Colorado, USA this April 28, 2014

Dana A. Strength

Dana A. Strength
Environmental and Social Specialist

Document Control Sheet

Client	
Xtierra Inc.	
Report Name	**Date**
Preliminary Economic Assessment of the Bilbao Silver-Lead-Zinc Project, 720,000 Tonnes per Year Processing Plant	April 28, 2014
Report No.	**Revision No.**
Report No: ADV-TO-00011	Rev. 0

Authorisations				
Name		**Position**	**Signature**	**Date**
Prepared By:	Kevin Tanas, P.Eng.	Principal Mining Consultant	{Signed}	April 28, 2014
Reviewed By:	Richard Kehmeier	Chief Geologist	{Signed}	April 28, 2014
Approved By				

Distribution				
Organisation	**Recipient**	**No. Of Hard Copies**	**No. Of Electronic Copies**	**Comment**
RungePincockMinarco	Xtierra Inc.		1	

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1. Summary

1.1 Introduction

Xtierra Inc. (Xtierra) retained RungePincockMinarco (Canada) Ltd. (RPM) to prepare a Preliminary Economic Assessment of the Bilbao silver-lead-zinc project located in the State of Zacatecas, Mexico and to act as Qualified Person in accordance with National Instrument 43-101. RPM updated a previous resource model taking into account additional drilling completed in both 2011 and 2013 and coordinated and supervised various third party independent consultants to carry out various studies: Nordmin Engineering Ltd. (Nordmin) developed the mine design and production schedule; DRA Americas Inc. (DRA) analyzed metallurgical testing and recovery methods and designed the process plant; Golder Associates (Golder) carried out various environmental studies including the design of the Tailings Disposal Facility; Micon International Limited (Micon) carried out a market study review.

1.2 Property Description and Location

The Bilbao project (the “Project”) is located in the Municipality of General Panfilo Natera, in the State of Zacatecas, at an elevation of between 2,145 and 2,160 meters above sea level. The small community of Panfilo Natera is located approximately 3.9 km from the site. The Bilbao mineral deposit is covered by several claims comprising a total of 1,407 hectares. This area is surrounded by flat active and fallow farmlands, with a relatively flat morphology and overall arid conditions. The majority of irrigated farmland in the region is owned by the ejidos of Panfilo Natera and Ojo Caliente, with the balance privately owned.

The State of Zacatecas has experienced centuries of mining development, and the overall region has a high density of active and inactive mining works. Access to the Project is straightforward, with immediate connection from a paved highway. The Bilbao site has undergone historic development, which is evidenced by an abandoned shaft present on site and numerous existing open pits. Oxide “glory hole” materials were historically accessed via crude excavation at these open pits, and from underground workings on two levels accessed via the abandoned shaft.

1.3 Mineralization

The Bilbao Deposit is a contact metamorphic deposit, classified as skarn type. It is developed in the marbleized limestone at the contact with the La Blanca granodiorite (granite). The mineralization occurs as sulphide replacement bodies formed along the bedding in the limestone (mantos) and as minor replacement bodies in the intrusive (endoskarn). The highest grades are found in contact with the main intrusive body. Grades sharply decrease from the main intrusive body toward the west. The principal contained economic metals are silver, zinc, copper and lead together with lesser amounts of gold, cadmium and tin. The deposit is weathered to an average depth of 120 metres so that the upper part of the mineralized body consists of iron oxides containing Ag-Zn-Pb-Cu. Below the oxide capping, the metals occur as sulfides.

1.4 Exploration and Drilling

Since 2006, Xtierra has drilled 113 diamond drill-holes in the Bilbao deposit, with geological logs provided to RPM. The ASCII database provided to RPM contained assay data for 108 holes. The geological model was generated using 113 holes (all the logged drill holes). Of the 113 holes with geological logs, 6 were not included in the database (X47, X47A, X57, X87, X90, and X91) and 2 did not have assays (G3 and Z11). In the database one hole had assays but no geological log and one hole (X100) was duplicated. The block resource model was estimated using 105 holes which had assays. Table 1-1 lists the 105 holes by drill campaign.

Table 1-1 Drill Hole Campaigns

Campaign	Year	N	(m)
Phase I	2006	28	7222
Phase II	2008	15	4138
Phase III	2008 - 2009	7	1900
Phase IV	2010 - 2011	31	7688
Phase V	2011 - 2012	18	4875
Phase VI	2013	6	1785
Total		**105**	**27609**

All of the drill-holes are diamond NQ-HQ core holes with most (104) being vertical. They have been drilled by the company through six campaigns since 2006 completing a general grid of 50 m by 50 m and a tighter drilling grid of 35 m by 35 m in the high grade core. After the last 2010 resource estimation, 26 infill holes were drilled to complete the tight grid in the central high-grade zone. The drilled zone extends over an area of 530m along north-south axis and 580m along east-west axis.

Xtierra collected the geological and assay data from the drill program and compiled it in an MS Access database and plotted it on a series of N-S sections. The lithology table contains the codes for six sedimentary or volcanic rock units plus granite, fault, vein and non-recovery (Table 1-2).

Table 1-2 Lithology Codes and Geological Units

Geological Model	Codes in 2013 DB
Upper Sedimentary Unit	Alluvium_Soil
	Basalt
	Lithic_Arenite
	Piedmont_Breccia
Lower Sedimentary Unit	Limestone
	Exoskarn
Intrusives	Granite
	Endoskarn
Others	Fault
	Vein
	No_Recovery

1.5 Mineral Resource Estimates

This resource model is an update of previous models incorporating twenty holes drilled during 2011 and 2013, which completed a total of 105 drill holes in the deposit. A lithology model was built and Indicator and Ordinary Kriging (OK) were used to estimate Zn, Pb, Ag and Cu resources. Density was updated using 224 new density determinations completed since the last 2010 model was constructed. The previous 2010 model (revised in 2011) had assigned a density of 3.6 g/cc to sulphide blocks based on the average of 14 measurements.

Previous resource models have been completed for Xtierra at Bilbao by Parker beginning in 2007. The last resources model completed was in 2011 and included 84 drill holes. The resources (including both oxide and sulphide) reported in 2011 are 10,617,891 tonnes @ 6.48% Zneq in the indicated category and 430,000 tonnes @ 5.19% Zneq in the inferred category, based on an estimation distance of 40 m.

The previous resource estimation was originally carried out by Bilbao geologists along with a modeling consultant and QP, Richard Parker Consulting Geologist. Lithology and a 3% equivalent zinc (Zneq) grade shell were the geological constraints used to complete an inverse distance estimation of Ag, Pb, Zn and Cu resources.

The Zn/Pb/Ag/Cu estimation of the Bilbao Deposit for Xtierra in Zacatecas, Mexico was executed by RPM to incorporate new drilling information acquired during 2011-2013.

The scope of this estimation started with compositing and ended with resources classification. Database and QAQC of 2011-2013 campaigns were checked by RPM. The historical database was assumed accurate.

For the purpose of determining resources at various cutoff grades, Zn equivalent values were defined, based on the average price of the last 3 years. The utilized prices were US$0.935 lb/Zn, US$1.008 lb/Pb, and US$30.235 oz/Ag. Metallurgical recoveries were applied in the equivalent equation as 76.7%, 90.6% and 73.4% for Zn, Pb, and Ag, respectively. The Zn equivalent equation is as follows:

$$Zneq = Zn + 0.969*Pb + 0.09947*Ag$$

The total sulphide resources are listed by Zneq cutoff in Table 1-3. The total indicated sulphide resources are listed by Zneq cutoff in Table 1-4.

Table 1-3 Total Sulphide Resources

Cutoff	Zn equiv. (%)	Indicated Tonnes	Inferred Tonnes	Total Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
0.0	0.458	17,831,825	209,299,540	227,131,365	0.105	0.078	6	0.015
0.5	1.994	12,978,084	28,432,051	41,410,135	0.449	0.378	24	0.06
1.0	2.906	9,730,543	14,496,555	24,227,098	0.656	0.566	35	0.083
1.5	3.781	7,491,992	8,413,865	15,905,857	0.893	0.742	44	0.101
2.0	4.934	6,014,809	4,177,905	10,192,714	1.273	0.993	54	0.128
2.5	5.979	5,124,220	2,234,647	7,358,867	1.658	1.214	63	0.15
3.0	6.883	4,555,809	1,201,032	5,756,841	2.025	1.403	69	0.167
3.5	7.569	4,138,652	708,864	4,847,516	2.319	1.545	74	0.177
4.0	8.081	3,801,363	474,136	4,275,499	2.546	1.651	77	0.184
4.5	8.551	3,481,995	328,528	3,810,523	2.757	1.751	80	0.189
5.0	8.966	3,183,043	252,741	3,435,784	2.945	1.83	83	0.194
5.5	9.412	2,878,188	189,440	3,067,628	3.13	1.925	86	0.2
6.0	9.844	2,601,525	142,867	2,744,392	3.303	2.019	89	0.206

Table 1-4 Total Indicated Sulphide Resources

Cutoff	Zn equiv. (%)	Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
0.0	2.633	17,831,825	0.756	0.554	27	0.077
0.5	3.521	12,978,084	1.015	0.748	35	0.102
1.0	4.451	9,730,543	1.294	0.952	44	0.126
1.5	5.412	7,491,992	1.599	1.154	53	0.146
2.0	6.316	6,014,809	1.909	1.332	61	0.163
2.5	7.026	5,124,220	2.171	1.467	68	0.175
3.0	7.562	4,555,809	2.382	1.571	72	0.183
3.5	7.997	4,138,652	2.561	1.657	75	0.188
4.0	8.374	3,801,363	2.72	1.733	77	0.192
4.5	8.753	3,481,995	2.88	1.814	80	0.196
5.0	9.129	3,183,043	3.041	1.886	83	0.201
5.5	9.54	2,878,188	3.203	1.969	86	0.206
6.0	9.943	2,601,525	3.361	2.055	89	0.212

Total resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore can be seen in Table 1-5.

Table 1-5 Total Resources

Ore Type	Zn equiv. (%)	Indicated Tonnes	Inferred Tonnes	Total Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.50	791,082	3,069,582	3,860,664	1.70	2.33	42	0.17
Mixed	7.10	778,336	238,923	1,017,259	2.06	2.17	52	0.18
Sulphide	6.88	4,555,809	1,201,032	5,756,841	2.03	1.40	69	0.17
Total	6.76	6,125,227	4,509,537	10,634,764	1.91	1.81	58	0.17

Indicated resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore can be seen in Table 1-6.

Table 1-6 Indicated Resources

Ore Type	Zn equiv. (%)	Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.69	791,082	1.73	2.53	39	0.18
Mixed	7.93	778,336	2.52	2.48	51	0.21
Sulphide	7.56	4,555,809	2.38	1.57	72	0.18
Total	7.5	6,125,227	2.31	1.81	65	0.19

Inferred resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore can be seen in Table 1-7.

Table 1-7 Inferred Resources

Ore Type	Zn equiv. (%)	Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.38	3,069,582	1.69	2.23	42	0.16
Mixed	4.43	238,923	0.59	1.13	55	0.11
Sulphide	4.31	1,201,032	0.67	0.77	60	0.11
Total	5.73	4,509,537	1.36	1.78	47	0.15

1.6 Mineral Processing and Metallurgical Testing

Metallurgical testwork on samples from the sulfide zone of the Bilbao deposit has been carried out in three separate programs since 2007. A grindability study on composites from the sulfide and transition zones was used to generate data for grinding circuit simulation. Results indicated that the material is of average hardness with Bond Ball Mill Work Index (BBWI) values ranging from 11.8 kW/t for transition ore to 16.2 kWh/t for sulphide ore.

Flowsheet development testwork was conducted on the Sulfide Master Composite and indicated that with a moderate primary grind P80 of 100 µm, and a sequential float, high lead and zinc recoveries and good selectivity against pyrite could be achieved. Conventional lead-zinc techniques were used consisting of cyanide depression of the zinc, followed by lead flotation, copper sulfate conditioning, and zinc flotation.

Rougher lead and zinc concentrates were reground to target P80's of 45 µm and 40 µm, respectively, prior to three stages of cleaning at elevated pH. Final lead concentrate grades of 55-60% Pb were realized, while zinc concentrate grades ranged from 43-47% Zn with the grade limited as a result of the inherently low zinc content of the zinc mineral particles, primarily a consequence of the high iron content of the marmatite mineral. Locked cycle testing of a sulfide composite achieved good stability after six cycles of flotation and produced acceptable concentrate grades at lead and zinc recoveries of 90.6% and 76.7%, respectively. Silver recovery to the lead concentrate was 73.4%. Minor element analyses of the locked cycle test concentrates indicate the possibility of penalties for bismuth in the lead concentrate and for iron and cadmium in the zinc concentrate.

1.7 Mining

1.7.1 Geotechnical

Golder Associates Ltd. (Golder) performed the rock mechanics studies. Design parameters were provided and incorporated into the mine design and cost estimates. An update memo report entitled "Empirical and Numerical Analyses for Stope Sizing" was prepared in 2013 by Golder for the present proposed stoping configurations.

All permanent openings will have arched backs. Ground support will generally consist of pattern bolting using grouted rebar and welded wire mesh.

Stope dimensions are based on rock quality determinations and expected achievable open spans in stopes. The stope dimensions and extraction ratios anticipated will not be achieved without the use of a good quality cemented rock backfill.

1.7.2 Mine Design and Production Planning

Given the results of metallurgical testwork on the oxide and transition mineral zones, the mine plan incorporated in this study targets the extraction of the sulphide zone only. Underground mining methods will be used to access the sulphide zone located approximately 50 meters below surface, and accessed via a portal and ramp system.

The main proposed mining method is Longhole Open Stoping using downholes, while near the top of the deposit Longhole Open Stoping using upholes will be employed. Stopes will have maximum nominal dimensions of 24 metres wide by 12 metres long and a 24-metre vertical height. Longhole stopes will be backfilled with a cemented rock fill. Also, near the edges of the deposit, remnant Longhole Open Stoping with downholes will be used with no backfill placed in mined out stopes.

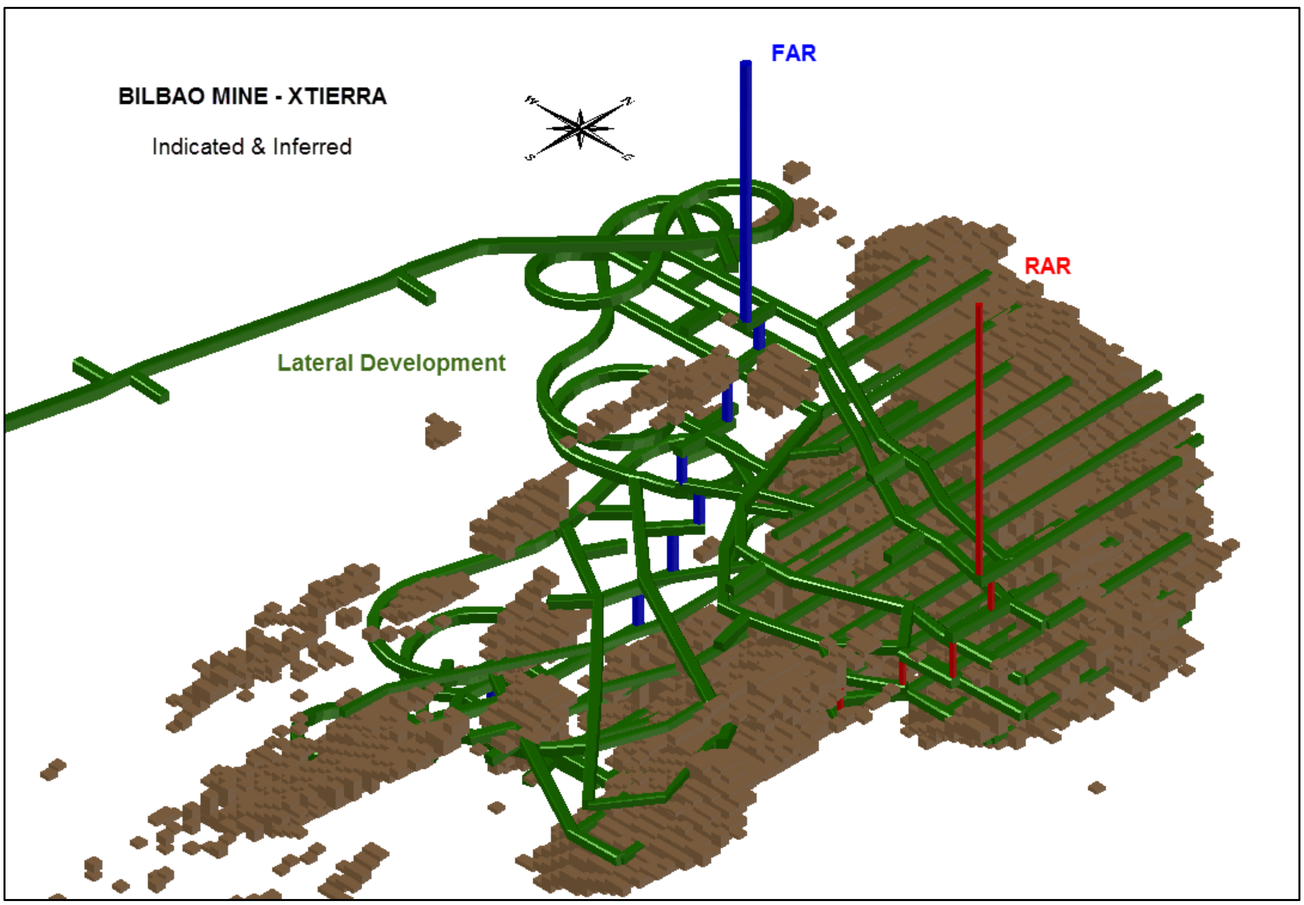
The main access to the underground mine will be via a main ramp from surface to the 1860 Level. The main ramp will connect from surface to all production levels in the mine, and provide a passage way for transportation of ore and waste material, personnel, materials and equipment. From the main ramp level accesses in waste at approximately 24 metre vertical intervals will be developed from the 2000 to 1860 Levels. Figure 1-1 shows a three dimensional view of the proposed underground mine design.

Levels accessing mining panels will be developed from the ramp as shown on a representative plan view of the 1940 Level development in Figure 1-2.

Stope development will consist of a panel (series of stopes across the width of deposit) access crosscut developed from the main level drift through the ore. This access crosscut will be used as a drawpoint for mucking each stope in a primary or secondary stoping panel.

The deposit geometry with a length of up to 150 to 200 metres and width in excess of 100 metres in areas requires that stopes be combined into 24 metre wide mining panels comprising a number of stopes accessed along a single panel access crosscut. Stope panels will be mined from the centre of the deposit to the outsides of the potentially economic mineralization, where most stopes will be backfilled with cemented rock fill to facilitate mining of the adjacent stope(s).

Figure 1-1 Mine Development – Looking North West



Backfill in stopes will consist of cemented rock fill in primary stopes and uncemented rock fill in secondary and isolated stopes. Remnant stopes will not be backfilled. Waste rock will be quarried approximately 2 kilometres from the mine, crushed to minus 0.3 metres and trucked to the mine.

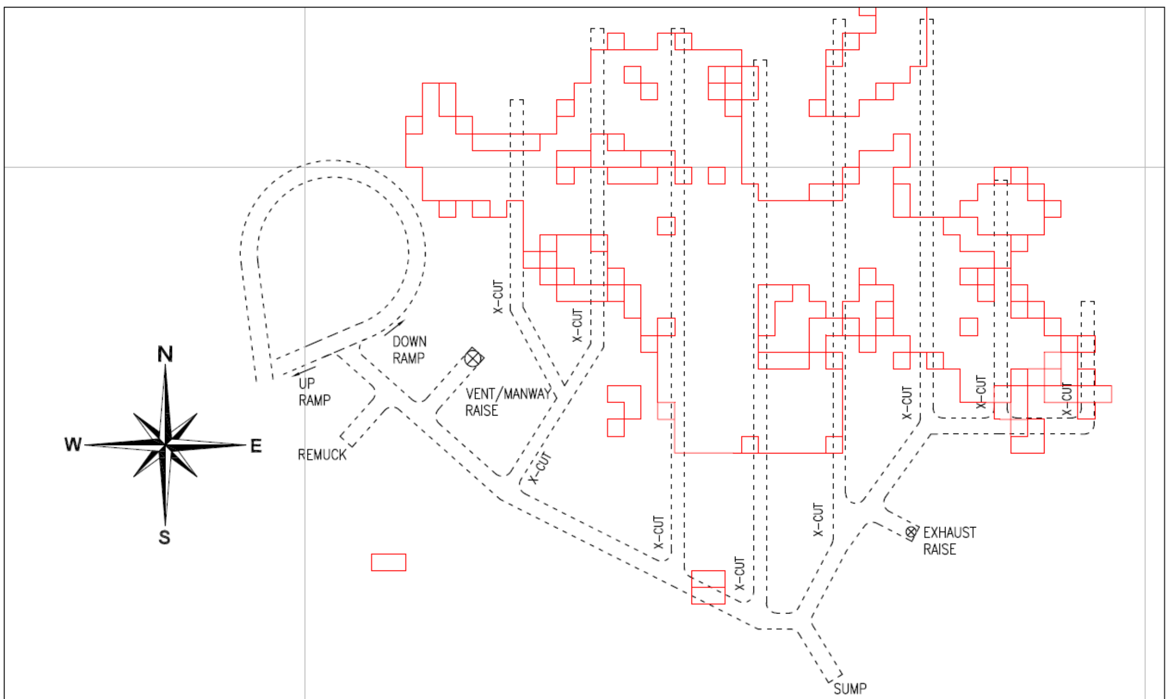
The cement slurry plant on surface will comprise of a cement silo equipped with a screw feeder measuring cement into a mixing tank where water is added to create cement slurry. The slurry will be pumped a short distance to the collar of the cement slurry delivery borehole from surface.

Secondary stopes will have uncemented rock backfill trucked to the stopes from the backfill raise and dumped directly into the mined out stopes.

Based on the selected mining method a dilution factor of 10% is applied which allows for dilution from hanging and footwall wall exposures and cemented backfill dilution which results from blasting against backfilled stopes. Mining recovery of 95% is assumed for this deposit.

Manpower and materials will enter and leave the mine via the main access ramp from surface. Personnel will travel in vehicles and/or personnel carriers to workplaces or equipment parking areas. Materials will be moved on a services truck, equipped with a boom crane, operating in the ramp. Materials will be transported to and placed in designated storage areas close to mining.

Figure 1-2 Typical Level Configuration



NORDPRO MINE & PROJECT MANAGEMENT SERVICES	Xtierra inc.	LOC.	BILBAO PRELIMINARY ECONOMIC ASSESSMENT		
		AREA	UNDERGROUND DESIGN		
		TITLE	1940 LEVEL DESIGN PLAN VIEW		
THIS DRAWING MIGHT BE REDUCED FROM ITS ORIGINAL SIZE DO NOT SCALE	SCALE: NTS	DATE: 10-16-13	PROJECT No.: NP13-008	DWG. No.: NP13008-XAG-P-006	REV. No.: A

NORDPRO MINE & PROJECT MANAGEMENT SERVICES
160 LOGAN AVENUE
THUNDER BAY ON P7A 6R1
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1.7.3 Major Mine Equipment

Underground mining will utilize mobile rubber tired diesel powered equipment to mine 2,000 tonnes per day of potentially economic mineralization or the equivalent of 720,000 tonnes per year. 40 tonne haul trucks will be loaded by LHD's at truck loading stations on each level. The truck loading stations will consist of a 20 metre length area with the back height increased to 10 metres to facilitate LHD bucket height for dumping.

1.7.4 Mine Services

The majority of underground infrastructure will be associated with facilities located on the 1860 Level and include a breakdown maintenance shop, main dewatering sumps, fuel and lube stations, explosives magazine, refuge station and storage areas.

Ventilation will be provided to the mine by a Fresh Air Raise (FAR) and the main ramp from surface. A network of lateral development on each level will connect the mining areas to a Return Air Raise (RAR).The mining operation to support the mining equipment fleet will require ventilation air volumes of approximately 210 to 230 cu. metres per second (450,000 to 500,000 cfm). The ventilation system will consist of a push-pull system utilizing the ventilation raises and the main access ramp.

Primary electrical power for the mine will be provided from the main surface substation connected to the outside powerline. The powerline will be connected to a surface substation located near to the mine portal. Power from the main substation will feed the main underground power line, a 500 mcm cable, installed in the main access ramp from surface.

Compressed air will be supplied by 2 compressors in enclosures located in a small covered structure, near the ramp portal. Service water will be sent underground in a pipeline located in the trackless access ramp from surface. This will feed the main distribution lines on the levels, which will send water to the stope access crosscuts. The mine will also have a communications network to provide voice communications and some PLC monitoring within the mine.

1.7.5 Mine Support Facilities

Water collection sumps will be located on each level. The sumps will be located near the point where the ramp and level access crosscuts intersect and will be designed to prevent water entering the ramp from the levels. Main collection sumps will be located on the 1860 Level. The main sump will be comprised of two dirty water sumps and one clear water sump.

A mobile equipment breakdown maintenance shop will be used to perform all breakdown maintenance on mobile mining equipment. The shop will be constructed near the 1860 Level, off the ramp. The shop will consist of a main shop area for one large piece of equipment or a couple of smaller units.

Portable self-contained fuelling stations will be located on levels where mining equipment will be parked. The units have built in isolation doors and fire suppression. A lube bay will be included in the maintenance shop complex.

A main refuge station will be located on the 1940 Level.

All blasting will utilize ANFO explosives. ANFO will be delivered in bulk bags, to explosives magazines. Explosives magazines will be located on the 1860 and 1940 Levels. Detonator magazines will be located near the explosives magazines.

Storage areas, specially constructed for the purpose, for storing mining consumables including pipe and fittings, ground support materials, ventilation supplies, etc. will be developed on the 1860 & 1940 Levels. Portable toilet units equipped with a mine toilet and small sink will be located on appropriate working levels and near to refuge stations.

Surface support facilities will include a main maintenance shop; backfill plant; explosives magazines; laydown yard; mine rescue station; water collection ponds; mine supervision, geology, engineering and administration offices; and power substation.

1.7.6 Net Smelter Return Cutoff Value

The NSR cutoff value of $45.21 per tonne of ore used for the Bilbao Project stope tonnes and grade determination is derived in Table 1-8:

Table 1-8 NSR Cutoff Value

Component	Cost ($/t)
Mining	27.00
Processing	13.21
G&A	5.00
Total	45.21

1.7.7 Potentially Mineable Resource

The potentially mineable underground resource is estimated to be 5.2M tonnes at grades of 2.10 % Zn, 1.40 % Pb and 63.96 grams Ag per tonne. The tonnes and grade include an average dilution of 10 percent, at zero grade, as well as mining losses of 5%. This Preliminary Economic Assessment relies on Indicated Mineral Resources of the sulphide zone (approximately 75 percent of the total sulphide resource tonnes) as well as Inferred Mineral Resources of the sulphide zone.

It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that will enable them to be categorized as Mineral Reserves. Also the cost projections range in accuracy from PEA to Feasibility level. Therefore, there is no guarantee that the economic projections contained in this Preliminary Economic Assessment will be realized.

1.7.8 Development and Production Schedule

The life of mine development schedule is shown in Table 1-9. All development work in waste will be performed by mining contractors. Xtierra personnel will undertake sill development in potentially economic mineralization.

The mine production schedule is shown in Table 1-10. The schedule is based on a production rate of 2,000 tpd of potentially economic mineralization, or 720, 000 tonnes per year. This provides for a mine life of approximately 8 years, mining out the indicated and inferred sulphide resources available.

Table 1-9 Mine Development Schedule

Heading	Year									Total
	-1	1	2	3	4	5	6	7	8	Metres
Ramp - Surface to 2020	1,007									1,007
Ramp - 2020 to 2000	130									130
Ramp - 2000 to 1980	164									164
Ramp - 1980 to 1960	143									143
Ramp - 1960 to 1940	132									132
Ramp - 1940 to 1920		178								178
Ramp - 1920 to 1900					162					162
Ramp - 1900 to 1880					126					126
Ramp - 1880 to 1860					147					147
2020 Level Development	78			485						563
2000 Level Development	99			675						774
1980 Level Development	87	621	429							1,137
1960 Level Development	81	629								710
1940 Level Development	112	550								662
1920 Level Development							696			696
1900 Level Development						1,047				1,047
1880 Level Development						537				537
1860 Level Development					420					420
Vent Raise #1 - 2020 to Surface	124									124
Vent Raise #1 - 2000 to 2020	12									-
Vent Raise #1 - 1980 to 2000	21									-
Vent Raise #1 - 1960 to 1980	13									-
Vent Raise #1 - 1940 to 1960		14								14
Vent Raise #1 - 1920 to 1940					18					18
Vent Raise #1 - 1900 to 1920					15					15
Vent Raise #1 - 1880 to 1900					12					12
Vent Raise #1 - 1860 to 1880					16					16
Exhaust Raise #1 - 2020 to Surface		117								117
Exhaust Raise #1 - 2000 to 2020		15								15
Exhaust Raise #1 - 1980 to 2000		18								18
Exhaust Raise #1 - 1960 to 1980		13								13
Exhaust Raise #1 - 1940 to 1960			16							16
Backfill Raise- 2020 to Surface	129									129
Backfill Raise - 2000 to 2020		17								17
Backfill Raise - 1980 to 2000		26								26
Backfill Raise - 1960 to 1980		18								18
Backfill Raise - 1940 to 1960		19								19
Backfill Raise - 1920 to 1940					23					23
Backfill Raise - 1900 to 1920					20					20
Backfill Raise - 1880 to 1900					17					17
Backfill Raise - 1860 to 1880					21					21
Total Lateral Development	**2,033**	**1,978**	**429**	**1,160**	**855**	**1,584**	**696**	**-**	**-**	**8,735**
Total Raising	**299**	**257**	**16**	**-**	**142**	**-**	**-**	**-**	**-**	**668**

Table 1-10 Life of Mine Production Schedule

Stoping Area	Year									Total
	-1	1	2	3	4	5	6	7	8	Tonnes
1825 Inferred							16,168			**16,168**
1860 Total							74,714			**74,714**
1860 Remnant Total							13,398			**13,398**
1860 Inferred							29,554			**29,554**
1880 Total							165,157			**165,157**
1880 Remnant Total							28,231			**28,231**
1880 Inferred							169,440			**169,440**
1900 Total							157,685	321,541		**479,226**
1900 Remnant Total								45,577		**45,577**
1900 Inferred								211,963		**211,963**
1920 Total								140,918	171,122	**312,040**
1920 Remnant Total									39,480	**39,480**
1920 Inferred									356,080	**356,080**
1940 Total		298,992	466,897							**765,889**
1940Remnant Total			2,123							**2,123**
1940 Inferred			118,204							**118,204**
1960 Total			132,777	570,868						**703,645**
1960Remnant Total				55,415						**55,415**
1960 Inferred					128,303					**128,303**
1980 Total				93,717	452,191					**545,908**
1980Remnant Total						20,650				**20,650**
1980 Inferred					137,612					**137,612**
2000 Total					1,894	510,249				**512,143**
2000 Remnant Total						61,999				**61,999**
2000 Remnant Upholes Total						32,995				**32,995**
2000 Inferred						92,831				**92,831**
2000 Inferred Upholes						1,276	65,653			**66,929**
Total Tonnes	**0**	**298,992**	**720,000**	**720,000**	**720,000**	**720,000**	**720,000**	**720,000**	**566,682**	**5,185,674**
Grades										
Zn		2.37	2.27	2.91	2.34	2.99	1.94	0.96	0.97	
Pb		1.70	1.63	1.88	1.55	1.66	1.07	0.85	0.95	
Cu		0.16	0.16	0.18	0.19	0.19	0.16	0.16	0.12	
Ag		60.08	62.21	68.28	63.90	61.34	68.86	72.65	48.84	

1.7.9 Equipment Fleet Requirements

The mining equipment required to develop the mine and produce 2,000 tonnes per day of potentially economic mineralization is presented in Table 1-11.

Table 1-11 Mine Equipment Fleet

Trackless Equipment	# Units
4.6 cu. m. LHD	3
2 boom E/H Dev. Jumbo	1
102mm L/H Drill	2
40 t Haul Truck	3
Block Holer	1
Scissor Lift (3m wide Platform)	4
Service Truck	1
Grader	1
Maintenance Vehicle with Boom and Basket	1
Personnel Carrier	1
Toyota Hilux (or equiv.)	5
Small Equipment	**# Units**
Stopers	10
Jacklegs	10

1.7.10 Mining Personnel Requirements

All mine manpower, except for the technical staff, will be contractor employees. Manpower estimates for the mine total approximately 186 people. These numbers include mine and surface employees, mine site management, engineers and geology personnel. Table 1-12 shows the mining personnel complement.

Table 1-12 Mining Personnel Compliment

Position	Shifts	Complement D/S	Complement A/S	Complement N/S	Absent Replacement	Complement Per Day	Total Complement
Development Miners	3	12	12	12	4	40	**40**
Longhole Driller	3	2	2	2	1	7	**7**
Longhole Driller Helper	3	2	2	2	1	7	**7**
Blaster	3	1	1	1		3	**3**
Blaster Helper	3	1	1	1	1	4	**4**
Stope LHD Operator	3	3	3	3		9	**9**
40 t Haul Truck operator	3	3	3	3	2	11	**11**
Total Direct Mine		**24**	**24**	**24**	**9**	**81**	**81**

The complement for mine services is estimated to be approximately 20 persons and the maintenance department 33 persons. Table 1-13 shows the mine services complement and Table 1-14 the mine maintenance department complement.

Contractor staff will include a Superintendent, 4 supervisors, a safety coordinator and a clerk. All mine personnel will work three 8 hour shifts, 6 days per week.

The mine owner staff complement of 52 is presented in Table 1-15.

Table 1-13 Mine Services and Support Personnel Complement

Position	Shifts	Complement D/S	Complement A/S	Complement N/S	Absent Replacement	Complement Per Day	Total Complement
Serviceman	3	1	1	1	1	4	**4**
Grader Operator	D	1	1	1		3	**3**
Construction/Backfill Leader	3	1	1	1		3	**3**
Lamproom/Dryman	D	1	1	1		3	**3**
General Labourer	3	2	2	2	1	7	**7**
Total Mine Support Services		**6**	**6**	**6**	**2**	**20**	**20**

Table 1-14 Mine Maintenance Department Complement

Position	Shifts	Complement D/S	Complement A/S	Complement N/S	Absent Replacement	Complement Per Day	Total Complement
Leadhand Mechanic	3	1	1	1		3	**3**
Leadhand Electrician	3	1	1	1		3	**3**
Mobile Mechanic	3	1	1	1		3	**3**
Mechanic	3	1	1	1		3	**3**
Electrician	3	4	1	1		6	**6**
Electrician Helper	D	4				4	**4**
Instrumentation Technician	2	1	1			2	**2**
Instrumentation Helper	2	1	1			2	**2**
Parts Warehouseman	D	1				1	**1**
Welder	2	1	1			2	**2**
Welder Helper	2	2	2			4	**4**
Total Mine Maintenance Manpower		**18**	**10**	**5**	**0**	**33**	**33**

Table 1-15 Mine Staff Complement

Position	Total Complement
Mine Superintendent	1
Maintenance Superintendent	1
Mine Supervisor 1	3
Mine Supervisor 2	16
Maintenance Supervisor	1
Electrical Supervisor	1
Mine Services Supervisor 1	1
Mine Services Supervisor 2	4
Mine Trainer/H&S Coordinator	1
Maintenance Planner	2
Chief Engineer	1
Mine Planning Engineer	2
Ventilation	1
Blasting Engineer	1
Surveyor	2
Surveyor Helper	4
Chief Geologist	1
Mine Geologist	2
Geology Modeller	1
Geological Technicians	2
Muestrarios	4
Total Mine Staff	**52**

1.8 Recovery Methods

The mineral processing plant described is for the treatment of a silver-lead-zinc sulfide ore at a design throughput rate of 2,000 t/d. The mineral processing plant will produce lead-silver and zinc concentrates which will be transported off-site. A general site map showing the location of the plant and sulphide tailings area is shown in Figure 1-3. A simplified process flow diagram for a 2,000 t/d processing rate can be seen in Figure 1-4.

The process flow sheet selected for the Bilbao process plant comprises of two stages of crushing, two stages of grinding, lead rougher flotation, lead regrind, lead cleaner and lead concentrate and dewatering stages, zinc rougher flotation, zinc regrind, zinc cleaner flotation and zinc concentrate and dewatering stages.

Tailings from the zinc flotation circuit is pumped to the tailings thickener to produce a thickened tailings with 65% solids. The thickener is of conventional design with the addition of flocculant. Thickener overflow flows by gravity to the process water tank and thickener underflow is pumped to the tailings treatment facility.

Figure 1-3 General Site Layout

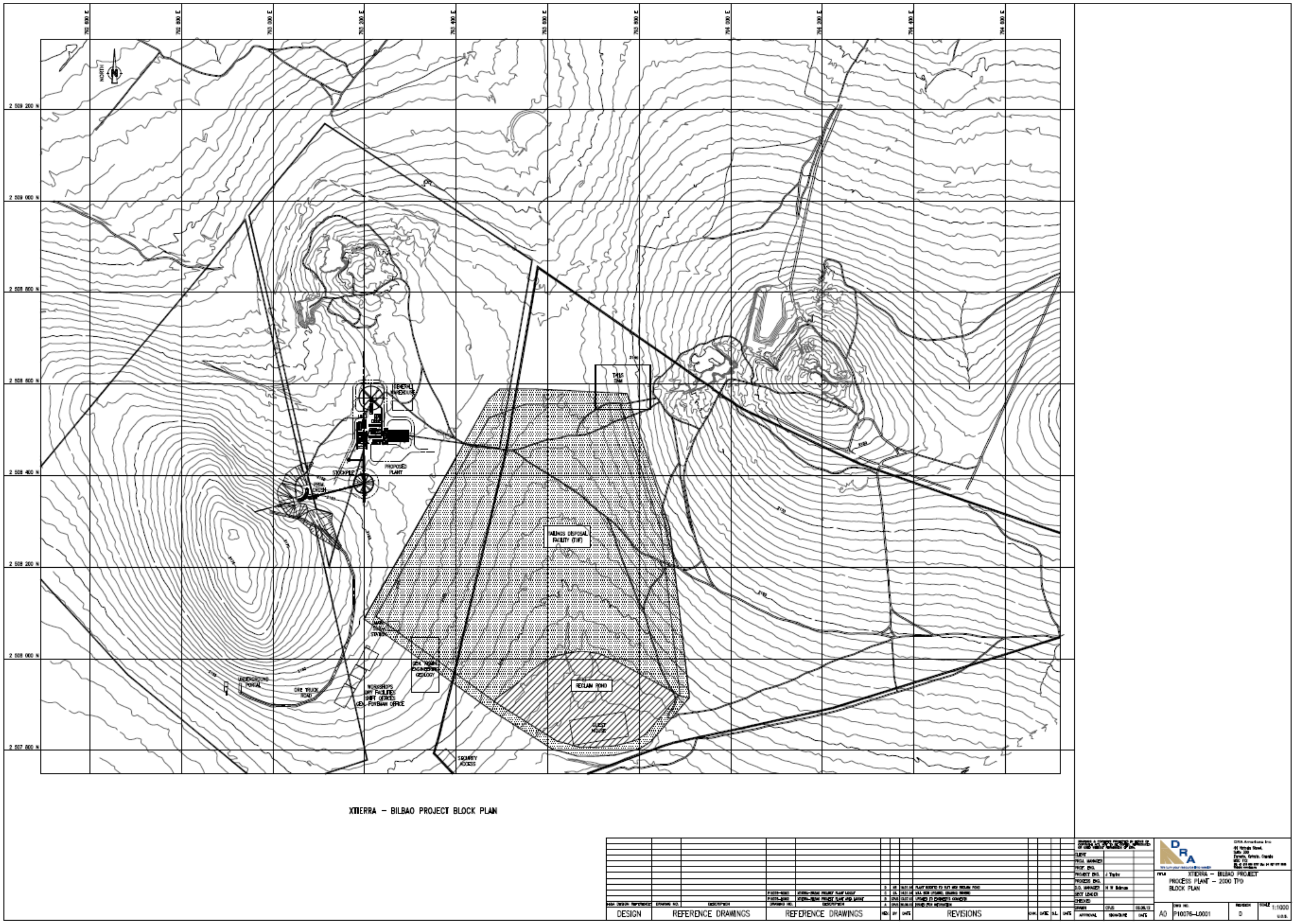
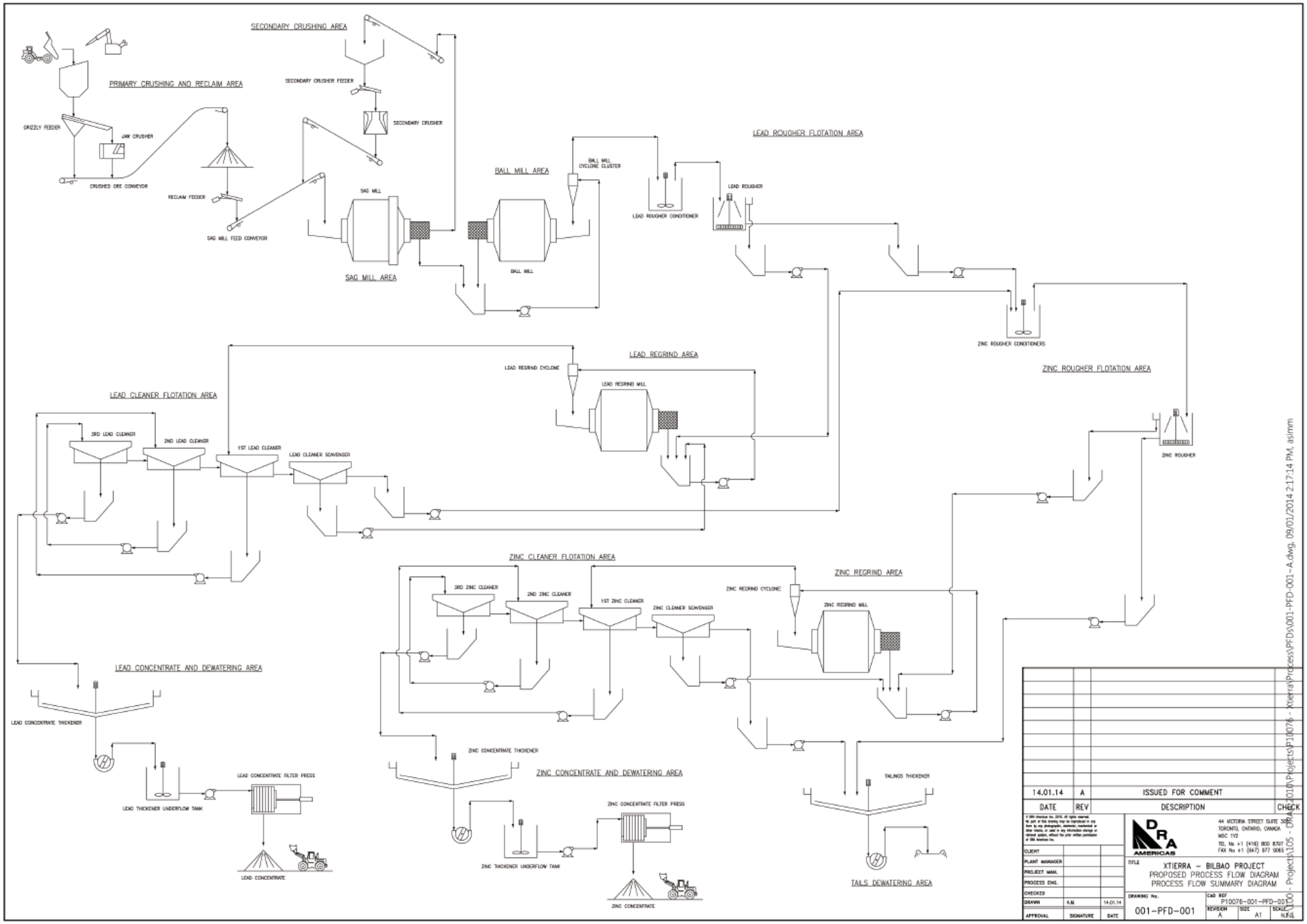


Figure 1-4 Simplified Process Flow Diagram



The plant will be capable of processing 720,000 t/a with an average grade of 2.1%, 1.4% and 63.96 g/t of zinc, lead and silver respectively. The plant has an operating regime of 360 d/a, 7 d/w, 24 h/d and a plant utilization of 92%, resulting in an average nominal throughput of 91 t/h.

The plant will produce, on average, 16,913 dry t/a of silver-rich lead concentrate, and 26,966 dry t/a of zinc concentrate. Plant recovery is estimated to be 76.7% for zinc, 90.6% for lead and 73.4% for silver over the life of the mine.

Table 1-16 provides a summary of key process design criteria used for the process design.

Table 1-16 Process Parameters

Parameter	Unit of Measure	Rate
Plant Capacity		
Annual	thousand tonnes/year	720,000
Daily	tonnes/day	2,000
Ore Grade		
Silver	grams/tonne ore	63.96
Lead	percent	1.40
Zinc	percent	2.10
Operating Parameters		
Crush Size	microns (80% passing)	95,000
Primary Grind Size	microns (80% passing)	100
Pb Regrind Size	microns (80% passing)	43
Zn Regrind Size	microns (80% passing)	37
Reagent Consumptions		
Lime	kilogram/tonne ore	3.06
Sodium Cyanide	kilogram/tonne ore	0.055
Zinc Sulphate	kilogram/tonne ore	0.385
Copper Sulphate	kilogram/tonne ore	0.525
A211	kilogram/tonne ore	0.043
Aerophine 3418A	kilogram/tonne ore	0.020
MIBC	kilogram/tonne ore	0.053
Unifroth 250	kilogram/tonne ore	0.038
Aero 3894	kilogram/tonne ore	0.0025
Flocculant	kilogram/tonne ore	0.051
Flotation Lab Time		
Pb Circuit		
Pb Rougher	minutes	6
1st Pb Cleaner	minutes	3
2nd Pb Cleaner	minutes	6
3rd Pb Cleaner	minutes	6
1st Pb Cleaner-Sca	minutes	3
Zn Circuit		
Zn Rougher	minutes	6
1st Zn Cleaner	minutes	3
2nd Zn Cleaner	minutes	6
3rd Zn Cleaner	minutes	3
1st Zn Cleaner-Scav	minutes	3

Parameter	Unit of Measure	Rate
Recovery		
Lead Concentrate		
Silver	percent	73.4
Lead	percent	90.6
Zinc	percent	5.7
Zinc Concentrate		
Silver	percent	6.7
Lead	percent	0.8
Zinc	percent	76.7
Concentrate Grade		
Lead Concentrate		
Silver	grams/tonne concentrate	1,335.00
Lead	percent	54.00
Zinc Concentrate		
Silver	grams/tonne concentrate	91.40
Zinc	percent	43.00
Production		
Lead Concentrate	dry tonnes/year	16,913
Contained Silver	thousand troy ounces/year	726
Contained Lead	pounds/year	20,135,243
Zinc Concentrate	dry tonnes/year	26,966
Contained Silver	thousand troy ounces/year	79
Contained Zinc	pounds/year	25,563,714

1.8.1 Process Plant Personnel Requirements

Manpower estimates for the process plant total approximately 74 people. These numbers include management, administration, laboratory, process and maintenance personnel. Table 1-17 shows the plant personnel complement.

Table 1-17 Process Plant Personnel Complement

	Area	Responsibility	Grade	No.
Management				
Process Superintendent	Management	Plant	E1	1
Maintenance General Manager	Management	Engineering	D3	1
Laboratory Supervisor	Management	laboratory	D1	1
Process General Foreman	Management	Process	D3	1
Metallurgist	Management	Process	D1	1
		Total		**5**
Administrative				
Site Administrator	Administration	Secretary	B5	1
Materials Controller	Administration	Stores	C1	1
Stores Controller	Administration	Stores	B3	1
Stores Assistant	Administration	Stores	A3	0
Buyer	Administration	Stores	B3	1
Cost Control & Data Clerk	Administration	Driver	A3	1
Admin Clerk / HR Assistant	Administration	Training	B5	0
Safety & System Coordinator	Administration	Safety	C4	1
Loss Prevention Officer	Administration	Safety	B5	0
		Total		**6**
Laboratory				
Laboratory Analysts	Services	Laboratory	C3	4
Laboratory Sampler	Services	Laboratory	B3	4
		Total Laboratory		**8**
Process				
Shift Supervisor	Process	Process Plant	C4	4
Process Operator	Process	Process Plant	C1	4
Control Room Operator	Process	Process Plant	C2	4
Process Operator	Process	Process Plant	B3	4
Process Labourer	Process	Process Plant	A3	16
Equipment Operator	Process	Casuals	B3	4
Driver	Process	Casuals	A3	2
EIT / Coop	Process	Plant	B5	0
		Total Process		**38**
Maintenance				
Maint Coordinator	Engineering	Plant	C1	1
Mechanical Foreman	Engineering	Mechanical	C4	1
Welder	Engineering	Mechanical	C1	3
Welder Apprentice	Engineering	Mechanical	A3	0
Pipe Fitter	Engineering	Mechanical	C1	0
Pipe Fitter Apprentice	Engineering	Mechanical	A3	0
Millwright / Mechanic	Engineering	Mechanical	C1	6
Millwright Apprentice	Engineering	Mechanical	A3	0
Electrical Foreman	Engineering	Electrical	C4	1
Electricians	Engineering	Electrical	C1	3
Electrical Apprentice	Engineering	Electrical	A3	0
Instrumentation	Engineering	Instrumentation	C3	2
Instrumentation Apprentice	Engineering	Instrumentation	A3	0
		Total Maintenance		**17**
		Total Staff Complement		**74**

1.9 Environmental, Permitting and Social or Community Impact

1.9.1 Environmental Studies

Several environmental or environmentally-related studies have been developed for the Bilbao project. These studies have provided detail on biodiversity baseline conditions, groundwater resources available in the region, and information on the potential for the Project to result in environmental contamination. Existing studies are summarized in the following subsections.

1.9.1.1 Biodiversity Studies

The Mexican environmental consultancy Bufete de Servicios Tecnicos Forestales y de Fauna Silvestre prepared a 2006 study entitled Bilbao Project, Biologic, Climatic and Access Route Aspects. This study presented baseline flora and fauna data for the project site, as well as climate information (average rainfall and temperatures). Six species of cacti were identified which have legal protection status. A subsequent report by the same firm entitled "Aviso de Apego a la NOM-120-SEMARNET-1997, Para Actividades de Explotacion Minera del Proyecto Bilbao" was filed with SEMARNET in 2006, detailing efforts that would be undertaken by the Project to avoid any sensitive cacti during exploration drilling, and to reclaim drilling pad locations. These mitigations have been implemented and appropriate rehabilitation is undertaken at the completion of all exploration drilling activity. Additional biodiversity studies will be detailed in the MIA.

1.9.1.2 Hydrology Studies

Climate records available from the nearby climate stations provide an historical average precipitation rate of approximately 412 mm/year, and average evapotranspiration rate at 1,486 mm/year, or approximately 3.6 times the rate of precipitation.

In 2009 Bilbao Resources contracted with Schlumberger Water Services ("SWS") to characterize and identify a potential water source for the Project. The results of this initial investigation concluded that the Project would need to rely exclusively on ground water for needed make-up water during operations, as very little surface water exists in the region.

A follow-up Phase 2 Hydrologic Assessment was issued by SWS in July, 2011 entitled "Draft Phase 2 Hydrogeologic Assessment". The Phase 2 Hydrologic Assessment identified a total of 184 production wells in existence within a study area of 10km from the Project site. It is noted in the study that from the time period of 1997 – 2007 water levels dropped between 0.4 to 1.8 meters/year in the Ojo Caliente aquifer, reflecting a high rate of overexploitation.

Since the early 1960's there has been a ban on additional groundwater exploration in the Municipality of General Panfilo Natera. As a result the Project will be required to purchase water rights from existing users. There are two options to obtain these water rights: (1) purchase an existing well that has been previously permitted, then pipe the water to the Project site; or (2) purchase a permitted well, then transfer the groundwater concession rights for this well to a new well located nearer to the mine site. The second option requires identification of a suitable location for pumping of groundwater, then purchase and transfer of an existing concession to allow production to occur at the newly identified location.

1.9.1.3 Geochemistry

Geochemical modelling of waste rock samples has been performed to identify the potential for acid rock drainage. A total of 19 waste rock samples were collected by Xtierra geologists from boreholes drilled to intersect the proposed ramp to underground works.

Acid Base Accounting (ABA) testing was performed on the 19 waste rock samples using criteria identified in the Mining Environment Neutral Drainage (MEND) Program promulgated, by Natural Resources Canada (2009). These samples included eight limestone, six granite, and five sandstone country rock origins. All samples were characterized with low sulphide concentrations (between <0.01 and 0.35 weight percentage as sulphide). As a result acid generation is identified as an insignificant issue, as the Neutralization Potential Ratio (NPR) of all samples was greater than two.

The results provided in the Geochemical Results of Waste Rock and Tailings Samples Report are preliminary, and additional sampling is required to assess short-term and potential long-term metal leaching characteristics of the waste rock. Further sampling may be required to ensure conformance with best practice sampling guidance.

A total of 283 kg of tailings from pilot testing of the sulfide ore was provided to Golder in May, 2013. Testing for tailings solids included elemental analysis, ABA and net acid generation (NAG) testing, and short-term leaching potential. The acid generation potential was found to be variable depending on the method of assessment, and the testing program completed to date suggests that tailings should be assumed to generate acidity.

Short-term leach tests for tailings material indicated barium, manganese and zinc may leach at concentrations that are greater than the applicable Mexican Standards for Receiving Body of Water. In addition, NAG leach testing results indicate barium and manganese may leach at concentrations that exceed Mexican regulatory standards. Tailings process decant water (supernatant) quality has been modelled with indications that total ammonia, beryllium, manganese, selenium, and zinc concentrations would exceed applicable Mexican regulatory criteria. Given these findings, the Project plans to construct the tailings disposal facility (TDF) with a HDPE liner to prevent infiltration into groundwater.

1.9.2 Waste and Tailings Disposal, Site Monitoring and Water Management

The TDF will be constructed with an HDPE liner to prevent seepage to underlying soils and groundwater. Tailings discharged to the TDF will be thickened to approximately 65% solids prior to disposal. As a result significant amounts of water will be recycled through the process circuit. The TDF will have a single tailings cell with enough capacity to contain the estimated 3.4 million m^3 of tailings material to be generated over the life of the Project. The configuration of the TDF is shown in Figure 1-5. The perimeter dams for TDF cell will be constructed with rockfill. A settling pond will be allowed to form at the toe of tailings beach, and upstream of the separation (South) dam. This will allow settling of finer material prior to discharge to a reclaim pond which will be used to recycle water back to the processing circuit.

The rockfill berms (dams) will be constructed in conformance with the Canadian Dam Association's Dam Safety Guidelines (2007), which will be used to guide design criteria for slope stability, necessary freeboard to accommodate flood events, and earthquake stability. The TDF has been designed to accommodate an Environmental Design Flood (EDF), which is a 1,000 year return, 24 hr event (73.2 mm).

1.9.2.1 Water Balance

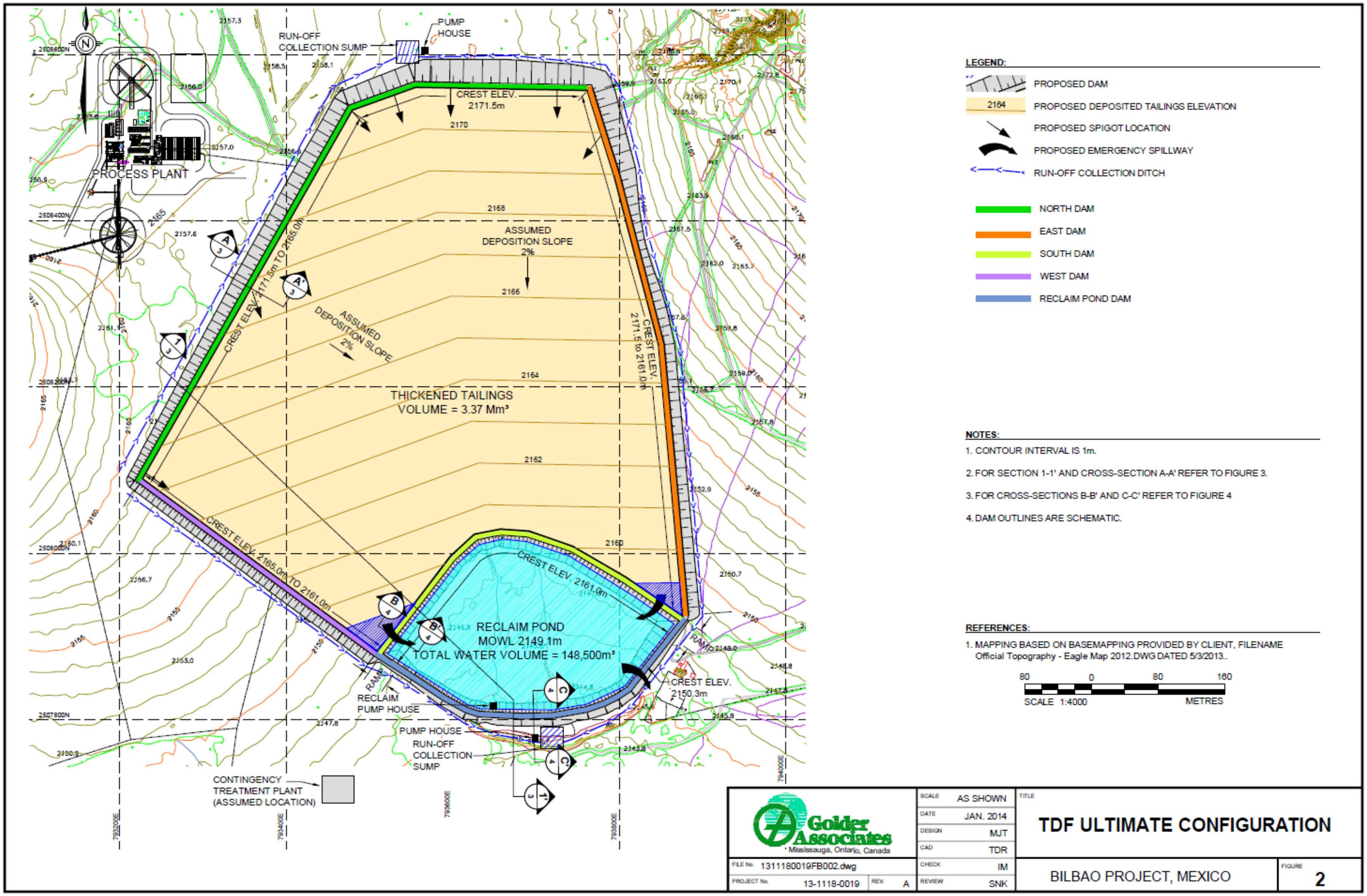
All runoff at the plant site (i.e., contact water) will be captured in drainage channels and routed to a mill runoff pond. The mill runoff pond will also receive as input treated domestic effluent, excess water generated and direct precipitation and water from dewatering the mine. Water collected in the mill runoff pond will be pumped to the reclaim pond for use in the process circuit.

All runoff at the TDF perimeter, as well as collected seepage, will be routed to runoff collection ditches which lead to runoff collection sumps located to the north and south of the TDF. Runoff collection ditches are designed to accommodate peak flows associated with a 24-hour precipitation event with a 100-year recurrence interval. Water collected at the two runoff collection sumps will be diverted to the reclaim pond for use in the process circuit.

Therefore, flows available in the reclaim pond will consist of the following:

- any excess supernatant in the TDF;
- input from the mill runoff collection pond, including mine water;
- input from the runoff collection sumps; and
- direct input from precipitation.

Figure 1-5 Tailings Disposal Facility Configuration



Losses to the reclaim pond water balance will include evaporation, seepage, and any water used for dust control. The remainder will be available for use as make-up water in the processing circuit.

The supplemental Prefeasibility TDF Study Update provides the anticipated water balance based on return of all tailings to the TDF (i.e., no use of paste backfill). For steady state mining operations and under average climatic conditions the mill will require 238,134 m^3 of water on an annual basis. Resultant annual water balance calculations for the Project are summarized in Table 1-18 for average, 25-year wet, and 25-year dry precipitation conditions.

Table 1-18 Summary of Annual Water Balance for Various Precipitation Conditions

Water Balance	**25- year Wet Conditions** (662.7 mm precipitation; 1,227.9 mm evaporation)	**Average Conditions** (412.3 mm precipitation; 1,486.1 mm evaporation)	**25-year Dry Conditions** (197.4 mm precipitation; 1,756.4 mm evaporation)
Make-up Water Required at Mill (in m^3)	None	107,101	245,459
Accumulation of Excess Water (in m^3)	56,108	None	None

Based on this updated water balance Golder estimates an annual water deficit in the Project process circuit of 107,101 m^3 under average meteorological conditions. Excess water may accumulate under 25-year wet meteorological conditions, and 245,459 m^3 of make-up water would be required under 25-year dry meteorological conditions.

Other conclusions from the water balance analysis include the following:

- During average precipitation years (2-year return period) and for wet years with a return period of 5 years or less make-up water will be required to support the mill.
- During exceptionally dry years with a 100-year return period approximately 270,000 m^3 of make-up water will be required to support the mill.
- During a wet year with a return period of 10 years or more there will be an accumulation of water beyond that needed to support the mill. A contingency plan for water storage would then be required.

Any water deficit will need to be addressed via supply from external water sources. The deficit may be less if actual underground mine inflow rates are greater than the 50 m^3/day currently anticipated. The Tailing Disposal Facility and Water Management Pre-Feasibility Report recommends that water be stored prior to commissioning and operation of the Project. The TDF conceptual design includes construction of the reclaim pond during the start-up phase of the Project.

Exploratory drilling for a suitable water source will be pursued in target zones identified in the 2011 Phase 2 Hydrologic Study. This exploratory drilling will include hydrologic pump tests to verify suitability of the identified resource over time. As mentioned previously water rights will need to be purchased or transferred from existing users in the region, as there is a long-standing ban on further groundwater withdrawal from the limited aquifers of Municipality of General Panfilo Natera.

1.9.3 Permitting

Prior to construction all mining projects must first prepare an MIA and Environmental Risk Study ("Estudio de Riesgo Ambiental" and "ERA"). The MIA and ERA studies detail results of baseline studies, characterize potential environmental and social impacts, and identify appropriate mitigations. In addition management plans and monitoring programs are identified to ensure successful environmental and social performance. These completed studies are jointly submitted to SEMARNET, which then reviews the document and either rejects or accepts the MIA with corresponding conditions of approval in a Resolution Letter (the "Resolucion").

In addition to the MIA Resolution Letter a project must also obtain a Change in Land Use ("Cambio de Uso de Suelos" or "CUS") permit which is granted after submission and approval of a technical study justifying the change in land use of the project area from its current use to development of a mine. The CUS permit has an associated cost, based on the current land use of the area to be developed.

On March 27, 2013 Bilbao Resources, S.A. de C.V. contracted with the Mexican environmental consultancy SIICA to complete the MIA and ERA. At the time of writing these documents were under development, incorporating information from existing environmental studies as well as studies that are in progress. In addition to the MIA and ERA the environmental consultancy SIICA will assist in development of the required technical study to issue the CUS, and a Program for the Prevention of Accidents (PPA).

1.9.4 Social and Community Impact

Details of potential social and community impact will be addressed in the pending MIA. The State of Zacatecas has experience centuries of mining development, and anticipated impacts to the Project area of influence are expected to be positive including employment opportunities.

1.9.5 Closure Planning

Per regulatory requirement the MIA will contain information for how closure and reclamation will be accomplished at the end of mining. Typical design features include the channelling of surface waters into natural drainages, and scarifying and reseeding of waste rock features. Down gradient monitoring of water quality will be performed to ensure no remnant groundwater contamination is present. Conceptual closure information for the TDF is provided in the Tailing Disposal Facility and Water Management Pre-Feasibility Report. A 0.5 m thick compacted sand and gravel cover will be emplaced over the entire tailing surface and runoff sumps will be decommissioned. A small wetland will be allowed to form upstream of the reclaim pond dam to allow for sedimentation and evaporation of accumulated surface runoff.

1.10 Market Studies and Contracts

1.10.1 Market Studies

Micon International Limited has reviewed indicative terms and conditions from MRI Trading AG (MRI) of Zug, Switzerland, relating to the delivery of zinc and lead concentrates from the Bilbao project to the port of Manzanillo, Mexico. The terms and conditions are dated 18 May, 2012. Micon considers that the document demonstrates that, based on the typical specifications put forward by Xtierra to MRI, the zinc and lead concentrates are likely to be saleable and acceptable to smelters. The report by ConsuMet, dated 19 April, 2013 concluded that the analysis of concentrate samples from the locked cycle test did not indicate the presence of elements of concern in terms of concentrate marketing.

Micon has not undertaken a formal market study relating to potential metal production from the Bilbao project but has provided background around historical global demand and production for zinc, lead and silver products. This is summarized in Section 19.

1.10.2 Prices Used for Economic Analysis

At the request of Xtierra, RPM has based its economic analysis of the Bilbao project on three-year average metal prices. For the three-year period ending 31 October, 2013, the rolling average prices based on LME cash buyer quotes for zinc and lead, and as reported by Kitco on www.kitco.com for silver are as follows:

Zinc	US$0.92/lb
Lead	US$1.00/lb
Silver	US$30.38/oz

In order to test the sensitivity of the Bilbao project to metal demand and, therefore, decreased and increased metal prices over the projected life of the operation, reductions and increases on the three-year average prices are evaluated in Section 22.

At the time of writing of this report, Micon understands that there are no contracts in place that are material to the issuer relating to property development or marketing of concentrates from the Bilbao project.

1.11 Capital and Operating Costs

Project Capital Costs, as of April 2014, are estimated to be USD 99.5M including an allowance for contingencies of USD 8.7M, equivalent to 8.8% of total capital expenditure. The capital cost summary as presented in Table 1-19 outlines total pre-production capital of USD 91.2M and remaining other capital and sustaining capital costs of USD 8.3M for the 8 year production life, including acquisition to replace mine equipment fleet, plant and infrastructure.

Table 1-19 Capital Cost Summary - USD

Capital Expenditures	Pre-Production Year -1	LOM Production Year 1-8	Total
Exploration	600,000	-	600,000
Mine Facilities & Equipment	11,529,000	-	11,529,000
Mining Equipment - Leased		-	-
U/G Mine Development	3,509,000	3,229,000	6,738,000
Backfill Plant & Distribution System	500,000	600,000	1,100,000
Infrastructure	6,942,462	-	6,942,462
Surface Mobile Equipment	700,000	-	700,000
Processsing Plant	38,321,221	-	38,321,221
Tailings Disposal Facility	6,615,067	4,694,080	11,309,147
EPCM & Contractor O/H	10,318,448	-	10,318,448
Owners Costs	3,980,000	-	3,980,000
Reclamation and Closure		1,181,000	1,181,000
Working Capital	2,017,503	- 2,017,503	- 0
Additional Contingency	6,138,247	660,858	6,799,105
Total Capital Expenditures	**91,170,948**	**8,347,435**	**99,518,383**

The operating expenditure is based on all development work in waste being performed by contractors, and stope development by Xtierra personnel and equipment fleets. The strategy was determined as the most cost effective for the operation and ensures sustainability of a skilled labor force.

The average total unit cost for the operational activities is USD 66.90/t of ore. The breakdown of mining, processing, general and administration, freight and insurance, smelting, refining and penalties is presented in Table 1-20.

Table 1-20 Average Unit Operating Cost

Operating Cost	USD/Tonne ROM
Mine	25.73
Process	13.21
Site G&A	5.00
Freight and Insurance	2.08
Smelting, Refining, Penalties	20.88
Total Unit Operating Cost	**66.90**

The lifetime annual average of all operating costs included from Years 1 to 8 amounts to USD 43.4M.

Mining and Process Plant operating costs are largely variable per tonne of product while the General and Administrative costs are fixed per year. RPM has reviewed the basis of the operating cost estimate and considers the costs to be appropriate for evaluating economic viability of the project.

1.12 Economic Analysis

The economic analysis was completed for a 720,000 tonne per year processing plant capacity and is based on the mineable resources outlined in Table 1-21.

This preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The market prices projected in the cash flow analysis for zinc, lead and silver are based on USD 0.92/lb, 1.00/lb and 30.38/oz respectively.

Total revenue for the project is based on 720 kt/y production to be reached in production period 2 and continuing for the life of the project average USD 73.5 million per year (gross revenue). The current plan estimates 11k tonnes of zinc concentrate and 7k tonnes of lead concentrate in the first production year.

1.12.1 Pre-Tax Cash Flow

A pre-tax cash flow was determined excluding corporate tax, profit sharing and mining duty payable to the Mexican government.

Pre-tax earnings total USD 59.9 million over the 8 year designated mine life. Economic results of the Project cash flow model indicate in Internal Rate of Return (IRR) of 13.2% and a Net Present Value (NPV) of USD 11.0M at a 10% discount rate. The ten percent discount rate is considered appropriate for this evaluation as the overall project risks are considered to be relatively low in terms of total capital committed, geological risk and market risk.

The pre-tax cash flow can be seen in Table 1-21.

Table 1-21 Pre-Tax Project Cash Flow

	Production Period Financial Year Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Production											
Ore Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Total Tonnes Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Processed Grades											
Zinc	%		2.37%	2.27%	2.91%	2.34%	2.99%	1.94%	0.96%	0.97%	2.10%
Lead	%		1.70%	1.63%	1.88%	1.55%	1.66%	1.07%	0.85%	0.95%	1.40%
Silver	g/t		60.08	62.21	68.28	63.90	61.34	68.86	72.65	48.84	63.96
Contained Metal											
Zinc	lb		15,602,022	36,052,764	46,167,102	37,142,844	47,462,359	30,829,074	15,285,653	12,063,275	240,605,091
Lead	lb		11,196,280	25,829,904	29,788,348	24,663,037	26,280,765	17,021,685	13,558,841	11,924,471	160,263,330
Silver	oz		577,491	1,440,065	1,580,649	1,479,163	1,419,872	1,594,069	1,681,793	889,747	10,662,850
Mill Recovery											
Zinc	76.7%		76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%
Lead	90.6%		90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%
Silver	73.4%		73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%
Recovered Metal											
Zinc	lb		11,966,751	27,652,470	35,410,167	28,488,561	36,403,629	23,645,900	11,724,096	9,252,532	184,544,105
Lead	lb		10,143,829	23,401,893	26,988,243	22,344,712	23,810,373	15,421,647	12,284,310	10,803,570	145,198,577
Silver	oz		423,878	1,057,008	1,160,197	1,085,706	1,042,186	1,170,047	1,234,436	653,075	7,826,532
Concentrate Production											
Zinc Concentration Ratio	26.70		26.70	26.70	26.70	26.70	26.70	26.70	26.70	26.70	
Zinc Concentrate Produced (tonnes)			11,198	26,966	26,966	26,966	26,966	26,966	26,966	21,224	194,220
Lead Concentration Ratio	42.57		42.57	42.57	42.57	42.57	42.57	42.57	42.57	42.57	
Lead Concentrate Produced (tonnes)			7,024	16,913	16,913	16,913	16,913	16,913	16,913	13,312	121,815
Payability of Metal - HG Zn											
Zinc	85%		85%	85%	85%	85%	85%	85%	85%	85%	
Lead	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Silver	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Payable Metal											
Zinc	lb		10,171,738	23,504,599	30,098,642	24,215,277	30,943,085	20,099,015	9,965,481	7,864,652	156,862,489
Lead	lb		9,636,638	22,231,799	25,638,831	21,227,476	22,619,855	14,650,564	11,670,094	10,263,392	137,938,648
Silver	oz		402,684	1,004,158	1,102,187	1,031,420	990,077	1,111,544	1,172,714	620,421	7,435,205
METAL PRICES											
Zinc	$0.9229		$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229
Lead	$1.0047		$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047
Silver	$30.3761		$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761
Revenue From Metal Sales											
Zinc	US$		$9,387,497	$21,692,395	$27,778,037	$22,348,279	$28,557,373	$18,549,381	$9,197,143	$7,258,287	$144,768,391
Lead	US$		$9,681,930	$22,336,288	$25,759,333	$21,327,245	$22,726,168	$14,719,422	$11,724,944	$10,311,630	138,586,960
Silver	US$		$12,231,978	$30,502,392	$33,480,138	$31,330,526	$30,074,664	$33,764,381	$35,622,492	$18,845,964	225,852,534
Total Sales Revenue	**US$**		**$31,301,405**	**$74,531,075**	**$87,017,508**	**$75,006,050**	**$81,358,205**	**$67,033,184**	**$56,544,578**	**$36,415,881**	**$509,207,885**

Table 1-21 Pre-Tax Project Cash Flow (cont.)

	Production Period	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
	Financial Year	0	1	2	3	4	5	6	7	8	
	Unit										
Operating Costs											
Exploration - Definition Drilling	US$0.58/t		$173,415	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$328,675	$3,007,691
Mobile Mine Equipment Leasing	US$		$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$2,741,894	$24,665,894
U/G Mining - Development	US$			$653,000	$1,722,000	$1,474,000	$2,352,000	$1,034,000	$0	$0	$7,235,000
U/G Mining - Ore	US$		$7,722,000	$15,534,000	$18,381,000	$11,728,000	$15,717,000	$11,182,000	$12,944,000	$5,287,000	$98,495,000
Processing	$13.21		$3,949,684	$9,511,207	$9,511,198	$9,511,205	$9,511,194	$9,511,203	$9,511,198	$7,485,867	$68,502,757
General and Administration	$5.00		$1,494,960	$3,600,002	$3,599,999	$3,600,002	$3,599,998	$3,600,001	$3,599,999	$2,833,409	$25,928,371
Concentrate Transportation - Zinc	$35.00		$391,937	$943,821	$943,820	$943,821	$943,820	$943,821	$943,820	$742,841	$6,797,700
Concentrate Transportation - Lead	$27.00		$189,636	$456,660	$456,660	$456,660	$456,659	$456,660	$456,660	$359,418	$3,289,011
Insurance - Zinc	$2.72		$30,459	$73,348	$73,348	$73,348	$73,348	$73,348	$73,348	$57,729	$528,278
Insurance - Lead	$1.51		$10,606	$25,539	$25,539	$25,539	$25,539	$25,539	$25,539	$20,101	$183,941
Smelting - Zinc ($/tonne conc.)	$205.00		$2,295,631	$5,528,094	$5,528,089	$5,528,093	$5,528,087	$5,528,092	$5,528,089	$4,350,928	$39,815,102
Smelting - Zinc penalty ($/tonne Conc.)	$8.85		$99,104	$238,652	$238,652	$238,652	$238,652	$238,652	$238,652	$187,833	$1,718,847
Refining - Zinc ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Smelting - Lead ($/ tonne conc.)	$300.00		$2,107,061	$5,073,999	$5,073,995	$5,073,998	$5,073,993	$5,073,997	$5,073,995	$3,993,530	$36,544,568
Refining - Lead ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Refining - Silver ($/ounce)	$4.06		$1,634,898	$4,076,880	$4,474,879	$4,187,566	$4,019,711	$4,512,870	$4,761,221	$2,518,908	$30,186,933
Total Operating Costs	**US$**	**$0**	**$23,231,392**	**$49,264,802**	**$53,578,778**	**$46,390,484**	**$51,089,600**	**$45,729,782**	**$46,706,120**	**$30,908,134**	**$346,899,093**
Unit Operating Costs											
Mine	US$/tonne ore		$36.88	$27.41	$32.85	$23.27	$30.03	$21.90	$22.91	$14.75	$25.73
Process	US$/tonne ore		$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21
Site G&A	US$/tonne ore		$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00
Freight and Insurance	US$/tonne ore		$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08
Smelting, Refining, Penalties	US$/tonne ore		$20.52	$20.72	$21.27	$20.87	$20.64	$21.32	$21.67	$19.50	$20.88
Total Unit Operating Cost	**US$/tonne ore**		**$77.70**	**$68.42**	**$74.41**	**$64.43**	**$70.96**	**$63.51**	**$64.87**	**$54.54**	**$66.90**
Net Smelter Return											
NSR Zinc	US$		$6,600,824	$14,981,828	$21,067,476	$15,637,714	$21,846,815	$11,838,817	$2,486,582	$1,976,685	$96,436,742
NSR Lead	US$		$7,385,233	$16,805,629	$20,228,679	$15,796,587	$17,195,516	$9,188,765	$6,194,289	$5,958,682	$98,753,381
NSR Silver	US$		$10,597,079	$26,425,512	$29,005,259	$27,142,960	$26,054,953	$29,251,512	$30,861,271	$16,327,056	$195,665,602
Total Net Smelter Return	**US$**	**$0**	**$24,583,137**	**$58,212,969**	**$70,301,414**	**$58,577,261**	**$65,097,284**	**$50,279,093**	**$39,542,142**	**$24,262,423**	**$390,855,724**
Royalties											
Royalty (1.5% NSR) - Minera Portree	US$			$873,195	$1,054,521	$878,659	$976,459	$754,186	$593,132	$363,936	$5,494,089
Total Operating Cost Including Royalties	**US$**		**$23,231,392**	**$50,137,997**	**$54,633,299**	**$47,269,143**	**$52,066,059**	**$46,483,969**	**$47,299,253**	**$31,272,071**	**$352,393,182**
Operating Income		**$0**	**$8,070,013**	**$24,393,078**	**$32,384,208**	**$27,736,907**	**$29,292,146**	**$20,549,215**	**$9,245,325**	**$5,143,811**	**$156,814,703**

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Table 1-21 Pre-Tax Project Cash Flow (cont.)

	Production Period	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
	Financial Year	0	1	2	3	4	5	6	7	8	
	Unit										
Capital Expenditures											
Initial Capital	US$										
Exploration	US$	$600,000									$600,000
Mine Facilities & Equipment	US$	$11,529,000									$11,529,000
Mining Equipment - Leased	US$										$0
U/G Mine Development	US$	$3,509,000	$3,229,000								$6,738,000
Backfill Plant & Distribution System	US$	$500,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$1,100,000
Infrastructure	US$	$6,942,462									$6,942,462
Surface Mobile Equipment	US$	$700,000									$700,000
Processsing Plant	US$	$38,321,221									$38,321,221
Tailings Disposal Facility	US$	$6,615,067		$1,937,767			$2,756,313				$11,309,147
EPCM & Contractor O/H	US$	$10,318,448									$10,318,448
Owners Costs	US$	$3,980,000									$3,980,000
Reclamation and Closure	US$									$1,181,000	$1,181,000
Working Capital	US$	$2,017,503								-$2,017,503	$0
Additional Contingency	US$	$6,138,247	$165,200	$197,527	$3,750	$3,750	$279,381	$3,750	$3,750	$3,750	$6,799,105
Total Capital Expenditures	**US$**	**$91,170,948**	**$3,469,200**	**$2,210,293**	**$78,750**	**$78,750**	**$3,110,695**	**$78,750**	**$78,750**	**-$757,753**	**$99,518,383**
Cost of Capital per tonne ore mined	*US$*										*$19.19*
Depreciation											
Depreciation	US$		**$5,737,962**	**$13,817,545**	**$13,817,532**	**$13,817,542**	**$13,817,527**	**$13,817,539**	**$13,817,532**	**$10,875,203**	**$99,518,383**
Salvage											
Mobile Equipment	US$									$0	$0
Fixed Equipment	US$									$430,554	$430,554
Building / Infrastructure	US$									$2,137,439	$2,137,439
Total Salvage	**US$**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$2,567,993**	**$2,567,993**
EBITA											
EBITA (Annual)	US$	$0	$2,332,051	$10,575,533	$18,566,676	$13,919,365	$15,474,619	$6,731,676	-$4,572,207	-$3,163,400	$59,864,314
EBITA (Cumulative)	US$	$0	$2,332,051	$12,907,584	$31,474,260	$45,393,625	$60,868,244	$67,599,920	$63,027,713	$59,864,314	
Before Tax Cash Flow	US$	-$91,170,948	$4,600,813	$22,182,784	$32,305,458	$27,658,157	$26,181,451	$20,470,465	$9,166,575	$8,469,557	$59,864,314
Discount Rate	10%										
Discount Factor		1.00	0.91	0.83	0.75	0.68	0.62	0.56	0.51	0.47	
Discounted Cash Flow	US$	-$91,170,948	$4,182,557	$18,332,880	$24,271,569	$18,890,894	$16,256,621	$11,555,044	$4,703,903	$3,951,111	$10,973,630
Cumulative Discounted Cash Flow	US$	-$91,170,948	-$86,988,391	-$68,655,511	-$44,383,942	-$25,493,049	-$9,236,427	$2,318,617	$7,022,519	$10,973,630	
Net Present Value	10,973,630										
Internal Rate of Return	13.24%										

RPM developed a sensitivity analysis for the pre-tax cash flow model based on variations in key project elements of metal price, operating and capital costs. The sensitivity of the Project's IRR and NPV to +/- 15 percent changes to key assumptions is shown in Table 1-22.

Table 1-22 Pre-Tax Sensitivity Analysis

Item	NPV (USD Million)	IRR (%)
Base Case	**11.0**	**13.2%**
Capital Cost +15%	-3.7	9.01%
Capital Cost -15%	25.7	18.53%
Operating Cost +15%	-23.0	2.08%
Operating Cost -15%	45.0	21.99%
Sale Price (Zinc) +15%	25.4	17.22%
Sale Price (Zinc) -15%	-3.5	8.93%
Sale Price (Lead) +15%	24.8	17.04%
Sale Price (Lead) -15%	-2.8	9.14%
Sale Price (Silver) +15%	32.6	18.94%
Sale Price (Silver) -15%	-10.6	6.55%
Mill Recovery (Zinc) +15%	25.4	17.22%
Mill Recovery (Zinc) -15%	-3.5	8.93%
Mill Recovery (Lead) +15%	24.8	17.04%
Mill Recovery (Lead) -15%	-2.8	9.14%
Mill Recovery (Silver) +15%	29.7	18.22%
Mill Recovery (Silver) -15%	-7.7	7.52%

Spider charts are shown in Figure 1-6 and Figure 1-7 below for the Project's pre-tax sensitivity to metal prices, capital cost, operating cost, and mill recovery, with key assumptions varying plus and minus 15 percent.

Figure 1-6 Pre-Tax Project NPV

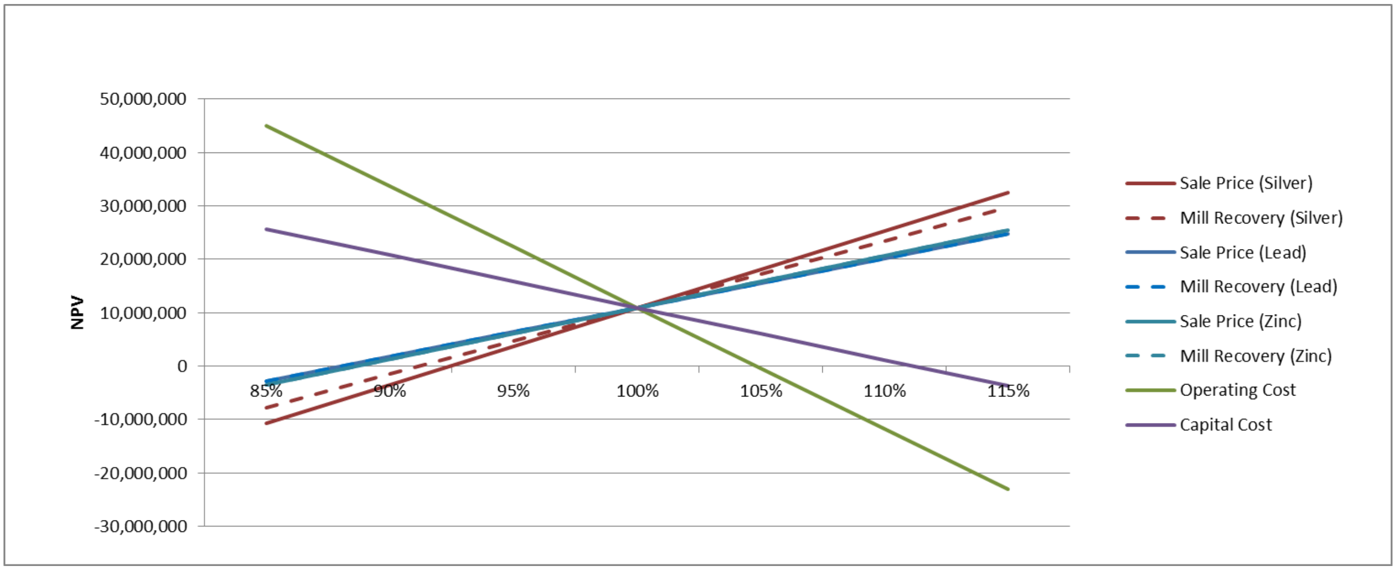
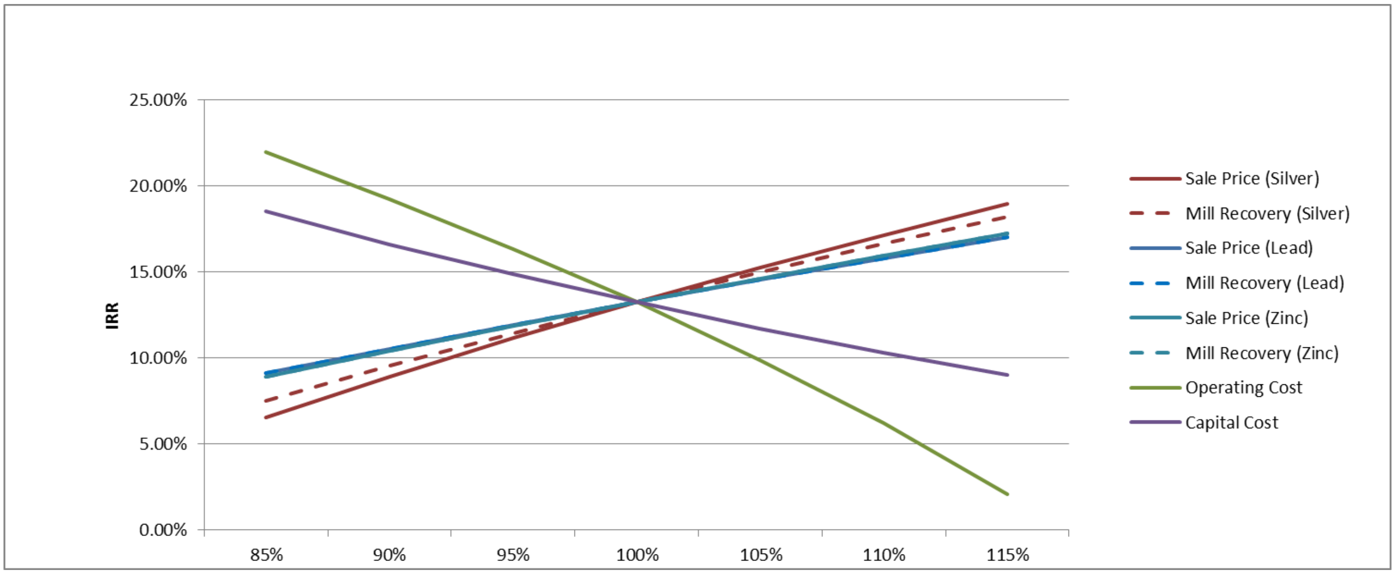


Figure 1-7 Pre-Tax Project IRR



1.12.2 After-Tax Cash Flow

After-tax net cash flow totals USD 32.6 million over the 8 year designated mine life. Economic results of the Project cash flow model indicate in Internal Rate of Return (IRR) of 8.1% and a Net Present Value (NPV) of USD -5.8M at a 10% discount rate. The ten percent discount rate is considered appropriate for this evaluation as the overall project risks are considered to be relatively low in terms of total capital committed, geological risk and market risk.

The after-tax cash flow can be seen in Table 1-23.

RPM developed a sensitivity analysis for the after-tax cash flow model based on variations in key project elements of metal price, operating and capital costs. The sensitivity of the Project's IRR and NPV to +/- 15 percent changes to key assumptions is shown in Table 1-24.

Table 1-23 After-Tax Project Cash Flow

	Production Period Financial Year Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Production											
Ore Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Total Tonnes Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Processed Grades											
Zinc	%		2.37%	2.27%	2.91%	2.34%	2.99%	1.94%	0.96%	0.97%	2.10%
Lead	%		1.70%	1.63%	1.88%	1.55%	1.66%	1.07%	0.85%	0.95%	1.40%
Silver	g/t		60.08	62.21	68.28	63.90	61.34	68.86	72.65	48.84	63.96
Contained Metal											
Zinc	lb		15,602,022	36,052,764	46,167,102	37,142,844	47,462,359	30,829,074	15,285,653	12,063,275	240,605,091
Lead	lb		11,196,280	25,829,904	29,788,348	24,663,037	26,280,765	17,021,685	13,558,841	11,924,471	160,263,330
Silver	oz		577,491	1,440,065	1,580,649	1,479,163	1,419,872	1,594,069	1,681,793	889,747	10,662,850
Mill Recovery											
Zinc	76.7%		76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%
Lead	90.6%		90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%
Silver	73.4%		73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%
Recovered Metal											
Zinc	lb		11,966,751	27,652,470	35,410,167	28,488,561	36,403,629	23,645,900	11,724,096	9,252,532	184,544,105
Lead	lb		10,143,829	23,401,893	26,988,243	22,344,712	23,810,373	15,421,647	12,284,310	10,803,570	145,198,577
Silver	oz		423,878	1,057,008	1,160,197	1,085,706	1,042,186	1,170,047	1,234,436	653,075	7,826,532
Concentrate Production											
Zinc Concentration Ratio	26.70		26.70	26.70	26.70	26.70	26.70	26.70	26.70	26.70	
Zinc Concentrate Produced (tonnes)			11,198	26,966	26,966	26,966	26,966	26,966	26,966	21,224	194,220
Lead Concentration Ratio	42.57		42.57	42.57	42.57	42.57	42.57	42.57	42.57	42.57	
Lead Concentrate Produced (tonnes)			7,024	16,913	16,913	16,913	16,913	16,913	16,913	13,312	121,815
Payability of Metal - HG Zn											
Zinc	85%		85%	85%	85%	85%	85%	85%	85%	85%	
Lead	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Silver	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Payable Metal											
Zinc	lb		10,171,738	23,504,599	30,098,642	24,215,277	30,943,085	20,099,015	9,965,481	7,864,652	156,862,489
Lead	lb		9,636,638	22,231,799	25,638,831	21,227,476	22,619,855	14,650,564	11,670,094	10,263,392	137,938,648
Silver	oz		402,684	1,004,158	1,102,187	1,031,420	990,077	1,111,544	1,172,714	620,421	7,435,205
METAL PRICES											
Zinc	$0.9229		$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229
Lead	$1.0047		$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047
Silver	$30.3761		$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761
Revenue From Metal Sales											
Zinc	US$		$9,387,497	$21,692,395	$27,778,037	$22,348,279	$28,557,373	$18,549,381	$9,197,143	$7,258,287	$144,768,391
Lead	US$		$9,681,930	$22,336,288	$25,759,333	$21,327,245	$22,726,168	$14,719,422	$11,724,944	$10,311,630	138,586,960
Silver	US$		$12,231,978	$30,502,392	$33,480,138	$31,330,526	$30,074,664	$33,764,381	$35,622,492	$18,845,964	225,852,534
Total Sales Revenue	**US$**		**$31,301,405**	**$74,531,075**	**$87,017,508**	**$75,006,050**	**$81,358,205**	**$67,033,184**	**$56,544,578**	**$36,415,881**	**$509,207,885**

Table 1-23 After-Tax Project Cash Flow (cont.)

	Production Period	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
	Financial Year	0	1	2	3	4	5	6	7	8	
	Unit										
Operating Costs											
Exploration - Definition Drilling	US$0.58/t		$173,415	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$328,675	$3,007,691
Mobile Mine Equipment Leasing	US$		$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$2,741,894	$24,665,894
U/G Mining - Development	US$			$653,000	$1,722,000	$1,474,000	$2,352,000	$1,034,000	$0	$0	$7,235,000
U/G Mining - Ore	US$		$7,722,000	$15,534,000	$18,381,000	$11,728,000	$15,717,000	$11,182,000	$12,944,000	$5,287,000	$98,495,000
Processing	$13.21		$3,949,684	$9,511,207	$9,511,198	$9,511,205	$9,511,194	$9,511,203	$9,511,198	$7,485,867	$68,502,757
General and Administration	$5.00		$1,494,960	$3,600,002	$3,599,999	$3,600,002	$3,599,998	$3,600,001	$3,599,999	$2,833,409	$25,928,371
Concentrate Transportation - Zinc	$35.00		$391,937	$943,821	$943,820	$943,821	$943,820	$943,821	$943,820	$742,841	$6,797,700
Concentrate Transportation - Lead	$27.00		$189,636	$456,660	$456,660	$456,660	$456,659	$456,660	$456,660	$359,418	$3,289,011
Insurance - Zinc	$2.72		$30,459	$73,348	$73,348	$73,348	$73,348	$73,348	$73,348	$57,729	$528,278
Insurance - Lead	$1.51		$10,606	$25,539	$25,539	$25,539	$25,539	$25,539	$25,539	$20,101	$183,941
Smelting - Zinc ($/tonne conc.)	$205.00		$2,295,631	$5,528,094	$5,528,089	$5,528,093	$5,528,087	$5,528,092	$5,528,089	$4,350,928	$39,815,102
Smelting - Zinc penalty ($/tonne Conc.)	$8.85		$99,104	$238,652	$238,652	$238,652	$238,652	$238,652	$238,652	$187,833	$1,718,847
Refining - Zinc ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Smelting - Lead ($/ tonne conc.)	$300.00		$2,107,061	$5,073,999	$5,073,995	$5,073,998	$5,073,993	$5,073,997	$5,073,995	$3,993,530	$36,544,568
Refining - Lead ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Refining - Silver ($/ounce)	$4.06		$1,634,898	$4,076,880	$4,474,879	$4,187,566	$4,019,711	$4,512,870	$4,761,221	$2,518,908	$30,186,933
Total Operating Costs	**US$**	**$0**	**$23,231,392**	**$49,264,802**	**$53,578,778**	**$46,390,484**	**$51,089,600**	**$45,729,782**	**$46,706,120**	**$30,908,134**	**$346,899,093**
Unit Operating Costs											
Mine	US$/tonne ore		$36.88	$27.41	$32.85	$23.27	$30.03	$21.90	$22.91	$14.75	$25.73
Process	US$/tonne ore		$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21
Site G&A	US$/tonne ore		$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00
Freight and Insurance	US$/tonne ore		$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08
Smelting, Refining, Penalties	US$/tonne ore		$20.52	$20.72	$21.27	$20.87	$20.64	$21.32	$21.67	$19.50	$20.88
Total Unit Operating Cost	**US$/tonne ore**		**$77.70**	**$68.42**	**$74.41**	**$64.43**	**$70.96**	**$63.51**	**$64.87**	**$54.54**	**$66.90**
Net Smelter Return											
NSR Zinc	US$		$6,600,824	$14,981,828	$21,067,476	$15,637,714	$21,846,815	$11,838,817	$2,486,582	$1,976,685	$96,436,742
NSR Lead	US$		$7,385,233	$16,805,629	$20,228,679	$15,796,587	$17,195,516	$9,188,765	$6,194,289	$5,958,682	$98,753,381
NSR Silver	US$		$10,597,079	$26,425,512	$29,005,259	$27,142,960	$26,054,953	$29,251,512	$30,861,271	$16,327,056	$195,665,602
Total Net Smelter Return	**US$**	**$0**	**$24,583,137**	**$58,212,969**	**$70,301,414**	**$58,577,261**	**$65,097,284**	**$50,279,093**	**$39,542,142**	**$24,262,423**	**$390,855,724**
Royalties											
Royalty (1.5% NSR) - Minera Portree	US$			$873,195	$1,054,521	$878,659	$976,459	$754,186	$593,132	$363,936	$5,494,089
Total Operating Cost Including Royalties	**US$**		**$23,231,392**	**$50,137,997**	**$54,633,299**	**$47,269,143**	**$52,066,059**	**$46,483,969**	**$47,299,253**	**$31,272,071**	**$352,393,182**
Operating Income		**$0**	**$8,070,013**	**$24,393,078**	**$32,384,208**	**$27,736,907**	**$29,292,146**	**$20,549,215**	**$9,245,325**	**$5,143,811**	**$156,814,703**

Table 1-23 After-Tax Project Cash Flow (cont.)

	Production Period Financial Year Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Capital Expenditures											
Initial Capital	US$										
Exploration	US$	$600,000									$600,000
Mine Facilities & Equipment	US$	$11,529,000									$11,529,000
Mining Equipment - Leased	US$										$0
U/G Mine Development	US$	$3,509,000	$3,229,000								$6,738,000
Backfill Plant & Distribution System	US$	$500,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$1,100,000
Infrastructure	US$	$6,942,462									$6,942,462
Surface Mobile Equipment	US$	$700,000									$700,000
Processsing Plant	US$	$38,321,221									$38,321,221
Tailings Disposal Facility	US$	$6,615,067		$1,937,767			$2,756,313				$11,309,147
EPCM & Contractor O/H	US$	$10,318,448									$10,318,448
Owners Costs	US$	$3,980,000									$3,980,000
Reclamation and Closure	US$									$1,181,000	$1,181,000
Working Capital	US$	$2,017,503								-$2,017,503	$0
Additional Contingency	US$	$6,138,247	$165,200	$197,527	$3,750	$3,750	$279,381	$3,750	$3,750	$3,750	$6,799,105
Total Capital Expenditures	**US$**	**$91,170,948**	**$3,469,200**	**$2,210,293**	**$78,750**	**$78,750**	**$3,110,695**	**$78,750**	**$78,750**	**-$757,753**	**$99,518,383**
Cost of Capital per tonne ore mined	*US$*										*$19.19*
Depreciation											
Depreciation	US$		**$5,737,962**	**$13,817,545**	**$13,817,532**	**$13,817,542**	**$13,817,527**	**$13,817,539**	**$13,817,532**	**$10,875,203**	**$99,518,383**
Salvage											
Mobile Equipment	US$									$0	$0
Fixed Equipment	US$									$430,554	$430,554
Building / Infrastructure	US$									$2,137,439	$2,137,439
Total Salvage	**US$**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$2,567,993**	**$2,567,993**
EBITA											
EBITA (Annual)	US$	$0	$2,332,051	$10,575,533	$18,566,676	$13,919,365	$15,474,619	$6,731,676	-$4,572,207	-$3,163,400	$59,864,314
EBITA (Cumulative)	US$	$0	$2,332,051	$12,907,584	$31,474,260	$45,393,625	$60,868,244	$67,599,920	$63,027,713	$59,864,314	

Table 1-23 After-Tax Project Cash Flow (cont.)

	Production Period	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
	Financial Year	0	1	2	3	4	5	6	7	8	
	Unit										
Write-offs											
Total Write-Offs	**US$**									**$25,679,930**	**$25,679,930**
Total Taxes and Duties Payable											
ISR	US$	$0	$0	$0	$0	$2,660,484	$5,987,186	$4,514,939	$1,386,556	$0	$14,549,165
PTU	US$	$0	$0	$0	$0	$44,476	$44,476	$44,476	$44,476	$0	$177,903
Mining Duty EBITA (Lead and Zinc)	US$	$0	$344,345	$1,034,757	$1,473,183	$1,151,697	$1,361,528	$668,209	$161,991	$135,104	$6,330,814
Mining Duty EBITA (Silver)	US$	$0	$278,300	$917,561	$1,103,704	$1,060,768	$969,192	$991,516	$614,286	$296,509	$6,231,835
Total Taxes	**US$**	**$0**	**$622,645**	**$1,952,318**	**$2,576,886**	**$4,917,426**	**$8,362,382**	**$6,219,139**	**$2,207,309**	**$431,613**	**$27,289,717**
Total Taxes and Duties Paid											
ISR	US$		$0	$0	$0	$0	$44,476	$44,476	$44,476	$44,476	$177,903
PTU	US$		$0	$0	$0	$2,660,484	$5,987,186	$4,514,939	$1,386,556	$0	$14,549,165
Mining Duty EBITA (Lead and Zinc)	US$		$0	$344,345	$1,034,757	$1,473,183	$1,151,697	$1,361,528	$668,209	$297,095	$6,330,814
Mining Duty EBITA (Silver)	US$		$0	$278,300	$917,561	$1,103,704	$1,060,768	$969,192	$991,516	$910,795	$6,231,835
Total Taxes	**US$**	**$0**	**$0**	**$622,645**	**$1,952,318**	**$5,237,370**	**$8,244,127**	**$6,890,134**	**$3,090,757**	**$1,252,366**	**$27,289,717**
Net Earnings											
After Tax Earnings (Annual)	US$	$0	$1,709,406	$8,623,215	$15,989,790	$9,001,939	$7,112,237	$512,537	-$6,779,516	-$3,595,013	**$32,574,596**
After Tax Earnings (Cumulative)	US$	$0	$1,709,406	$10,332,621	$26,322,411	$35,324,351	$42,436,588	$42,949,125	$36,169,609	$32,574,596	
Net Cash Flow (After Tax)		-$91,170,948	$4,600,813	$21,560,140	$30,353,141	$22,420,787	$17,937,324	$13,580,331	$6,075,819	$7,217,191	$32,574,596
Discount Rate	10%										
Discount Factor		1.00	0.91	0.83	0.75	0.68	0.62	0.56	0.51	0.47	
Discounted Cash Flow	US$	-$91,170,948	$4,182,557	$17,818,297	$22,804,764	$15,313,699	$11,137,667	$7,665,743	$3,117,856	$3,366,873	**-$5,763,493**
Cumulative Discounted Cash Flow	US$	-$91,170,948	-$86,988,391	-$69,170,094	-$46,365,330	-$31,051,631	-$19,913,964	-$12,248,221	-$9,130,366	-$5,763,493	
Net Present Value	- 5,763,493										
Internal Rate of Return	8.11%										
Periods to Discounted Payback											

Table 1-24 After-Tax Sensitivity Analysis

Item	NPV (USD Million)	IRR (%)
Base Case	**-5.8**	**8.1%**
Capital Cost +15%	-18.5	4.57%
Capital Cost -15%	5.9	12.19%
Operating Cost +15%	-34.6	-3.35%
Operating Cost -15%	19.4	15.83%
Sale Price (Zinc) +15%	4.1	11.31%
Sale Price (Zinc) -15%	-16.0	4.59%
Sale Price (Lead) +15%	3.7	11.19%
Sale Price (Lead) -15%	-15.6	4.74%
Sale Price (Silver) +15%	9.1	12.80%
Sale Price (Silver) -15%	-21.7	2.32%
Mill Recovery (Zinc) +15%	4.1	11.31%
Mill Recovery (Zinc) -15%	-16.0	4.59%
Mill Recovery (Lead) +15%	3.7	11.19%
Mill Recovery (Lead) -15%	-15.6	4.74%
Mill Recovery (Silver) +15%	7.1	12.21%
Mill Recovery (Silver) -15%	-19.3	3.26%

The following table summarizes the sensitivity of the discount rate used on the before and after-tax NPV and IRR.

Table 1-25 Discount Rate Sensitivity Analysis

	Pre-Tax		After-Tax	
Discount Rate	NPV	IRR	NPV	IRR
0%	59,864,314	13.24%	32,574,596	8.11%
8%	18,724,880	13.24%	358,817	8.11%
9%	14,747,296	13.24%	- 2,780,353	8.11%
10%	10,973,630	13.24%	- 5,763,493	8.11%
11%	7,390,818	13.24%	- 8,600,429	8.11%
12%	3,986,785	13.24%	- 11,300,255	8.11%

Spider charts are shown in Figure 1-8 and Figure 1-9 below for the Project's after-tax sensitivity to metal prices, capital cost, operating cost, and mill recovery, with key assumptions varying plus and minus 15 percent.

Figure 1-8 After-Tax Project NPV

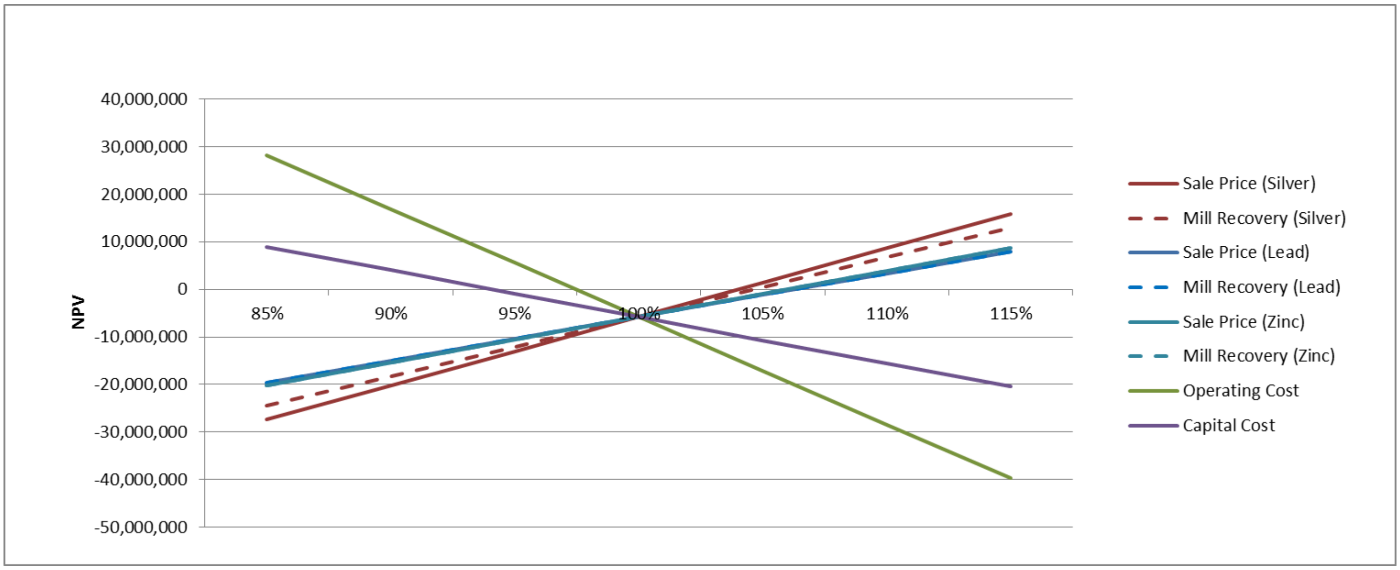
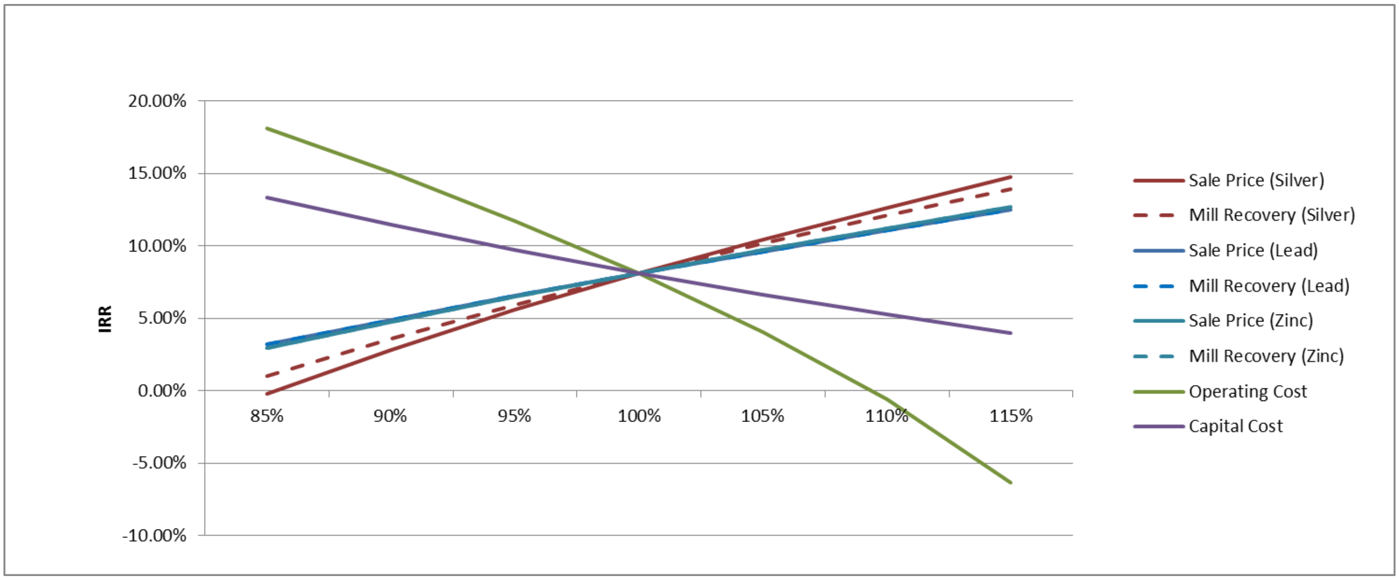


Figure 1-9 After-Tax Project IRR



1.13 Recommendations

The Bilbao deposit contains a reasonable quantity of mineral resources between the oxide, transition, and sulphide mineral zones; however, the lack of metallurgical test data available for the transition zone and identified recovery challenges for the oxide zone currently limit the scope of this PEA to the total mineable sulphide resources to offset the capital costs associated with the project. Recommendations have been made throughout this section identifying various opportunities to increase the mineable resource and reduce operating costs through additional exploration and engineering, improving the overall economics of the project.

1.13.1 Drilling

- Additional definition drilling targeted at the Bilbao transition and sulphide zones could lead to re-classification of inferred resources to indicated resources, potentially contributing to the total mineable resource studied at the pre-feasibility level;
- Exploration drilling at the Bilbao 2 area, approximately 1.5 km south of Bilbao, has potential to offer additional mineral resources to the project due to the fact that current trenching, sampling and resulting soil geochemistry information identifies similarities between the two areas. An additional source of feed to the plant designed in this study could either lengthen the overall life of the mine, increase the daily production rate, or result in a combination of the two, improving the NPV and IRR of the project;
- RPM recommends reporting the detection limit by campaign/laboratory/method and assigning half detection limit values to assays under the detection limit and negative codes to non-sampling and non-recovery intervals. Core loggers considered intervals as “not sampled” in the case they were barren;
- Duplicates show consistently good repeatability in the 2011 scatterplots. RPM recommends indicating the nature of duplicates, coarse or fine, and incorporating the relative error – data percent graphs. The maximum error, currently accepted by industry, is 10% and 20% for 90% data for fine and coarse duplicates, respectively.

1.13.2 Sample Preparation, Analyses and Security

- Results out of the 2 STD lower and upper limits are greater than the industry accepted results of ±5%. Zn has 12%, Pb 7%, Ag 65%, and Cu 30% of the results outside the limits. RPM strongly recommends researching the source of these poor reference sample results. If these out-of-limit results are confined to certain assay batches, RPM recommends re-assaying those batches along with the appropriate QA/QC samples. If the out-of-limit results are random with all batches, RPM recommends sending out at least 10% of the pulps along with the appropriate QA/QC samples to a second lab for a check. If the biases of the assays of the standard samples are representative of the laboratory accuracy and the results from the core samples are similarly biased, the estimation of grade from these samples would be conservative.

1.13.3 Data Verification

- RPM spot checked three lab certificates, ICP certificate 2010 – 4529, 4523 and 4523-2, of the drill hole X-71. RPM detected differences at the third decimal in Zn-Pb; this is irrelevant for resource estimation, however, RPM recommends completely matching lab certificate and database. Zn, Pb, and Ag grade database mistakes were not found by RPM. (Due to the fact that database verification was not part of the original scope, RPM simply spot checked some information.) However, RPM considers it is essential to complete data verification of at least 10% of holes prior to a feasibility study (FS). This data verification should include:
    i) Field check of drill hole location;
    ii) Logging review; and
    iii) Coordinate-logs-assay certificates – database comparison.

1.13.4 Mineral Resource Estimate

- RPM decided on defining grid spacing based on geological and grade continuity which shows a reasonable level of confidence to define measured, indicated and inferred resources. RPM recommends incorporating, in a feasibility study, the estimation errors associated with annual and quarterly production panels to define indicated and measured resources, respectively;

1.13.5 Mining

- Level spacing resulting from the proposed stope design (without the use of cable bolt support for backs and walls) is 24 metres. The potential to increase level spacing and correspondingly reduce level development, through use of cable bolts, may lead to lower development costs and should be further assessed;
- The backfilling approach used in this study includes the use of cemented and uncemented rock fill. Further analysis of hydraulic and sand backfilling options, in terms of preparation and distribution, may further reduce overall operating costs;
- There may also be opportunity to reduce operating costs significantly (~$5/t to $6/t) by reducing the number of stopes filled with backfill all together. Further geotechnical study would need to be carried out for this scenario to better understand possible ore losses with pillars left in place, and possible recovery of these pillars through caving activity. Potential also exists for deferral of ramp and associated development;
- Inclusion of transition zone material in the mine plan should be investigated (requiring additional metallurgical testwork) to extend the life of mine and/or potentially increase the mining rate per year;
- Some degree of stope sequencing was achieved in the mine plan to improve mined grades in the opening years of the operation, but further optimization of stope sequencing leading to improved cash flow may be achievable and should be studied.

1.13.6 Metallurgy

Recommendations for future work on samples from the Bilbao deposit include:

- Further scoping level testwork, mineralogy, and flowsheet development on composite and variability samples from the oxide zone to identify the potential for additional economic recovery of metal values. Bilbao contains a substantial in-situ oxide resource of 3.8 million tonnes (3 million inferred and 791,000 indicated) at a Zn equivalent grade of 6.5%, and opportunities may include new technologies for leaching and gravity recovery, or high-grading of the oxide zone to focus specifically on the zinc and/or silver minerals;
- Additional variability testing of samples from the transition zone to better characterize the extent of float recoverable mineralization in this area;
- Mineralogical characterization of transition zone samples from different drill holes to develop correlations between lead and zinc deportment and core log data;
- Mineralogical characterization of sulphide variability composite LS-3 to compare with the results of the transition zone samples and determine if this sample represents another area of altered material.

1.13.7 Environmental Studies

1.13.7.1 Hydrology

The 2011 Phase 2 Hydrologic Assessment provides the following recommendations:

- A program of baseline groundwater quality and water levels should be established, to allow environmental monitoring over time once the mine is in operation.
- A hydrogeologic drilling investigation should be completed at candidate well locations near the mine site, as data obtained from this investigation would be necessary to allow any water rights transfers. Water

rights in Mexico are administered by the *Comision Nacional de Agua* ("National Water Commission" and "CONAGUA").
- Six target zones are identified near the Project area for exploratory drilling (at Las Borregas, Bilbao, and La Ardilla).
- Potential groundwater inflow towards the mine should be investigated to incorporate any necessary dewatering costs in feasibility programs.

1.13.7.2 Geochemistry

- The results provided in the Geochemical Results of Waste Rock and Tailings Samples Report are preliminary, and additional sampling is required to assess short-term and potential long-term metal leaching characteristics of the waste rock. Further sampling may be required to ensure conformance with best practice sampling guidance. The Geochemical Results of Waste Rock and Tailings Samples Report recommends that elemental analysis and a review of total waste rock tonnages and rock types be performed to verify that the current number of samples is consistent with accepted characterization guidelines. These efforts will be undertaken during the feasibility study stage of Project development;
- With respect to the leach test results regarding tailings and tailings supernatant quality – the Project plans to construct TDF with an HDPE liner to prevent infiltration into groundwater. There is the potential for periodic releases of water from the TDF to the environment. In this event a water treatment facility may be required, and this potential should be evaluated during the feasibility stage of Project development. Additional static and possible kinetic testing should also be performed to allow for a more refined understanding of tailings geochemistry.

1.13.7.3 Waste and Tailings Disposal, Site Monitoring and Water Management

Additional studies are recommended in the Prefeasibility TDF Study Update. These studies will be completed during the feasibility stage of Project development and will include the following:

- Geotechnical drilling for a better understanding of geologic, geotechnical and hydrogeological conditions of the TDF area;
- Detailing of quantities of material available from potential borrow sources for TDF construction;
- Seismic hazard analysis to verify peak ground accelerations used in the TDF design;
- Confirmation of assumptions used to develop the water balance; and
- Additional geochemical testing of tailings material to determine the potential need for treatment of water which may be discharged during wet climatic conditions.

2. Introduction

2.1 Statement for Whom the Report is Prepared

This Preliminary Economic Assessment is prepared for such persons as may be interested in investing in the Bilbao property. Specifically, its intended audiences are those responsible for securing finance to develop future mining projects; such persons may be from the financial fraternity, large mining houses or similar institutions. It will also serve as a definitive record of the state of affairs at the Bilbao project as of April 28, 2014 which can be used to plan mining activities in the future.

2.2 Purpose of Report

It is expected that it will serve as a complete record of work carried out at the Bilbao project so that reviewers are able to pronounce on its worth and decide whether to invest in the property. The report is prepared following the guidelines of the Toronto Stock Exchange (TSE) rules for preparing NI 43-101 reports; indeed the preferred headings are taken verbatim from these rules so as to ensure that the report is comprehensive and accords precisely with its requirements. The objective of the report is to bring together all the relevant data which has been amassed during exploration and evaluation phases of work undertaken over the past eight years so that the whole will serve as an historical record, as well as provide an economic analysis of the potential viability of Bilbao mineral resources.

2.3 Information/Data Sources

Whilst there are several historical reports on the mineralization occurring at the Bilbao project these largely relate to the pre-war open-pit mining of the oxide resource and to metallurgical trials aimed at winning metals from that oxide resource. A list of the pertinent historical reports and published papers is seen in Section 27 of this report. These reports are now rather dated and have mostly been superseded by in- house investigations undertaken by Xtierra between 2006 and 2014, with the latest focus revolving around the deeper sulphide resource. Some 140 internal reports on various aspects pertaining to the Bilbao district have been prepared during that eight year period and serve as a strong database on which this report has been based. A list of the relevant in-house reports on Bilbao is given in the document “Internal Company Reports Concerning the Bilbao Property”.

2.4 Personal Inspection of Prospect

RPM have adjudicated on all aspects of work related to the preparation of the final Preliminary Economic Assessment. The principal authors of this report are independent experts who have examined the property as part of their input to earlier studies as well as during preparation of NI 43-101 reports. The names and contributions these persons have made are detailed in Section 2.5 of this report.

Qualified Persons of this report that have visited the project site include:

- Kevin Tanas, P.Eng., Principal Mining Consultant, RungePincockMinarco (Canada) Limited
- Esteban Acuña, Senior Geologist, RungePincockMinarco
- Rick Parker, Consulting Geologist
- Malcolm Buck, P.Eng., Mining Consultant, AMBUCK Investment Corporation
- Dana Strength, President, Strength GEC

2.5 Terms and Units

The following terms and definitions are used in this report.

- Xtierra refers to Xtierra Inc.
- RPM refers to RungePincockMinarco and its representatives.
- Bilbao Project references to the Bilbao Silver-Lead-Zinc Project located in the District of Panfilo Natera, Zacatecas State, Mexico, including the proposed mine area, process plant location, and other related facilities.
- SEMARNET refers to the Secretary of Environment and Natural Resources (SEMARNET)

RPM has based all measurements in the metric system, and has identified exceptions to this, notably when listing both English and Metric standards. Currencies are based on March 2014 US Dollar. Unless otherwise stated, Dollars are United States Dollars, Grades are described in terms of percent (%) or grams per metric tonne (g/tonne), with tonnages stated in metric tonnes of 1,000 kilograms (2,204.62 pounds).

The following abbreviations are used in this report:

Abbreviation	Unit or Term	Abbreviation	Unit or Term
Ag	Silver	mm	Millimeters
As	Arsenic	M	Million
Au	Gold	Mt	Million Tonnes
Bi	Bismuth	mtpd	Metric Tonnes per Day
Cd	Cadmium	Mtpy	Million Tonnes per Year
Cu	Copper	NPV	Net Present Value
Fe	Iron	Oz (oz/t)	Ounces (ounce/tonne)
g/tonne (g/t)	Grams per Tonne	Pb	Lead
ha	Hectare (10,000m^2)	%	Percent by Weight
Hg	Mercury	Sb	Antimony
kcal	Kilocalories	SiO	Silica
kg	Kilograms	T or t	Metric Tonne (2,204 lbs)
km	Kilometer(s)	tpa	Tonnes per Annum
k	Thousands	tpy	Tonnes per Year
LOM	Life of Mine	tpd	Tonnes per Day
Mn	Manganese	ug	Underground
m	Meters	Zn	Zinc
masl	Meters Above Sea Level	$	United States Dollars

3. Reliance on Other Experts

The authors of this Report state that they are the Qualified Persons as identified in Table 3-1.

Table 3-1 Qualified Persons

Qualified Person	Title	Company
Kevin Tanas, P.Eng.	Principal Mining Consultant	RungePincockMinarco (Canada) Limited
Esteban Acuña	Senior Geologist	RungePincockMinarco
Rick Parker	Consulting Geologist	Independent
Malcolm Buck, P.Eng.	Mining Consultant	AMBUCK Investment Corporation
Lyn Jones, P.Eng.	Principal Metallurgist	ConsuMet
Clinton Swemmer PMP PrEng (South Africa) MSAICE	Vice President: Projects	DRA Americas Inc.
Jane Spooner, MSc, P.Geo.	Vice President	Micon International Limited
Dana Strength	President	Strength GEC

During the preparation of this Report, RPM has relied on the contributions of a variety of specialist consultants who have provided reports and studies for the Technical Report.

The financial analysis that RPM have prepared are based on owner's cost estimates provided by Xtierra, which have not been audited by RPM.

Permitting status and the present status of mining rights and areas cited in this document have been provided by Xtierra. RPM does not have the expertise to properly assess the permitting status and mining rights and accepts the information provided by Xtierra.

The following persons undertook various investigations and furnished reports on geotechnical, metallurgical, hydrological and environmental matters integral to the overall study. All are fully qualified in their particular fields.

- Dr Shaoxian Song, Ing. de Minerales, Master of Mineral Processing, PhD Mineral Processing, Ing de Quimica— Mineral Processing Engineer
- Dr Alejandro López Valdivieso, Ing. Metalúrgica, MSc. South Dakota School of Mines, PhD, Univ California, Berkeley.—Metallurgical Processing Engineer.
- Eduardo Escárcega Rangel
- Carlos Garcia Herrera
- Shiu Kam - Golder Associates Ltd.
- Isaac Ahmed (BA.Sc, MA.Sc, P.Eng) - Golder Associates Ltd.
- Joe Carvalho (B.A.Sc.(Honours), M.A.Sc., Ph.D., P.Eng.) – Golder Associates Ltd.
- Berenice Rodriguez Ortega—Schlumberger Water Services

3.1.1 Extent of Reliance

The overall evaluation/assessment of the data on the property was undertaken by personnel of Xtierra, and verified by qualified independent persons expert in their particular field.

Analytical services were variously provided by several external commercial, internationally acceptable, analytical laboratories including SGS, Stewart Group, Acme and Inspectorate. Reliance on analytical veracity rests with these various laboratories.

Almost all of the metallurgical testwork was undertaken by external consultants and directed by the metallurgical department of the University of San Luis Potosí (UNSLP) under the supervision of Godfrey McDonald and others. Metallurgical tests utilizing a pilot plant for the oxide ores was undertaken in-house by Xtierra under the direction of Dr Song & Dr Lopez of the UNSLP and overseen by DRA.

An Environmental Impact study was prepared by Soluciónes de Ingeniéria y Calidad Ambiental S.A. de C.V, (SIICA) of Aguascalientes, under the direction of Eduardo Escárcega Rangel, which utilized data collected both in-house and by themselves.

Geotechnical data was originally undertaken in two campaigns by Tratamientos Geotecnicos, S.A. de C.V & Technología y Sistemas, S.A. de CV. The first part of the work involved a structural analysis of the diamond drill core and mapping fracture systems in the old open-pits and second phase was undertaken by drilling angled holes within the perimeter for the initially proposed open-pit.

Golder Associates Ltd. was later retained by Xtierra to prepare and run a numerical model to evaluate the proposed stope sizes and mining sequence.

Dowding & Reynard Associates, DRA, supervised the pilot-plant trials to ascertain processing characteristics of the oxide mineralization.

Hugo Renteria Felix of Servicios Electricos e Industriales, S.A. de C.V. furnished a report entitled “Reporte General Estudio de Factibilidad CFE Proyecto Bilbao Mining” to the Company in September 2009 concerning provision of power to the proposed mine site with quotes from the Comisión Federal de Electricidad (CFE).

Behre Dolbear de México, S.A.de C.V. undertook an update of the earlier Kilborn report in late 2006.

Terra Tecnología del Subsuelo undertook ground geophysical studies including magnetometry and Induced Polarization studies.

Golder Associates undertook a study concerning Tailings disposal at the Bilbao property.

Schlumberger Water Services undertook a hydrology study on the property.

3.1.2 References

The following is a list of those companies and departments who have contributed to the exploration activities at the Bilbao prospect and whose work has been material in enabling completion of this PEA report. It includes the addresses and contact numbers of commercial firms that have either undertaken the drilling, analysis or other relevant exploration activities.

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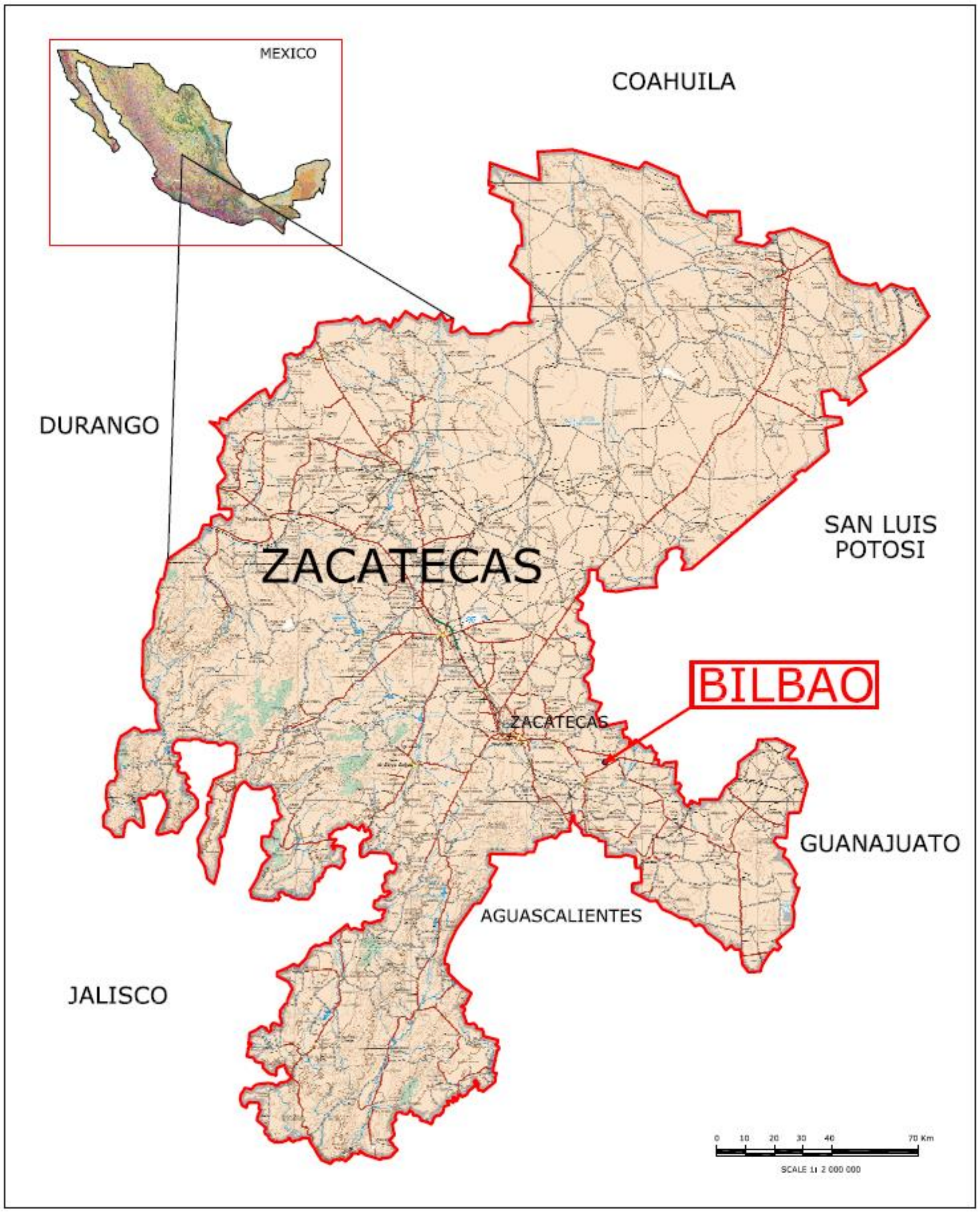
4. Property Description and Location

4.1 Property Location

The Bilbao project is situated in central México 44 km ESE of Zacatecas, the capital of Zacatecas State, México. It is located some 475 km north-west of México City. The main shaft at the Bilbao prospect is at Lat 22°39´47´´N & Long 102°08´46"W, [2508.905/793.281] exactly 3.9 km from the centre of the nearby town of Pánfilo Natera, at an altitude of about 2160 meters. Figure 4-1 shows the location of the project within the State of Zacatecas, México.

Figure 4-1 Bilbao Project Location



4.2 Property Status

The area of interest at Bilbao is covered by 9 claims totalling 1,406.69 hectares. The rights are vested in two companies Bilbao Resources, S.A. de C.V. and Bilbao Mining, S.A. de C.V. the relevant data is given below.

Table 4-1 Mineral Claim Blocks in the immediate vicinity of the Bilbao Project

Claim Name	Claim #	Company	Surface Area Ha	Start Date	End Date
Bilbao	222854	Bilbao Resources SA de CV	27.344	9-Sep-04	8-Sep-54
Bilbao	214309	Bilbao Resources SA de CV	422.766	6-Sep-01	5-Sep-51
Bilbao II	222638	Bilbao Resources SA de CV	870.032	3-Aug-04	2-Aug-54
El Trinque	211940	Bilbao Resources SA de CV	8.543	28-Jul-00	27-Jul-50
El Porvenir	177340	Bilbao Mining SA de CV	25.000	18-Mar-86	17-Mar-36
La Güera	198980	Bilbao Mining SA de CV	9.000	11-Feb-94	10-Feb-44
Mina Los Compadres	198978	Bilbao Mining SA de CV	25.000	11-Feb-94	10-Feb-44
El Milagro	223126	Bilbao Mining SA de CV	9.000	19-Oct-04	18-Oct-54
Leonor	210484	Bilbao Mining SA de CV	10.000	8-Oct-99	7-Oct-49

Additionally, five small claims (La Blanca, Ampliación, La Blanca, La Africana, Ampliación, El Cabezon, and La Fe) are located inside the Property, but beyond the limits of the proposed operation, and are not material to the project.

Figure 4-2 shows the location of the Bilbao claims in relation to the immediate environs and the locations of nearby working mineral deposits. The boundaries of the constituent claim blocks given in Table 4-1 are shown in Figure 4-3; it also shows the position of the mineralized body with respect to these claim boundaries.

In addition to the above claims, which cover the immediate area of interest at the Bilbao project, the Company has several other claims in close proximity to Pánfilo Natera which cover zones with similar geological setting and good exploration potential; these include Gaby Marina, Cata Negra, Piero y Gia and Orca 1. These claims could host similar skarn- hosted mineralization to that at Bilbao and indeed recent drilling on Gaby Marina supports this expectation. Results of the exploration carried out in these surrounding areas are detailed in Section 0.

4.3 Titles to the Claims

Copies of the Title deeds relating to the above claims can be seen in the report entitled “Title Deeds of Constituent Claims at the Bilbao Project”**Error! Reference source not found.**. All the claim titles covering the Bilbao Project are valid and in good stead with all relevant fees fully paid.

4.4 Surface Rights

Surface rights in the area are owned by private landowners and by ejidos (a federally supported system of communal land tenure). Figure 4-4 shows the surface rights owners on the Bilbao claims with respect to the disposition of the mineralized body and projected open pit limits. To date exploration has been facilitated through an agreement with the two principal persons involved namely Snrs Marcos Alvarez Delgado and Ismael Alvarez Delgado. The area required to be purchased is between 320 and 332 hectares which will be sufficient to accommodate the mine as well as the required infrastructure such as processing plant, tailings disposal area, ancillary buildings, offices etc.

Figure 4-2 Location of the Bilbao Claim Blocks

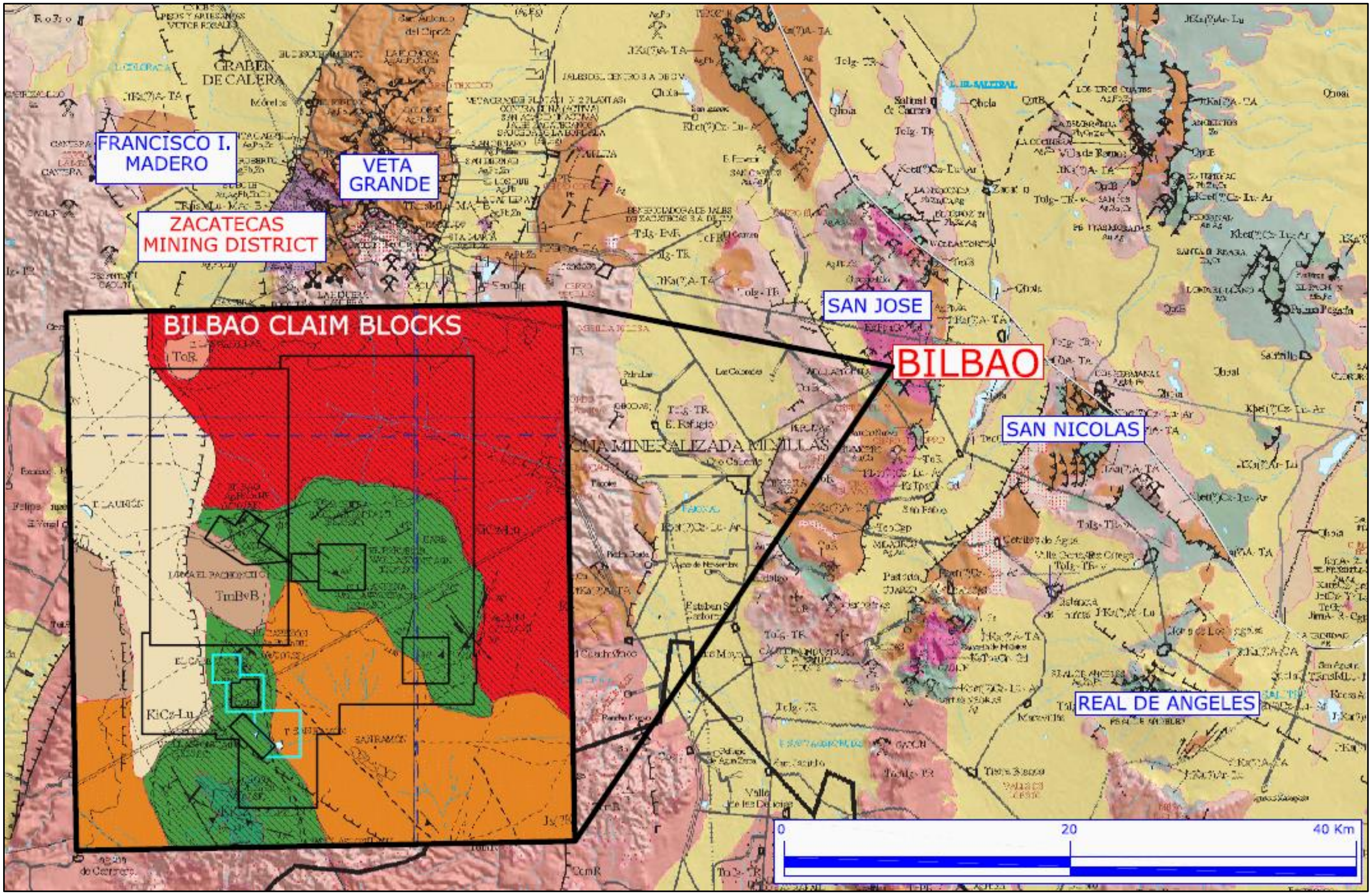
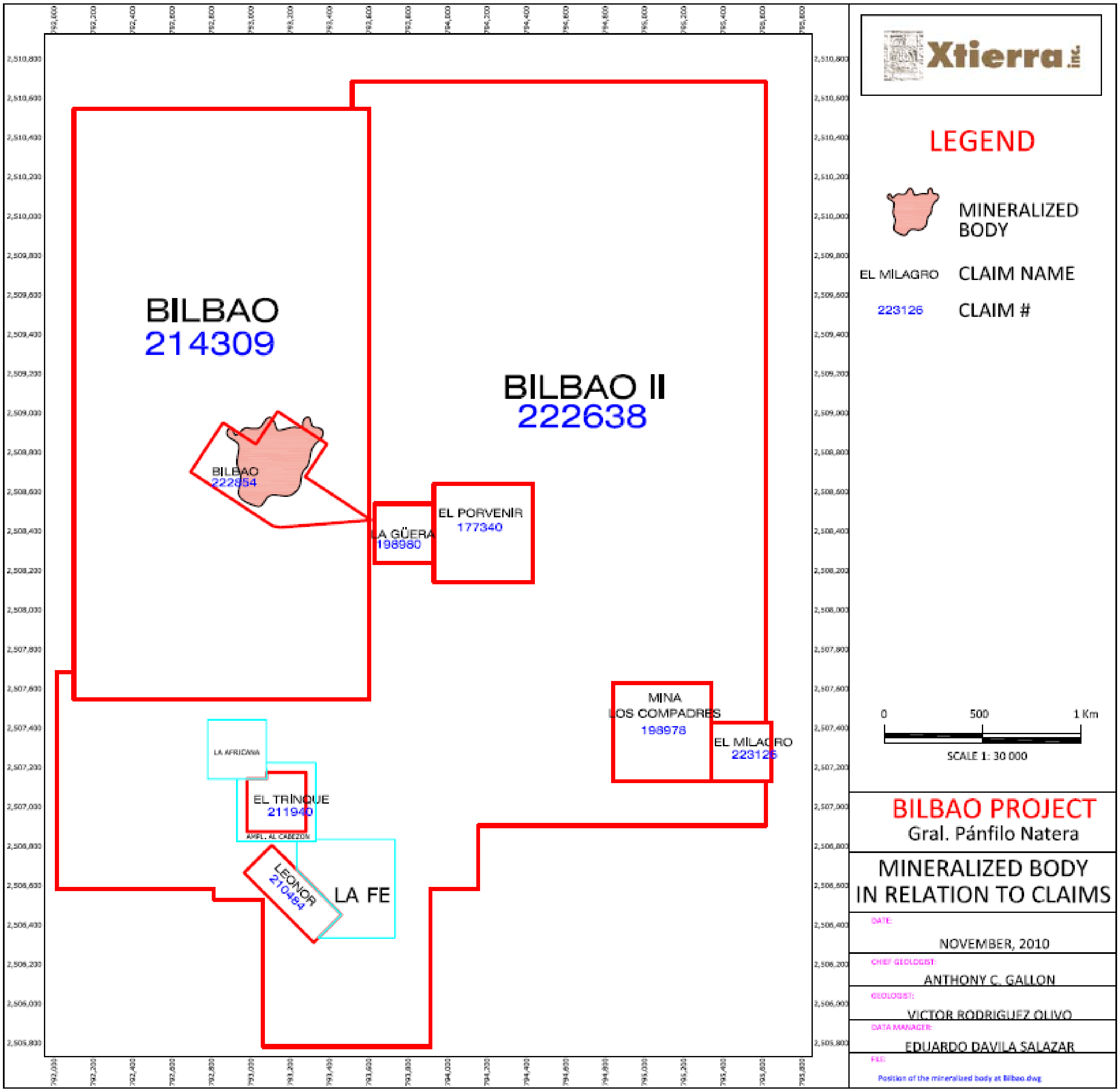


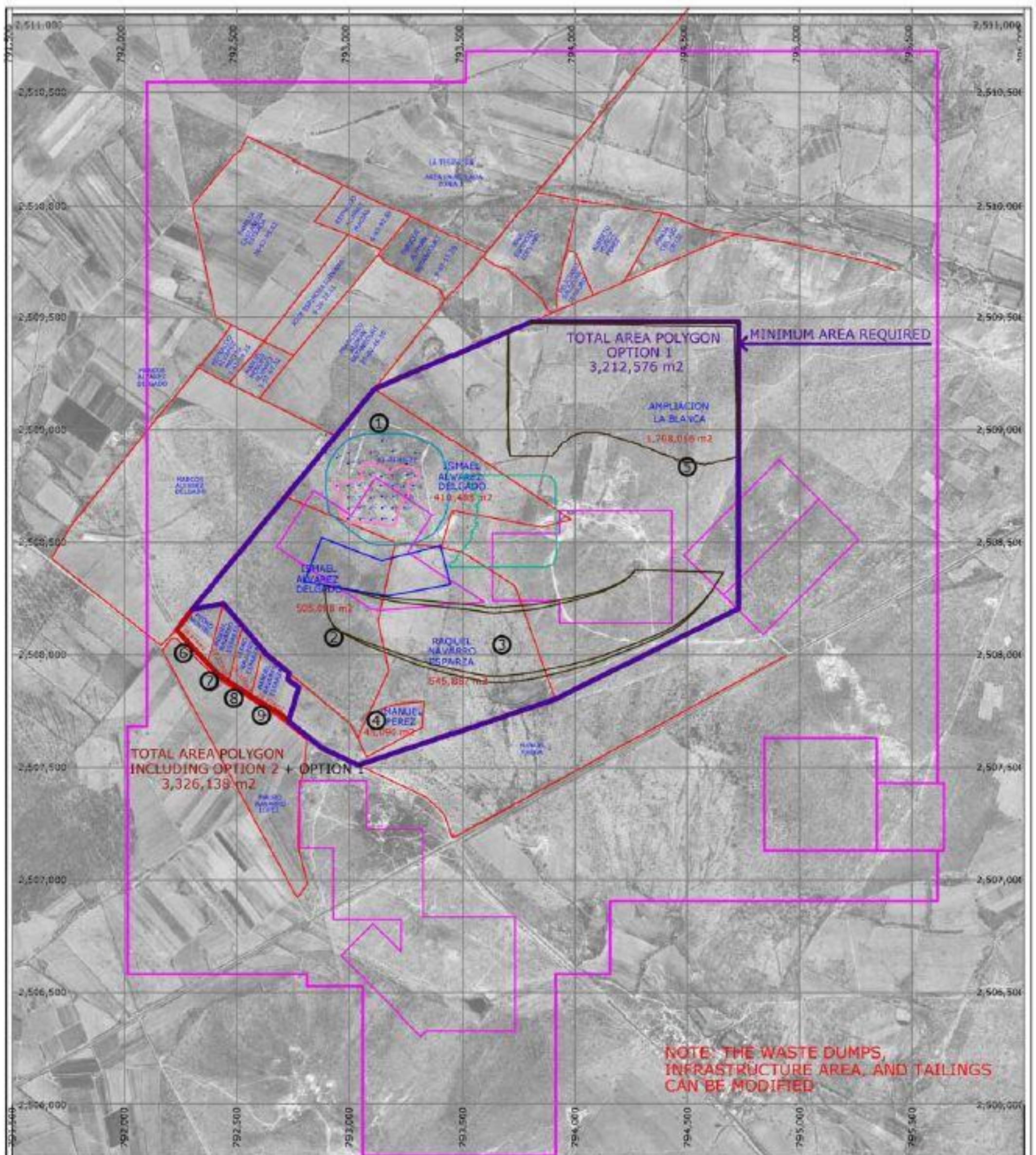
Figure 4-3 Position of the Mineralized Body in Relation to the Claim Blocks



4.5 Royalties payable/encumbrances

A Royalty of 1.5% is payable to Minera Portree, S.A de C.V. on NSR. The agreement which relates to this is found in the document entitled "Royalty Agreement Pertaining to the Bilbao Property".

Figure 4-4 Bilbao Surface Rights Ownership



OPTION 1			
OWNER	No.	M2	HECTARES
ISMAEL ALVAREZ DELGADO	1	410,485 m2	41.05 H
ISMAEL ALVAREZ DELGADO	2	505,098 m2	50.50 H
RAQUEL NAVARRO ESPARZA	3	545,887 m2	54.58 H
MANUEL PEREZ	4	43,090 m2	4.30 H
AMPLIACIÓN LA BLANCA	5	1,708,016 m2	170.80 H
TOTAL OPTION 1		3,212,576 m2	321.25 H
OPTION 2			
OWNER	No.	M2	HECTARES
PEDRO MONTEL	6	28,974 m2	2.89 H
MIGUEL NAVARRO ESPARZA	7	29,259 m2	2.92 H
PEDRO NAVARRO ESPARZA	8	26,355 m2	2.63 H
MANUEL NAVARRO ESPARZA	9	28,974 m2	2.89 H
TOTAL OPTION 2		113,562 m2	11.35 H
TOTAL INCLUDING OPTION 2 + OPTION 1		3,326,138 m2	332.61 H

LINES
CLAIM
POLYGON OPTION 1
POLYGON OPTION 2
OWNERS
COORDINATE
OPEN PIT
WASTE DUMP
TAILINGS
INFRASTRUCTURE

Xtierra Inc.

4.6 Environmental Liabilities

There are no legacy environmental liabilities for Bilbao. Development of the underground mine will occur around an existing open pit (or “glory hole”) that was operated in the 1910’s. Subsequent to this activity oxides of copper, lead, zinc and silver were sporadically mined at the site until approximately 1953. Wollastonite was also mined on a very small scale until mid-2006. A small limestone quarry is located on the Bilbao claim southwest of Panfilo Natera.

These activities have created surface disturbances, including the presence of an historic shaft used to access crude underground workings at two levels. Other surface disturbance includes the presence of small open pits used to access mineralized resources located near the ground surface. Although these features do currently represent a health and safety hazard, there is no evidence of environmental degradation nor is there any regulatory environmental liability associated with them.

A report entitled “*Aviso de Apego a la NOM-120-SEMARNET-1997, Para Actividades de Explotacion Minera del Proyecto Bilbao*” was filed with SEMARNET in 2006, detailing efforts that would be undertaken by the Project to avoid any sensitive cacti (Figure 4-5) during exploration drilling, and to reclaim drilling pad locations. These mitigations have been implemented and appropriate rehabilitation is undertaken at the completion of all exploration drilling activity.

Figure 4-5 Cacti Species on Bilbao

4.7 Permits that must be acquired to conduct the work proposed

A summary of significant permits that will be required to implement the Project is provided in Table 20-2, and discussed in Section20.3. Prior to construction all mining projects must first prepare an Environmental Impact Study (“*Manifestacion de Impacto Ambiental” and “MIA”)* and an Environmental Risk Study (“*Estudio de Riesgo Ambiental”* and “ERA”). These completed studies are jointly submitted to SEMARNET, which then reviews the document and either rejects or accepts the MIA with corresponding conditions of approval in a Resolution Letter (the “Resolucion”). In addition to the MIA Resolution Letter a project must also obtain a Change in Land Use (“*Cambio de Uso de Suelos”* or “CUS”) permit which is granted after submission and approval of a technical study justifying the change in land use of the project area from its current use (agricultural) to development of a mine (industrial).

On March 27, 2013 Bilbao Resources, S.A. de C.V. contracted with the Mexican environmental consultancy SIICA to complete the MIA and ERA, and to assist in overall permitting for the Project. At the time of writing the MIA and ERA documents were under development, incorporating information from existing environmental studies as well as studies that are in progress. The completion and approval of these studies is required prior to the initiation of construction. In addition to the aforementioned permits/approvals the following permits/approvals will be required:

- Unique Environmental License (from SEMARNET state office);
- Archaeological Release Letter (from National Institute of Anthropology and History)
- Use of Explosives Permit (from Secretary of National Defense);
- License of Construction (from Municipality of General Panfilo Natera);
- License of Land Use (from Municipality of General Panfilo Natera); and
- Hazardous Waste Permit (from Secretary of National Defense).

Thus far the Project has received regulatory approval for exploratory drilling. However SIICA has extensive experience in the construction and operational permitting of industrial projects in the region, and permitting of the Project will progress during the feasibility stage of development.

4.8 Other known Risks to Title or Ability to do work

- Any future mining is dependent upon securing the surface rights over the Bilbao claims which are separate from the mineral rights. This can be achieved either by leasing or purchasing the surface rights.
- Since the mineralization contains some potentially deleterious elements such as Hg, As and Cd care must be exercised to ensure that these are adequately dealt with.
- Loss of Pb adsorbed on hematite as magnetoplumbite may have implications with regard to the pre-concentration of the iron oxide fraction by electromagnetic separation which could take with it a portion of the lead component in particular; the reject fraction from this process should therefore be monitored for its Pb content in particular.
- Any open-pit mining operation will generate dust which could create an airborne health hazard. Such potential problems can be overcome by conventional water damping-down measures on the access and mine haulage roads.
- Care must be taken to ensure that all facilities are secure enough to withstand any “500 year storm event”. This is particularly relevant to any potential overspill of the tailings pond. This will be prevented by a double defensive berm (baffle embankments) around this facility.

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Introduction

The state of Zacatecas has a well-developed highway system including several Federal highways and well-maintained primary and secondary roads. A branch of the Mexican National railroad system crosses the central part of the State through the city of Zacatecas connecting Mexico City with Ciudad Juárez. Zacatecas is a large modern city with excellent facilities for business and a pool of experienced mining labour. The Zacatecas International airport is located 28km northwest of the capital with daily connections to Mexico City, Tijuana, Los Angeles, and less frequent services to other destinations in the United States.

The Pánfilo Natera district is located in a developed area of Zacatecas with good infrastructure and services. There are no obvious impediments to mine development in the district. Mining and agriculture have co-existed since early colonial times.

5.2 Accessibility

Figure 5-1 shows the integrated road network within the Pánfilo Natera district, specifically in relation to the Bilbao property and Zacatecas.

Access to the Bilbao prospect and the Pánfilo Natera exploration area is excellent as a divided highway (Mexico National Route 49) linking Zacatecas with San Luis Potosí passes across the northern limits of the property within 2.5 kilometres of the deposit. A paved road linking Pánfilo Natera with Ojo Caliente, (Zac144), passes the entrance turn-off to the project area, which is only 2.1km by all-weather dirt road from the main road (see Figure 5-1).

Virtually all the villages in the Pánfilo Natera district are interlinked by paved roads and there is access to most other areas via good farm roads. Most towns have garages capable of vehicle repairs. Bilbao is located about 27km away from the nearest rail head at Barriozabal near Ojo Caliente. There are two additional rail lines available at Salinas linking to San Luis Potosí at a distance of 43km.

Reaching the project is absolutely straightforward. There is a well-developed tarred road network throughout the central part of Zacatecas state which allows access to within 2km of the central part of the property. A flat all-weathered dirt road brings you to the main part of the prospect. (Figure 5-2)

As concerns external access from further afield, the main highway/motorway/autoroute, Mexico 49, runs close to the property, exactly 2.8km to the north. (Figure 5-3)

Visiting the property from abroad is straightforward since there are airports with direct international connections from North America at Zacatecas, and Aguascalientes each about an hour or so from the site. There are also international flights in and out of San Luis Potosí which is less than two hours away from site.

Bilbao is located 27km away from the nearest railhead at Barriozabal near Ojo Caliente and there are two additional railheads, each with loading facilities, at Salinas 43km away via the México 49 motorway. These rail systems are directly connected with the USA and, via San Luis Potosí, to the eastern seaboard ports.

Figure 5-1 Regional Transport Network in the Bilbao Vicinity

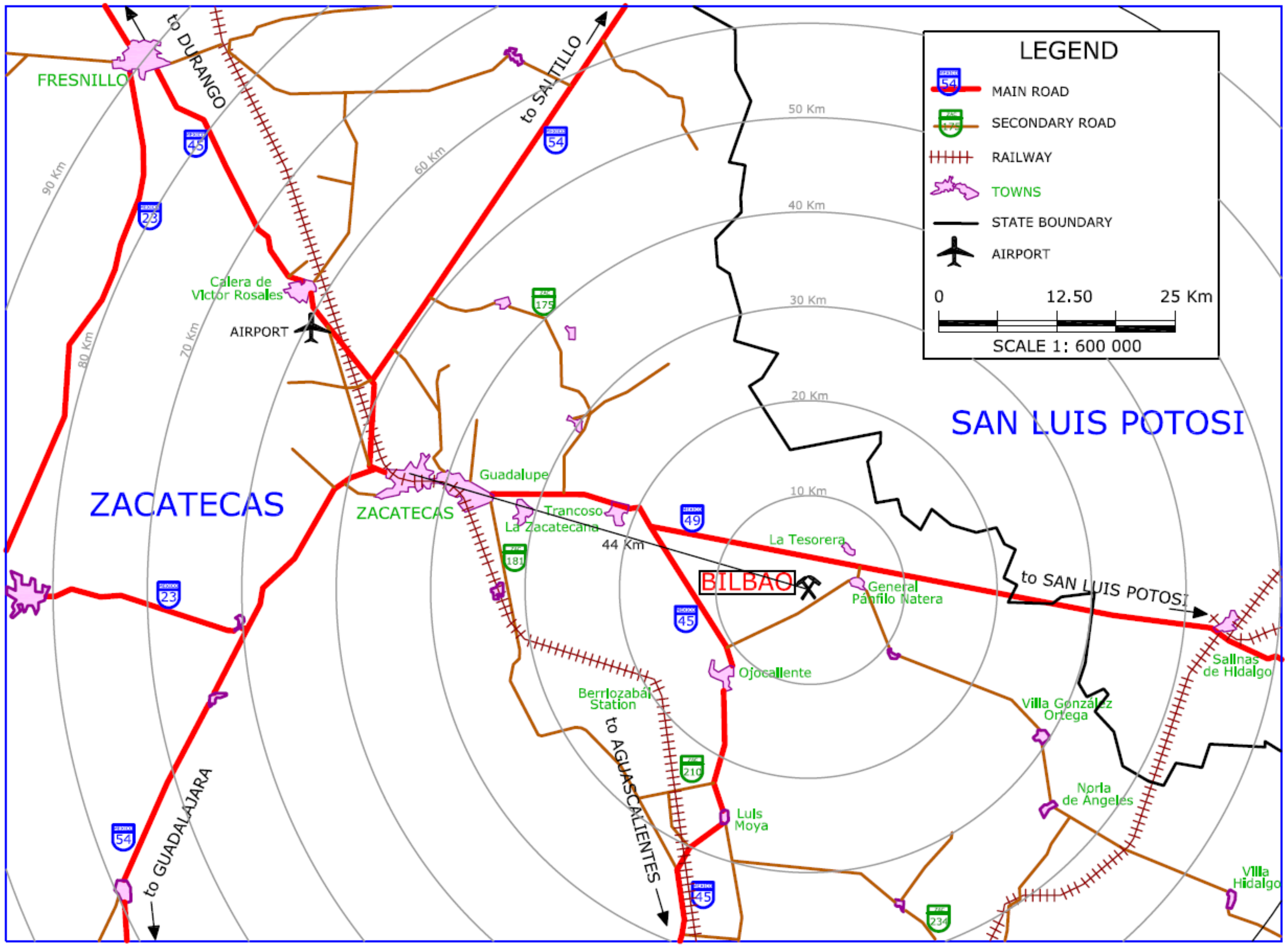


Figure 5-2 Dirt Farm Road to Project

Figure 5-3 Motorway Mexico 49 near to the Bilbao Project

5.3 Climate

The climatic regime at the Bilbao prospect can be described as semi-arid steppe with temperatures ameliorated by high altitude. The meteorological class type is BSkw(e) which equates to average annual temperatures below 180°C (64.4°F). Such relatively low temperatures for an area within the tropics are caused by the high elevation of the Mexican Meseta as a whole and for this reason some authorities prefer to regard this as a specialized Highland climatic zone.

Prevailing winds are light and from the south-east.

Analysis of the rainfall regime of the 10 meteorological stations closest to Bilbao by Schlumberger Water Services indicates an average annual precipitation at Bilbao of 395.6mm (<16") with most rainfall occurring between June and September as can be seen in Table 5-1 below:

Table 5-1 Average precipitation in mm for the Bilbao District

Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
13.3	9.6	6.0	6.8	23.0	69.3	89.4	79.4	72.9	28.5	8.0	8.6

As part of the requirements for the Environmental Impact Study for the Bilbao project a weather station was established in the village of Union de San Antonio which is situated at Lat 22^{o}39´46´´S & Long 102^{o}09´54"W at an altitude of 2144m amsl, precisely 1.73km due west of the Bilbao shaft. Readings were taken twice a day at 07h00 and 15h00 which more or less corresponded to the coolest and hottest parts of the day.

Parameters noted at these times were:

1. Temperature.
2. Relative Humidity in %
3. Cloud Cover in %
4. Occurrence of Rain
5. Air Pressure
6. Wind Direction
7. Wind Speed

In addition to these parameters a note was made on any significant weather phenomena that may have occurred. Table 5-2 below summarizes the monthly weather statistics.

Figure 5-4 shows the annual maximum and minimum diurnal temperature variations at Bilbao and Figure 5-5 the wind rose and direction statistics.

A comprehensive report on the weather has been prepared from the data collected at the weather station [Report # ACG/0308/XTR/249 entitled "*A Report on Weather Patterns at the Bilbao Project*"] which provides details on temperature, wind speed and direction, cloud cover and rainfall. Readers requiring further information on climatic data are referred to this report. The overall conclusion arising from an assessment of the weather data collected is that the Bilbao Project lies in a very benign climatic zone which is ideal for undertaking mining operations on a continual, uninterrupted basis throughout the year.

Figure 5-4 Annual Maximum and Minimum Diurnal Temperature Variation at the Bilbao Prospect

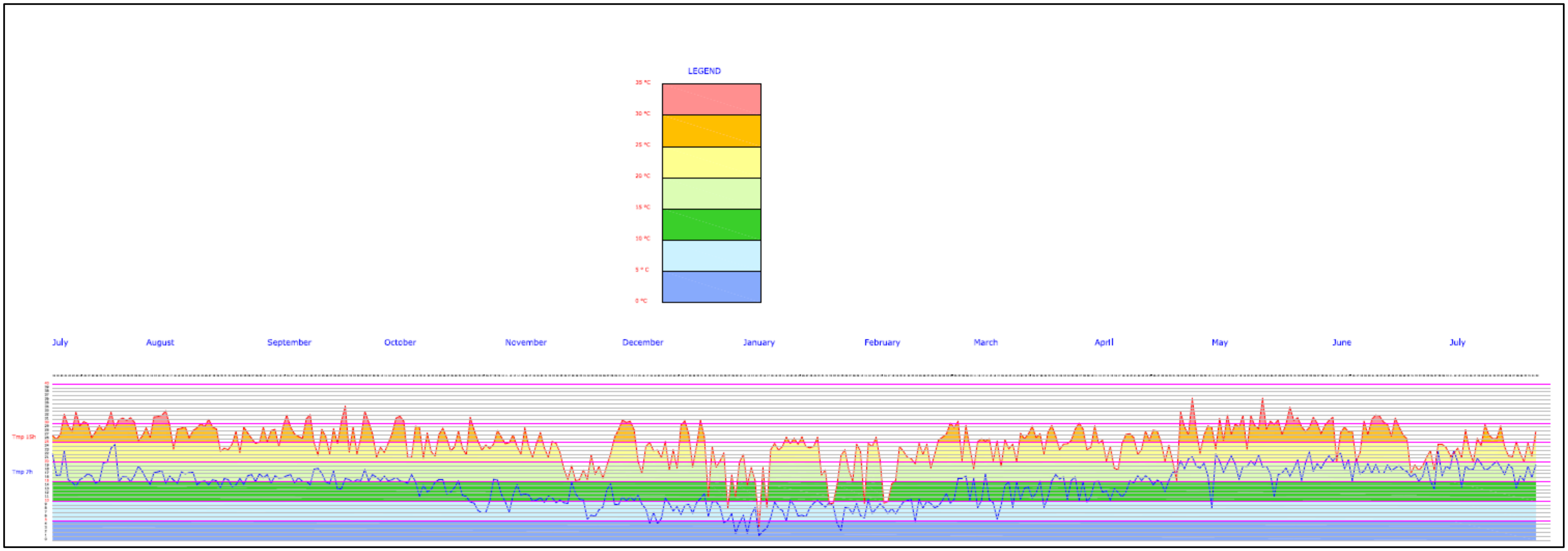
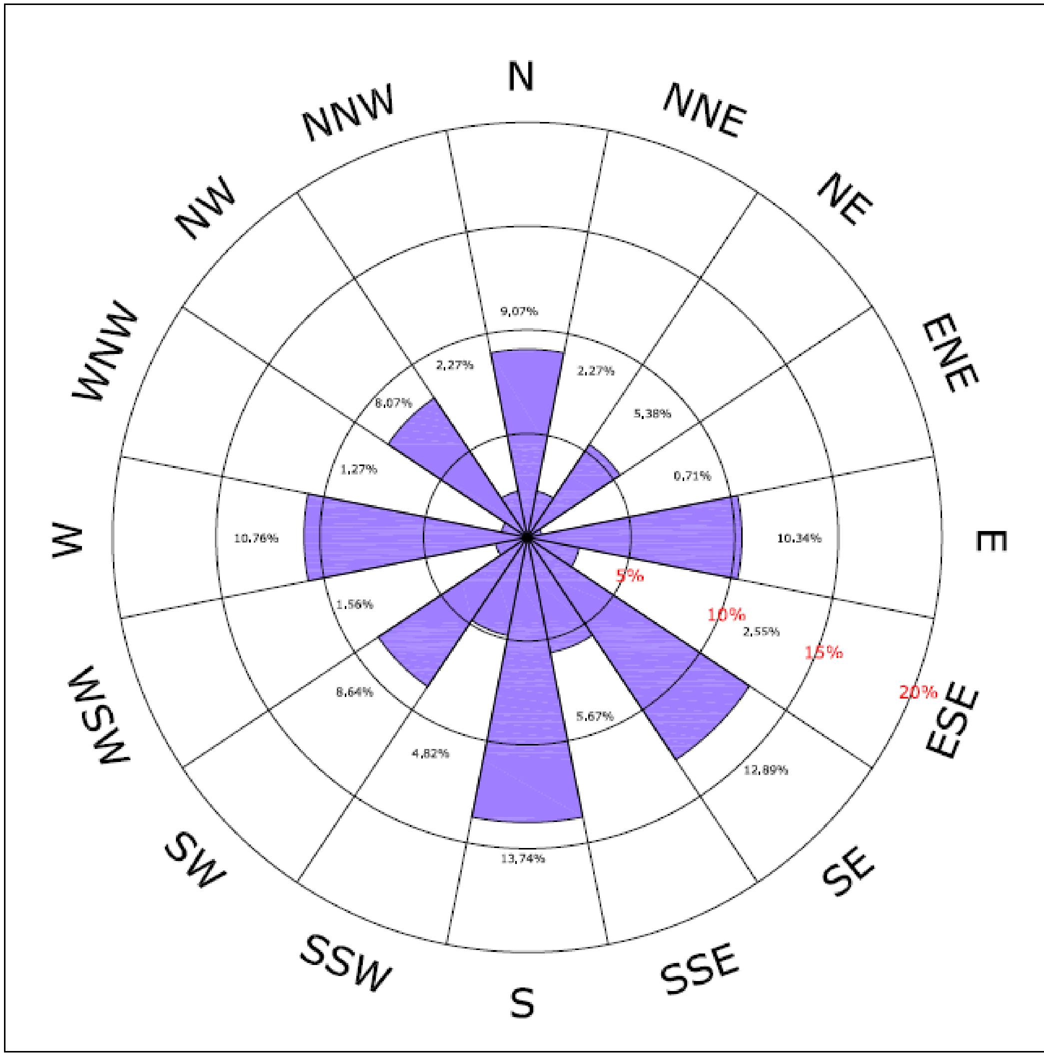


Figure 5-5 Wind Rose for Bilbao



07h00 & 15h00

WIND DIRECTIONS

	DAYS	%
MAGNETIC	07H15	% 07h15
S	97	13.74
SE	91	12.89
W	76	10.76
E	73	10.34
N	64	9.07
SW	61	8.64
NW	57	8.07
SSE	40	5.67
NE	38	5.38
SSW	34	4.82
ESE	18	2.55
NNE	16	2.27
NNW	16	2.27
WSW	11	1.56
WNW	9	1.27
ENE	5	0.71
	706	100 %

Table 5-2 Mean Monthly Weather Data for Bilbao

Month*	Season	Temp am	Temp pm	RH am	RH pm	Clouds am	Clouds pm	Wind speed am	Wind speed pm
January	Winter	6.8	19.9	65	33	34	45	5.8	9.9
February	Winter	8.0	18.9	70	37	59	65	10.9	12.9
March	Spring	11.8	25.4	51	20	28	30	6.8	12.9
April	Spring	13.9	25.0	51	20	20	33	6.6	10.1
May	Summer	18.3	28.6	59	21	9	27	8.4	9.2
June	Summer	**19.0**	28.5	65	29	19	52	6.0	6.6
July	Summer	17.9	23.1	52	**49**	**61**	**76**	9.2	11.0
August	Summer	16.6	**29.0**	69	31	22	62	4.2	7.7
September	Autumn	15.7	26.9	79	42	39	79	4.6	7.9
October	Autumn	14.8	26.2	78	36	16	55	7.2	9.7
November	Autumn	10.7	23.6	76	32	14	27	5.5	8.7
December	Winter	8.7	23.3	72	33	23	29	7.1	12.4
MEAN		**13.6**	**24.9**	**53**	**31**	**28**	**48**	**6.8**	**9.9**

**Temp 0C, RH= Relative Humidity in %, Clouds in % cover, Wind Speed in km/hr. Jan to July 2010, Aug to Dec 2009*

5.4 Local Resources

5.4.1 Power

Electric power is available from the Comisión Federal de Electricidad (CFE) and landline and cell phone telephone service is available from Telmex. The area is well supplied with electrical power with the main ultra-high tension national electricity power-line running north-south through the district at km138 of the Mexico 49 highway. This is less than 16km from the Bilbao shaft. On a more local level, the Pánfilo Natera electricity substation situated at 798.948/2510.675 is just 6km from the Bilbao shaft and has a capacity of 9.4MVA (Figure 5-6). It furnishes a 115KV output line that passes within 2km of the Bilbao shaft from which it is proposed to connect to the mine off pylon 41 at 22°39´0.80´´N/102°08´30´´W, to the mine site (Figure 5-7). This high tension line will be sufficient to provide the necessary power for the inital phases of work estimated at 4000kW. Moreover it has an inbuilt capacity to meet the final electricity demand at full production of 8500kW.

Hugo Renteria Felix of Servicios Electricos e Industriales, S.A. de C.V. furnished a report on the provision of electrical power to the Bilbao property entitled “Reporte General Estudio de Factibilidad CFE Proyecto Bilbao Mining to the Company in September 2009 concerning provision of power to the proposed mine site with quotes from the Comision Federal de Electricidad (CFE) from Jesus Reynoso Arzate, dated 12th June 2009 for Mexican Pesos 7,198,445 (roughly USD$600K) to provide such services.

Figure 5-8 shows the disposition of the extant electricity grid in the district.

Figure 5-6 Electricity Sub-Station near Pánfilo Natera

Figure 5-7 Powerlines Close to the Project Gate

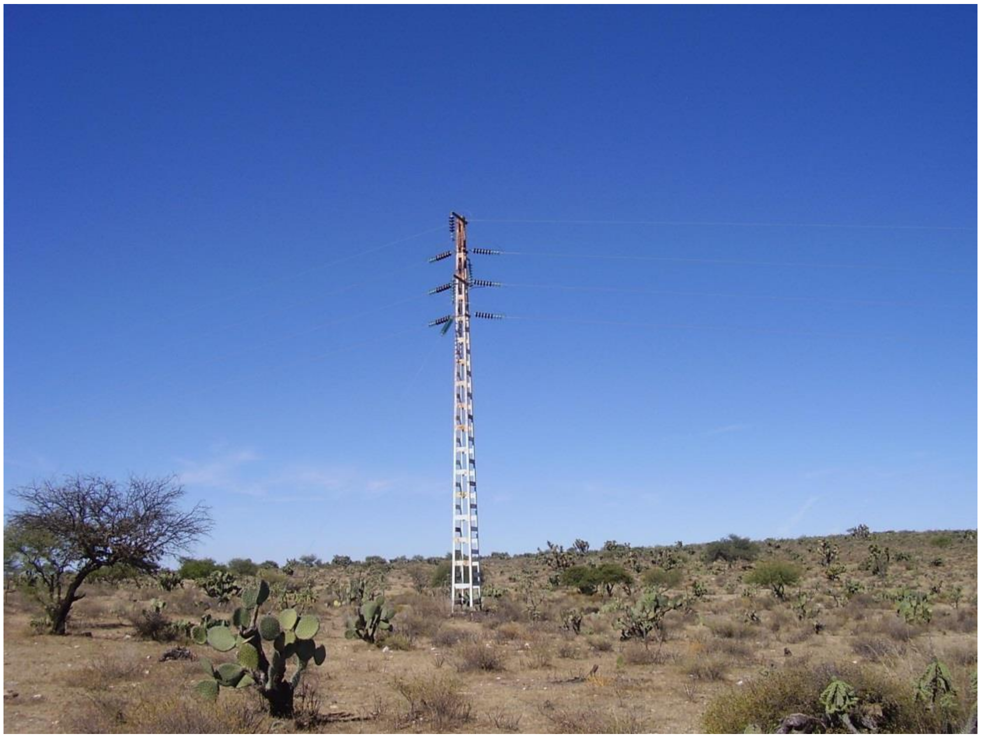
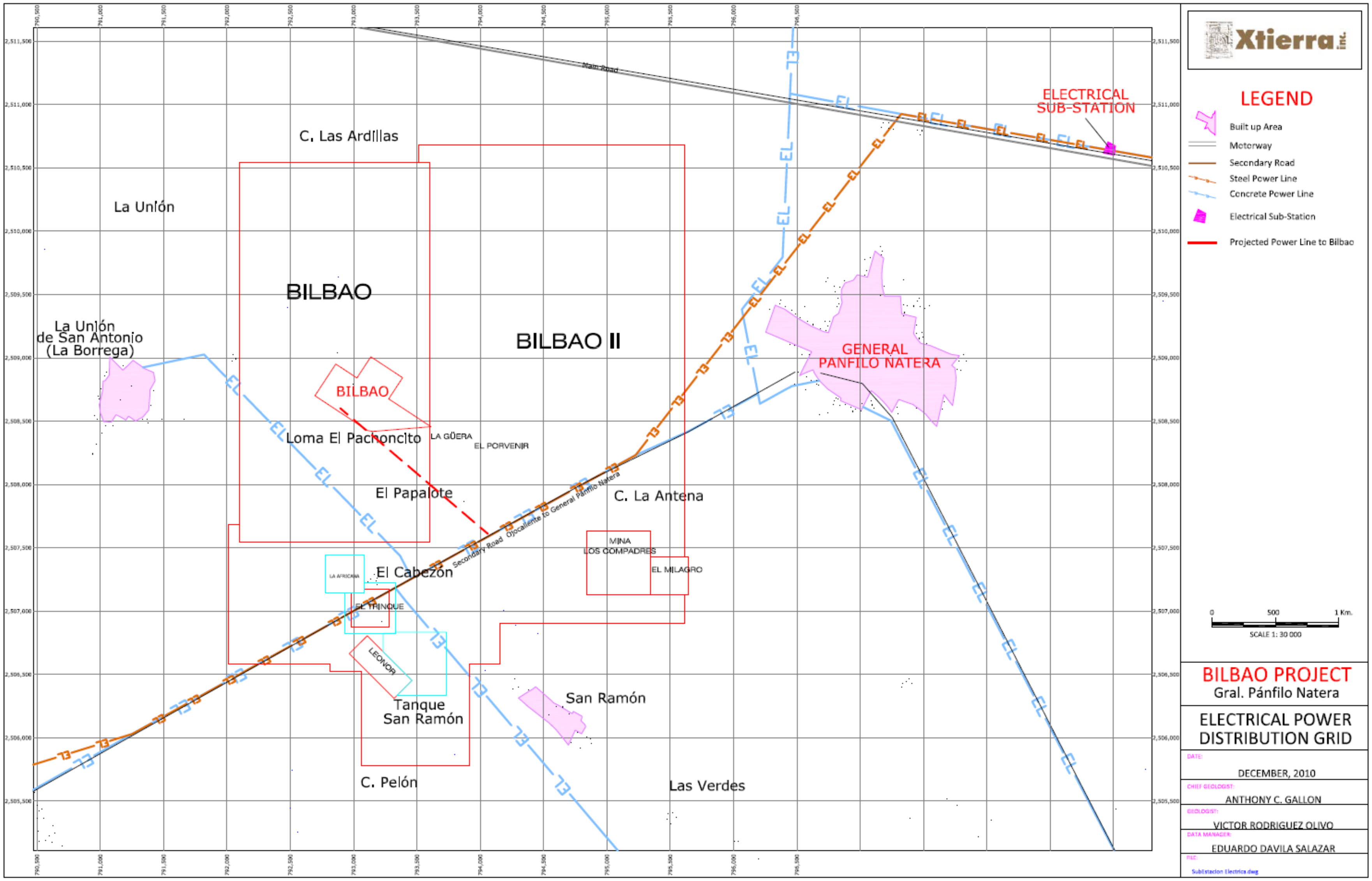


Figure 5-8 Availability of Electrical Power in the Vicinity of the Bilbao Project



5.4.2 Water

Given the arid conditions in Zacatecas, surface water is in short supply. There are no perennial rivers and the few water courses that have developed only run in the wet season after heavy rains. The local aquifers have been lowered over the past few years and groundwater is scarce.

A preliminary in-house study of the water resources in the Pánfilo Natera district was undertaken by the Company entitled "*A preliminary assessment of water resources available to the Bilbao Project*", Report # 174 of 15th Nov 2008 in which a list of water boreholes in the area was catalogued and potential water sources identified. This was followed up by a more detailed assessment of the hydrological regime by Schlumberger Water Services in December 2009. They furnished a report entitled "Bilbao Hydrogeologic Assessment (Phase I), Bilbao Silver-Lead-Zinc Property, Pánfilo Natera Mining District". Rpt 8003/R3, based on information obtained from the Zacatecas branch of the Comisión Nacional del Agua (CONAGUA), the national water management agency of Mexico.

The Bilbao project lies in the very northeast corner of the Ojo Caliente aquifer basin close to the triple junction with two others, La Blanca and Chupaderos. The aquifer basins are not well constrained but are related to an impervious substrate beneath sandstone delta spreads filling palaeovalleys in subsidiary faulted grabens (grabau). Within these three aquifers there are more than 1600 groundwater extraction wells officially registered by CONAGUA most of which are dedicated to agricultural use. Because the Bilbao property is located close to the recharge area of the three aquifer basins it has the advantage of choice on where to best source its water needs.

The water has low concentrations of dissolved solids such as sulfate and chloride which should minimize complications in hydrometallurgy. The aquifers are capable of supplying the required volume of water to the mine which is estimated at 2000m^3/day which equates to 23 litres per second. This could potentially be supplied by just a single well. It should be noted that within the surrounding aquifers there are strict groundwater pumping restrictions that preclude the granting of additional water rights in all the basins of Zacatecas. These bans are called "zona de veda" and effectively cap the volume of groundwater that can be removed from each basin and prohibit the granting of any new water concessions. It will therefore be necessary for the Company to purchase the water rights from already existing permitted wells and have the water rights officially transferred to their ownership, or transferring rights from an existing well to any new well contemplated.

Schlumberger Water Services have undertaken a preliminary evaluation of the potential water resources available to the Bilbao mine and conclude that well XOC-36 at Rancho San José (Marcos Alvarez Delgado) is likely to be the most useful (Figure 5-9). However this can supply only 240,000m^3 per year to a total of the 730,000m^3 required. Additional wells have been identified which could furnish the rest of the required water and there is the potential for capping an existing well and substituting this by a new well in the district. In this regard the sediment-filled graben to the north west of Bilbao has been identified as highly prospective for sourcing further water resources. Schlumberger undertook a study of this zone in early 2011.

A further technical memorandum by Schlumberger dated 22nd November 2010 and entitled "Summary of water availability for the Bilbao mine supply" notes that the lower levels of the open pit appear to be saturated which implies that a pore pressure management system will be required to help stabilize the pit walls. Furthermore recent data confirms that the regional groundwater flow is from SE to NW from the La Blanca (Aguascaliente) to the Chupaderos grabens following a structural control that is likely to contribute to groundwater inflows to the proposed pit. These aspects are being examined at the time of writing.

The various costs involved in acquiring the water rights and permits to construct a well would be between USD$342K and USD$444K. Construction of a new well or several wells would be between USD$150K and USD$400K and annual consumption fees for a volume of 730,000m^3 would be between USD$445K and USD$565K dependent upon which municipality the groundwater is won from.

Figure 5-9 Water pump in the Vicinity of the Bilbao Project on Marcos Alvarez´s Farm

5.4.3 Proximity to Town

The Bilbao project lies in close proximity to the town of General Pánfilo Natera (known colloquially as La Blanca), the distance between the Bilbao shaft and the centre of this town being 4km. The town itself has a population of about 22,000 with the usual ancillary services for such a population (Figure 5-10 through Figure 5-13). In particular local garage and repair services are available and there is a gas/petrol station just 1.3km from the project entry gate (Figure 5-14). Should complex engineering be required all services are available in the surrounding cities of Zacatecas or Aguascalientes. Simple hotel accommodation is available within the town.

Pánfilo Natera itself is connected to all surrounding urban centres by frequent local bus services.

5.4.4 Population

According to a census undertaken by the Instituto Nacional de Geografía e Informatica (INEGI) in 2005, the population of the various towns in descending order within the Pánfilo Natera area is as listed in Table 5-3.

From Table 5-3 it can be seen that generally there are more females than males in the area. The reason for this is that males frequently migrate outside the district and country to find work. (It is estimated that there are more Zacatecanos resident in Los Angeles than there are actually within the state itself) The closest town to the Property is General Pánfilo Natera which has a population of almost 22,000.

Figure 5-10 Scene in Pánfilo Natera town

Figure 5-11 Church in Pánfilo Natera

Figure 5-12 Town Scene in Pánfilo Natera

Figure 5-13 Town Scene in Pánfilo Natera

Figure 5-14 Gas-Petrol Station near the Project Gate

Table 5-3 Population of the principal towns in the Pánfilo Natera district in 2005

TOWN	MALE	FEMALE	TOTAL
San Luis Potosí	320,344	350,188	670,532
Zacatecas	59,493	64,406	123,899
Guadalupe	53,009	56,057	109,066
Pinos	31,075	33,340	64,415
Loreto	19,669	20,252	39,921
Ojocaliente	18,603	19,616	38,219
Villa de Ramos	16,776	17,656	34,432
Salinas	12,585	13,820	26,405
General Pánfilo Natera	**10,424**	**11,265**	**21,689**
Villa Hidalgo	7,577	8,169	15,746
Noria de Ángeles	6,792	7,022	13,814
Villa González Ortega	5,566	6,304	11,870
Luis Moya	5,525	5,893	11,418
Lagunillas	3,253	3,285	6,538

5.4.5 Mining Personnel

The economic activities around Pánfilo Natera are mainly concerned with agriculture but there is residual local manpower available versant and skilled in mining activities and who have worked on local mines and mineral properties in the district. Within a 50km radius there is a wealth of mining experience since the Zacatecas region has been the locus of mining activities for over the past 500 years since Spanish colonial times.

Within the region as a whole there are skilled miners, mineral processors, heavy machinery operators, civil and mechanical engineers, in fact all personnel required for mining activities can be sourced locally. In addition there are commercial mineral analytical laboratories available in Zacatecas city and all necessary heavy construction works can be undertaken by local companies either from San Luis Potosí, Zacatecas or Aguascalientes.

5.5 Physiography

The Bilbao area is located on a flat plateau in the highland (Meseta) of Mexico centrally between the two NW-SE trending mountain ranges of the eastern and western Sierra Madre (Figure 5-15). This situation causes the area to be in the rain shadows of both ranges and so limits the amount of precipitation from both westerly and eastern directions. The result is low rainfall which is reflected in the xerophytic vegetation typical of the area.

The project itself is located at an elevation of between 2045 and 2170m (6700-7120 feet) amsl with the main shaft situated at 2161m (7090 feet). The topography at the project is one of very gentle slopes within flat farmland with the actual outcrop of the mineralization occupying the highest point, 2170m, on a minor rounded hillock (Figure 5-16). The property is both large enough and topographically suitable for the development of facilities such as waste dumps and tailings disposal areas. A topographic map of the area is provided as Figure 5-17.

As concerns the vegetation, the land on which the mineralization is situated is classified as rough cattle grazing land unsuitable for cultivation of crops (Figure 5-18). The overall outlook is reminiscent of open thorn-bush grassland interspersed with Opuntia cacti and Yucca succulents (Figure 5-19Figure 5-20). The understory is characterized by poor grass cover which is utilized for cattle grazing (Figure 5-21).

A general environmental study of the Bilbao district entitled “Proyecto de Exploración Minera, Gaby Marina/Bilbao” was prepared for the company by an independent consultant, Marcial Chavez Quinto, and was lodged at the Secretaria del Medio Ambiente Y Recursos Naturales in March 2006 and further studies were completed during 2006 by Joel Espinosa Rivera of Bufete de Services Tecnicos Forestales y de Fauna Silvestre. This report entitled “Proyecto: Bilbao, Aspectos biologicos, climaticos y vias de acceso” concentrated on the flora and fauna found at the Bilbao prospect.

There are no endangered plant species within the claims although it is recognized that six species of cactus, mainly Echinocactus & Ferocactus species, deserve protection (Figure 4-5). If these occur in areas required to develop the mine, they will be translocated within the property. Most of these species requiring protection occur on the sparsely grazed basaltic hill, Loma El Pachoncito (Figure 5-22), which is within the Bilbao claims but peripheral to the projected mine and plant sites.

In additional to the study of the flora, Dr A C Gallon has prepared a preliminary checklist of the birds occurring within the Bilbao claims. The number of species recorded by casual observation is 65; all either typical of a thorn-bush fauna or passage migrants. A record of two overflying Golden Eagles (Aquila chrysaetos) on 7th October 2008 is perhaps the most noteworthy record. Two ephemeral water dams attract migrating waders/shorebirds on a casual basis but the claims have no special ornithological merit when compared with surrounding areas (Figure 5-23). The environmental report prepared by Rivera is entitled ”Environmental Report on the Bilbao Property” and the report entitled “The Bird Fauna of Bilbao” lists the birds so far identified on the property.

Figure 5-15 Flat Plateau of the Mexican Meseta in the vicinity of the Bilbao project

Figure 5-16 Outcrop of Siliceous Gossan on Hillock South of Glory Hole 1

Figure 5-17 Topographic Map of the Bilbao District

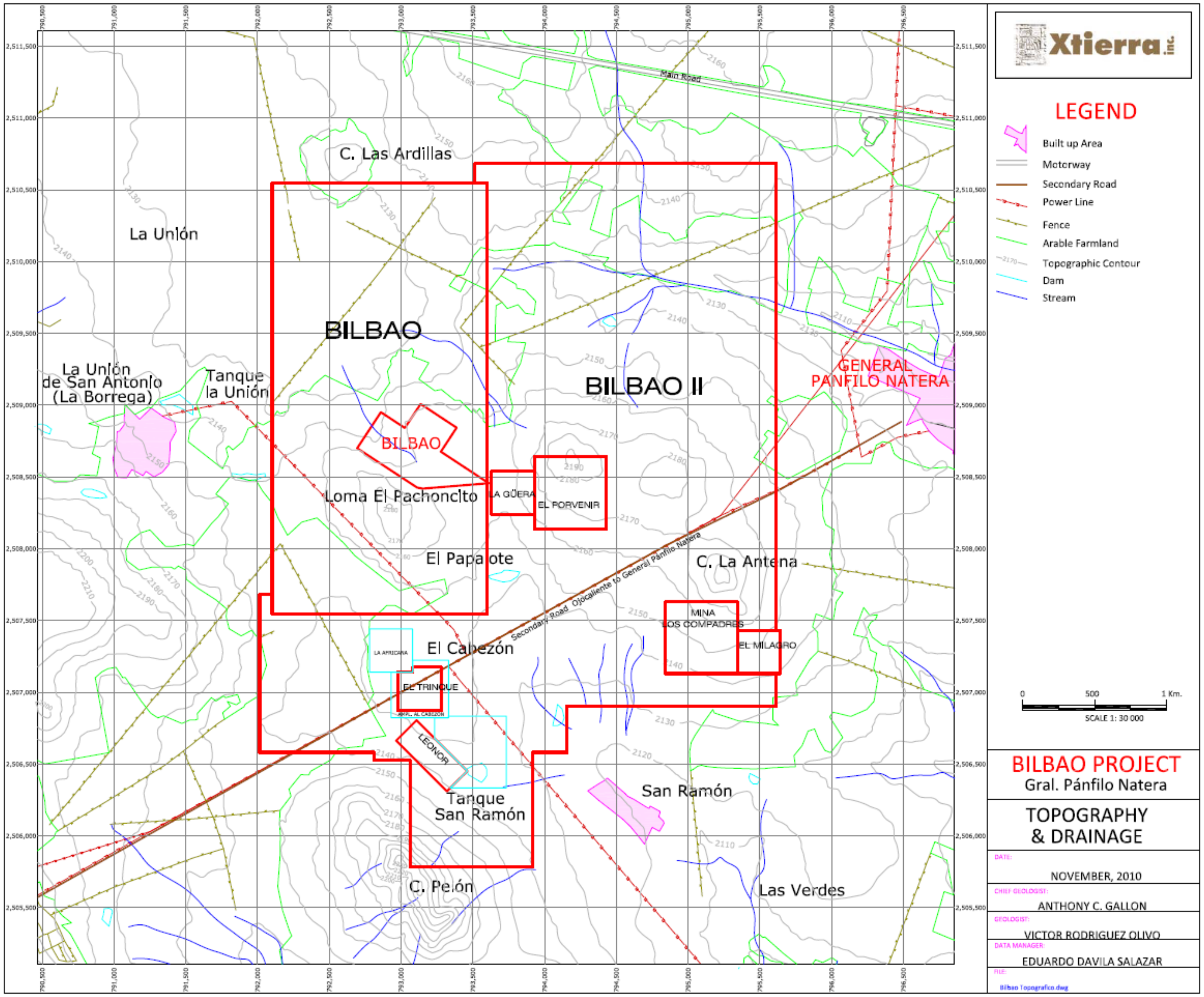


Figure 5-18 Rough Grazing Land on the Bilbao Claims

Figure 5-19 Opuntia Vegetation on the Bilbao Claims

Figure 5-20 Yucca Vegetation on the Bilbao Claims

Figure 5-21 Acacia and Cactus Scrubland in the Bilbao Claims

Figure 5-22 Loma El Pachoncito Hill, Basalt

Figure 5-23 Water Dam at El Papalote in the South of the Bilbao Project

6. History

6.1 Prior Ownership & Work undertaken before Xtierra involvement

The early development of the mine is not well documented but began in the early 1900's with development, by International Mining Company, of the Main and Azulaques shafts and driving on the 40 and 76m levels within the oxide mineralization.

The earliest recorded ownership of the Bilbao property dates back to 1928 when a North American investor, who controlled the property, defaulted on a loan from the U.S. Government and the Property went into receivership. In 1929, Mr. Sutti Snr, a mining engineer, acquired the property, repaid the loan and sold the property before the stock market crash of 1929 to Companía Fresnillo. Mr. Sutti Snr continued to be involved with the property through the end of World War II. During that time he was credited with sinking the Main Shaft (Figure 6-1) and construction of a narrow gauge railway to the main line in Guadalupe.

Figure 6-1 Main Shaft at the Bilbao Project

Archival records indicate the run-of-mine oxide mineralization was a direct shipping ore, which was initially shipped to the lead smelter in San Luis Potosí during World War II, and subsequently to Asarco's smelter in El Paso, Texas. The historical production has been estimated to be about 1.0 million tonnes. Companía Fresnillo (subsequently merged into Peñoles) held the property until it was dropped in 1986, in the belief that the deposit was a roof pendant and too small to be of significant value. No drilling appears to have been carried out at Bilbao prior to the Company's involvement.

In 1989, Martin Sutti Courtade acquired mineral exploration concessions over Bilbao and subsequently sold them to Minera Portree de Zacatecas, S.A.de C.V. Minera Portree retained Watts Griffith & McOuat ("WGM") to prepare a resource estimate of the remaining oxide mineralization above 2090m elevation (76m level) using data collected by sampling the underground workings in 1992-93. This was reported to be 3,211,400 tonnes grading 3.32% zinc (Zn), 3.76 % lead (Pb), 0.36% copper (Cu) and 76g/t silver (Ag).

In 1995, Kilborn Engineering ("Kilborn") prepared a Prefeasibility Study on the property, which was subsequently revised in 1997. The results of Kilborn studies presented a case for open-pit mining with differing scenarios for metallurgical processing and metal recovery. The best alternative had an open-pit oxide mining resource of 2.44 million tonnes averaging 3.73% Zn and 0.30% Cu (lead and silver were not recoverable using their proposed process methodology.)

Several companies subsequently optioned the property and Minera Portree's other concessions in the district, between 1989 and 2004, including Cyprus/Phelps Dodge who mapped the district and completed geophysical and geochemical surveys over favorable targets, including Bilbao.

In July 2004, Minera Portree sold the property back to Martin Sutti Courtade who then re- sold it to Shoshone Mexico, S.A. de C.V. in October of that year. In May 2005, Shoshone commissioned a geological report on the project by independent geologist, René G. von. Boeck.

Over the years oxides of copper/ lead/zinc and silver were mined sporadically on the property until about 1953. Wollastonite was also mined on a small scale to mid-2006. Visually the "ore" contained approximate 50% wollastonite by volume, which had to be hand-cobbed to produce a saleable product. The occurrence therefore suffered from competition from producers with grades up to 100%. The wollastonite was worked by the Mendez family who live locally at Noria del Cerro.

Additionally, a small limestone quarry located on the Bilbao claim at L794.904/P2508.659, southwest of Pánfilo Natera produced block limestone and road aggregate.

On February 27, 2006 the Company optioned the property from Shoshone and acquired a 100% stake in August 2008. Subsequent core drilling commencing in May 2006 was successful in demonstrating significant sulphide mineralization down dip of the shallow oxides.

6.2 Historical Mineral Resource Estimates

Resource estimates were prepared in 2007 on the basis of 28 diamond drill holes by Jeff Aucott of Mining and Exploration Geosystem Associates ("MEGA") and independently by R.T.G. Parker in 2007, 2008, 2009 and 2010.

The MEGA estimate was performed by wireframe modeling of the individual mineralized lenses and block modeling using Datamine software. Grade interpolation was performed using inverse distance squared as there were insufficient samples for Kriging.

A separate manual resource estimate was carried out in February 2007 by R. Parker, using the same database as MEGA, and employing the same 3% zinc equivalent cutoff grade. This estimate was performed by the polygonal cross-section method based on the northwest-southeast vertical geological sections prepared by the Company.

In March 2007, the Company retained A. H. Summers, B.Sc., P.E, P.G.; Consulting Engineer, to prepare a prefeasibility or scoping study on the Bilbao Project. This study was based on the MEGA (2007) estimates for the sulphide and mixed resources below the 2078 metre elevation.

The Summers´ study proposed ramp access and trackless mining, at the rate of 1,000 tonnes of ore per day, using sublevel stoping with delayed cemented backfill. Alternative methods, such as cut- and-fill, or room-and-pillar, could also be considered where the deposit is too thin for sublevel stoping.

A summary of the conclusions which Summers presented is as follows:

- The ore would be processed using flotation to produce silver/lead/copper, and zinc concentrates, and future metallurgical test work will evaluate the feasibility of applying sulfidizing techniques to maximize the recovery of oxide minerals. This study assumed the concentrates would have been processed by Peñoles in Torreón, but more favorable terms could be obtained by selling them overseas.

- The economic analysis was based on processing 330,000 tonnes of ore per year (1,000 tonnes per day) for eight years. This follows a 10 to 12 month program of drilling to upgrade and increase the oxide and sulfide resources, together with metallurgical test work and detailed design of the proposed sulfide operation at a cost of US$3.0 million. Bilbao would require an additional preproduction investment of US $43.3 million dollars, including US $8.3 million of contingency and working capital, and it would take 1½ years, following receipt of Notice-to-Proceed, to construct the mine-mill operation, including the development of a 1½ kilometre long access ramp. The operating costs, on a US $ cost per tonne mined and milled basis were estimated as $35.40 per tonne milled, which agrees closely with the operating costs of similar mines in Mexico.

- An economic evaluation using "Base Case" metal prices (US $) of: Silver $12.50 per oz, Lead $0.60 per lb, and Zinc $1.50 per lb, resulted in a pay-back Period of 1.53 years, a Net Present Value $71.0 million @ 10% discount rate, and an Internal Rate of Return 49.2%. The project was said to break-even if the zinc price dropped as low as US $0.60 per lb.

The parameters on which the Summers study was based had then been entirely superseded due to much changed metal prices, an increased resource base and improved oxide metallurgy, all of which suggested that open pit mining will provide a more feasible option.

The following table summarized the results of the manual resource estimate using a cut-off of 6% Zn_{eq}, completed by Richard Parker, Senior Geological Associate, Southampton Associates Inc. on the Bilbao Deposit and dated July 11th, 2008.

Table 6-1 Bilbao manual resource estimate of Parker 2008 using a 6% Zn_{eq} cut-off

Resource Category	Tonnage (tonnes)	Zinc (%)	Lead (%)	Copper (%)	Silver (g/t)	Zinc Equivalent (%)
Indicated Resources	3,600,000	3.53	2.75	0.29	88.23	10.1
Inferred Resources	2,380,000	2.52	2.79	0.28	83.08	8.95

Table 6-2 below summarizes the various components of the Parker resource estimate.

A more recent Parker resource estimate of February 2010, tables a significantly higher tonnage with commensurate lower grades than were given in previous estimates. These were as follows:

- Indicated Resource; 9.68 million tonnes at 2.09% Pb, 0.21% Cu, 2.43% Zn, 59.4g/t Ag (6.7% Zn_{eq})
- Inferred Resource; 4.04 million tonnes at 1.55% Pb, 0.17% Cu, 1.43% Zn, 53.64g/t Ag (4.93%Zn_{eq})

Table 6-2 Resource Estimate by resource category and mineral type: 2008

	Detailed Resource Category by Mineral Type								
Resource Category and Classification	Tonnes	Zinc (%)	Lead (%)	Copper (%)	Silver (g/t)	Zinc (tonnes)	Lead (tonnes)	Copper (tonnes)	Silver (tonnes)
Indicated Resources									
Oxide	1,030,000	2.35	3.1	0.39	85.08	24,227	31,932	4,042	88
Mixed	757,000	3.52	2.7	0.21	88.67	26,595	20,464	1,593	67
Sulfide	1,815,000	4.2	2.56	0.27	89.84	76,299	46,504	4,869	163
TOTAL	3,601,000	3.53	2.75	0.29	88.23	127,122	98,900	10,503	318
Inferred Resources									
Oxide	1,324,000	2.28	2.86	0.31	84.55	30,199	37,884	4,064	112
Mixed	516,000	2.58	2.94	0.25	89.14	13,328	15,173	1,281	46
Sulfide	538,000	3.05	2.49	0.26	73.65	16,400	13,376	1,393	40
TOTAL	2,378,000	2.52	2.79	0.28	83.08	59,927	66,433	6,738	198

The higher tonnages in this resource estimate were due to a number of factors, chief amongst which were:

- Use of a lower cutoff grade, 3% Zn_{eq}, compared with the 6% Zn_{eq} used for previous estimates
- Inclusion of infill drilling undertaken after 2007
- Inclusion of surface and underground channel samples taken after 2008
- Consideration of a larger cell size and the associated greater dilution
- And noting that manual polygonal methods would normally overestimate grade

Table 6-3 below summarizes previous resource estimates made for the Bilbao Project.

Table 6-3 A comparison of Historical Resource Estimates at the Bilbao Project

Year	Author	Method	Cutoff	Category	Million tonnes	Pb%	Cu%	Zn%	Ag g/t	Zneq	Tonnes ZnEq	Notes
1995	Kilborn/Aucott	ID2/Polygonal	6	Inferred	**5.79**	2.61	0.34	3.28	85.4	9.16	530,219	combined Oxide of Kilborn,& Sulfide of Aucott
2007	MEGA	Inverse Distance	3		**2.2**	2.64	0.24	4.09	83			Sulfide + Mixed
2007	MEGA	Inverse Distance	3		**1.23**	1.24	0.25	2.3	ND			Oxide
2007	Parker	Polygonal	6	Inferred	**2.03**	2.8	0.27	4.28	85.45	10.14	205,924	Prior to infill drilling, below 2078m, no oxide
2008	“Blue Book”	Sectional	>4% Pb+Zn	Indicated	**2.82**	2.43	0.25	3.28	97	NA		Sulfide
2008	“Blue Book”	Sectional	>4% Pb+Zn	Indicated	**0.94**	3.57	0.3	2.85	80	NA		Mixed
2008	“Blue Book”	Sectional	>4% Pb+Zn	Indicated	**1.61**	2.72	0.24	2.61	83	NA		Oxide
2008	Parker	Polygonal	6	Indicated	**3.6**	2.75	0.29	3.53	88.23	9.47	341,047	after infill drilling, Sulfide,Oxide and mixed
2008	Parker	Polygonal	6	Inferred	**2.38**	2.79	0.28	2.52	83.08	8.34	198,571	after infill drilling, Sulfide,Oxide and Mixed
2009	Parker	ID2	6	Indicated	**4.8**	3.04	0.38	3.61	81.08	9.94	477,065	2 metre cells
2009	Parker	ID2	3	Indicated	**8.49**	2.23	0.23	2.6	64.43	7.22	613,209	2 metre cells
2010	**Parker**	**ID2**	**3**	**Indicated**	**9.68**	**2.09**	**0.21**	**2.43**	**59.4**	**6.72**	**650,565**	**10 metre cells,**
2010	**Parker**	**ID2**	**3**	**Inferred**	**4.04**	**1.55**	**0.17**	**1.43**	**53.64**	**4.93**	**199,010**	**10 metre cells**
2010	**This Report**	**Sectional**	**2.5 Zneq**	**Total**	**12.49**	**1.85**	**0.19**	**1.97**	**59.19***	**5.99**		

*excludes X26 Ag

As per Table 6-3, Parkers´ estimates utilizing 2 metre & 10 metre cells at a cut-off of 3% Zn_{eq} are broadly comparable with the current estimates given in this report. However, the use of such large 10m cells may overestimate tonnage, particularly when the cells overlap abrupt mineral contacts at the edges of the main body of mineralization or when applied to horizons, less than 10m thick, away from the main body of mineralization. Given that a slightly lower cut-off has been preferred in this report of 2.5% Zn_{eq} compared with 3.0% Zn_{eq}, and in cognizance of the different methods used in their calculation, the estimates are closely similar; that noted, the main objective of the latest drilling phase in mid-2010, was to define the oxide resource more precisely, so as to move “Inferred” resources to a higher categorization of “Indicated” status.

6.3 Previous production at Bilbao

Over the years oxides of copper/ lead/zinc and silver have been mined sporadically on the property. Wollastonite was also mined on a small scale to mid-2006. Visually the “ore” contained approximate 50% wollastonite by volume, which had to be hand-cobbed to produce a saleable product. The occurrence therefore suffers from competition from producers with grades up to 100%. The wollastonite was worked by the Mendez family who live locally at Noria del Cerro.

Additionally, a small limestone quarry located on the Bilbao claim at L794.904/P2508.659, southwest of Pánfilo Natera produced block limestone and road aggregate.

7. Geological Setting and Mineralization

7.1 Regional Geology

The regional geological setting of Central Mexico has been summarized by several authors including Nieto-Samaniego et al (2005), Chavez Martinez (1999) and Megaw (1999). This digest of the regional geology draws on these studies, is supplemented by other data in the public domain, and is enhanced by personal observations in the field. Figure 7-2 shows the regional geological setting for the area and Figure 7-1 shows a generalized stratigraphic column for the Central Meseta in the Pánfilo Natera district.

Mesozoic and Tertiary rocks predominate in the Central Meseta of Mexico. The area is a distinct geographic province variously called the Altiplano, Mesa Central or Central Meseta. It sits between the east and western ranges of the Sierra Madre and comprises a high plateau exceeding 1,700m with individual peaks up to 2,600m. This geographical entity is bounded by these two mountain ranges to the east and west, and terminated by the Trans-Mexican Volcanic Belt to the south.

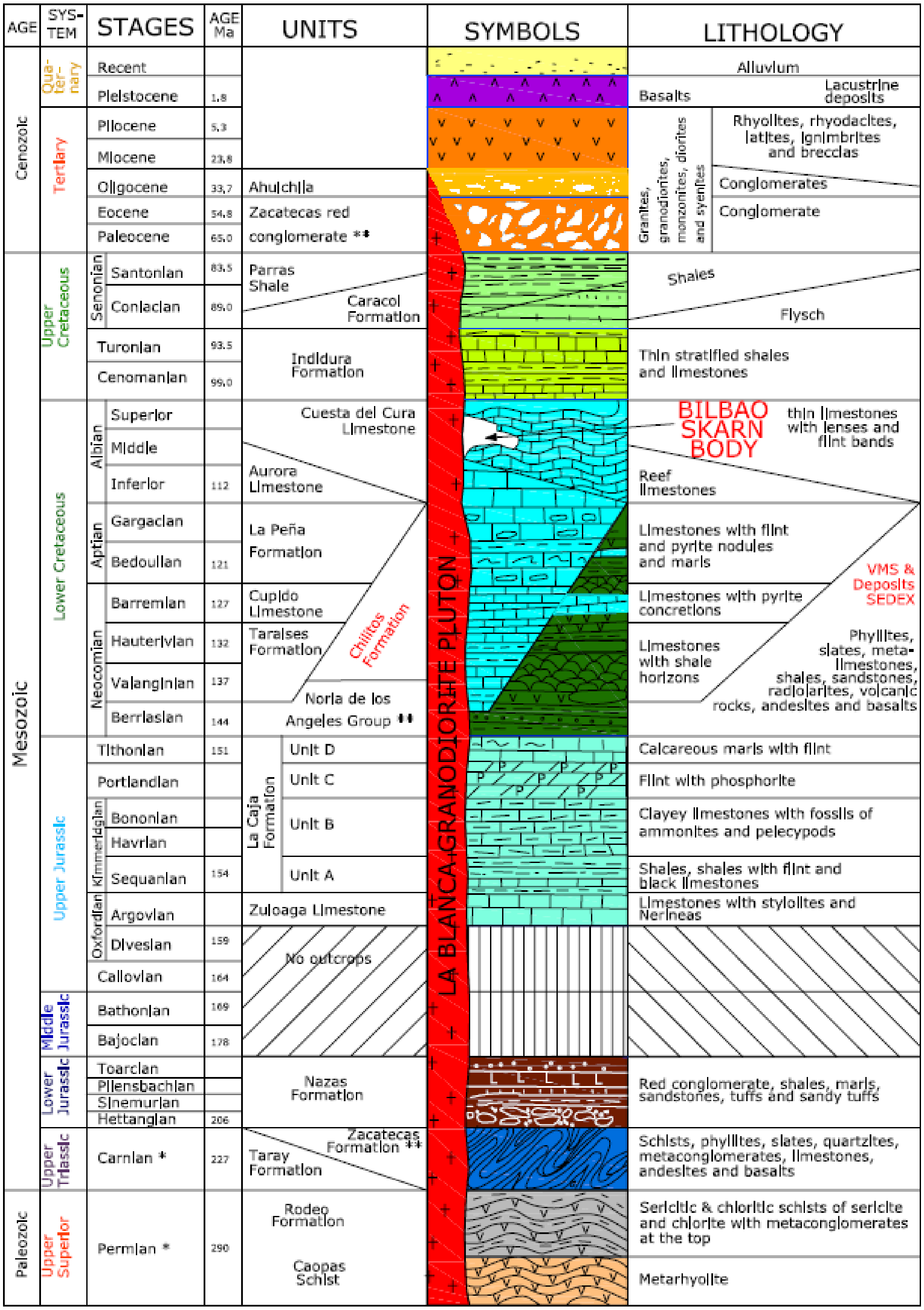
The crust of the Central Meseta is thinner (30km) compared with that beneath the Sierra Madre fold belts (40km) and is itself divided into two separate sub-zones of oceanic crust in the southwest and continental crust to the north-east. The junction of the oceanic and continental crust is marked by a crustal lineament, the San Luis Potosí-Tepehuanes Fault zone which is the focus for emplacement of many different types of mineral deposit in the area.

There are isolated basement outcrops of Paleozoic and Triassic rocks in central Mexico but these are much less common than Mesozoic and Tertiary sequences. Jurassic, and more commonly, marine Cretaceous calcareous sediments predominate with coeval Chilitos andesitic volcanic also developed. These earlier sequences were then intruded by the La Blanca granodiorite which resulted in contact metamorphism of the Cretaceous limestones and development of the skarnoid mineralization at Bilbao. Tertiary rhyolitic ignimbrites and rhyolitic intrusive dome rocks followed and were themselves covered by Oligocene topaz-bearing rhyolites. Late stage, Miocene-Quaternary alkaline amygdaloidal basaltic flows were then extruded in the Pánfilo Natera district as exemplified by thoseexposed on the Loma el Pachoncito, the hill immediately to the south-west of the Bilbao mineralization. The tuff and basalts actually overlie the Bilbao mineralization in the southwestern sector of the drilling grid.

Basin and Range faulting, reflecting a change from a compressional to an extensional environment, followed the Tertiary mineralizing phase resulting in deep grabens that were then filled with continental lacustrine and alluvial sediments during the Pleistocene. [Prospective sequences are likely to occur at considerable depth within these sediment filled grabens but mineralization would be difficult to discover.] The resultant Pleistocene sediment-filled grabens are important sources of groundwater and are being explored as potential sources of water for the Bilbao Project; they now form the flat agricultural plains throughout the district.

The Lower Cretaceous Chilitos Formation is of importance in respect to prospectivity for VMS deposits. The Chilitos Formation rocks consist of andesitic and basaltic flows, sometimes pillowed, together with marine sediments including radiolarites, minor limestones, sandstones and some black shales. The common manifestation of the Chilitos Formation rocks in outcrop being purplish-green andesites which are often referred to colloquially as ―greenstones‖. The outcrops frequently show deformation principally as parallel low-angle thrust faults being manifestations of the compressional obduction of these rocks over the continental part of the sequence.

Figure 7-1 Generalized Stratigraphic Column of Central Mexico
(modified from Servicio Geologico Mexicana map F13-B69, Ojo Caliente)



Rock sequences prospective for skarnoid type mineralization are not so stratigraphically constrained but rather depend upon what rocks the mineralizing granodiorite intrusives cut. The granodiorites are seen as the source of the mineralizing fluids but emplacement of these is dependent upon favourable channel ways and host rocks. Limestones adjacent to granitic intrusives are the favoured host rocks for mineralization.

7.2 Regional Mineralization

Bilbao is located within the northwest trending Mexican Silver Belt, a 600km long linear structure, centred on the San Luis-Tepehuanes fault system (―STFZ‖) which is coincident with a subduction zone between the oceanic and continental crust. The STFZ is the locus of major epithermal silver vein deposits including Sombrerete, Fresnillo and Zacatecas (Figure 7-2) which have accounted for a large proportion of the silver production of Mexico. These deposits may occur along the primary geo-suture or, more commonly, along complementary structures associated with it. The development of these late-stage high grade ―bonanza‖ type silver-gold vein deposits depends primarily upon development of hydrothermal convection systems within subaerial acid volcanic sequences (such as the Miocene/Pliocene rhyolites found in central Mexico) and emplacement within fault structures at boiling zones within them.

Other types of mineralization occurring within the Mexican Silver Belt include volcanogenic massive sulphide (VMS), Sedimentary Exhalative (Sedex) in the marine sequences and skarns, replacements, mantos and stockworks. VMS deposits are the oldest in the district being restricted to the obducted marine (oceanic) volcanic dominated sequences, whereas mineralization in the predominantly carbonate Mesozoic sequences is much younger and emplaced in structures caused by deformation during the Laramide orogeny between 80 and 40 million years ago. The age of the latter mineralization is contemporaneous with the emplacement of Tertiary extrusives and intrusives, particularly granodiorites, in the period between 50 and 30Ma.

The mineralized belt would appear to be related to mineral fluid generation processes at the subduction zone interface and it is interesting to speculate whether original VMS sulphides have been remobilized to be redeposited again as carbonate replacement deposits (CRD‘s) as suggested by Gallon (2006) at Bilbao, and first mooted as a possibility to be addressed regionally by Megaw (1988).

The concentration of deposits in the Pánfilo Natera exploration area lies at the junction of the later Aguascalientes graben with the STFZ that may reflect a reactivation of the older structure. Erosion of these rocks has resulted in development of minor, Pleistocene, tin and gold placers. Saline lakes occupy the deeper parts of the grabens and are exploited for salt, as at El Tule, just east of the Bilbao prospect and at Salinas some 40km to the east. There is also a potential for the development of lithium brines in these saline lakes.

7.3 Geology of the Pánfilo Natera District

The Pánfilo Natera Mining District overlies the central portion of the La Blanca batholith, a north-south elongated Tertiary granitic intrusive consisting mainly of monzonite and granodiorite that intrudes Jurassic and Cretaceous sedimentary rocks. The granitic intrusive has been eroded to a gentle topography. It does not form any significant hills. The Tertiary rhyolite porphyry intrusive plugs form the largest and tallest hills in the district with the El Morro hill particularly prominent.

The sediments include mostly massive marbled limestones of Jurassic age overlain by medium to thin bedded limestones and carbonaceous shales of Cretaceous age. Andesites and other volcanic facies (Chilitos Formation) underlie the Cretaceous limestones. Large Tertiary rhyolite porphyry bodies and latite dykes are present, cutting the older rocks. Some of the dykes are emplaced along dominant northwest trending structures and may represent the late intrusive event. Most of the northwest trending structures are mineralized and have formed important vein systems that have been mined successfully in the past. Figure 7-3 shows the general geological setting.

Figure 7-2 Mineral Occurrences in the Silver Belt of Central Mexico

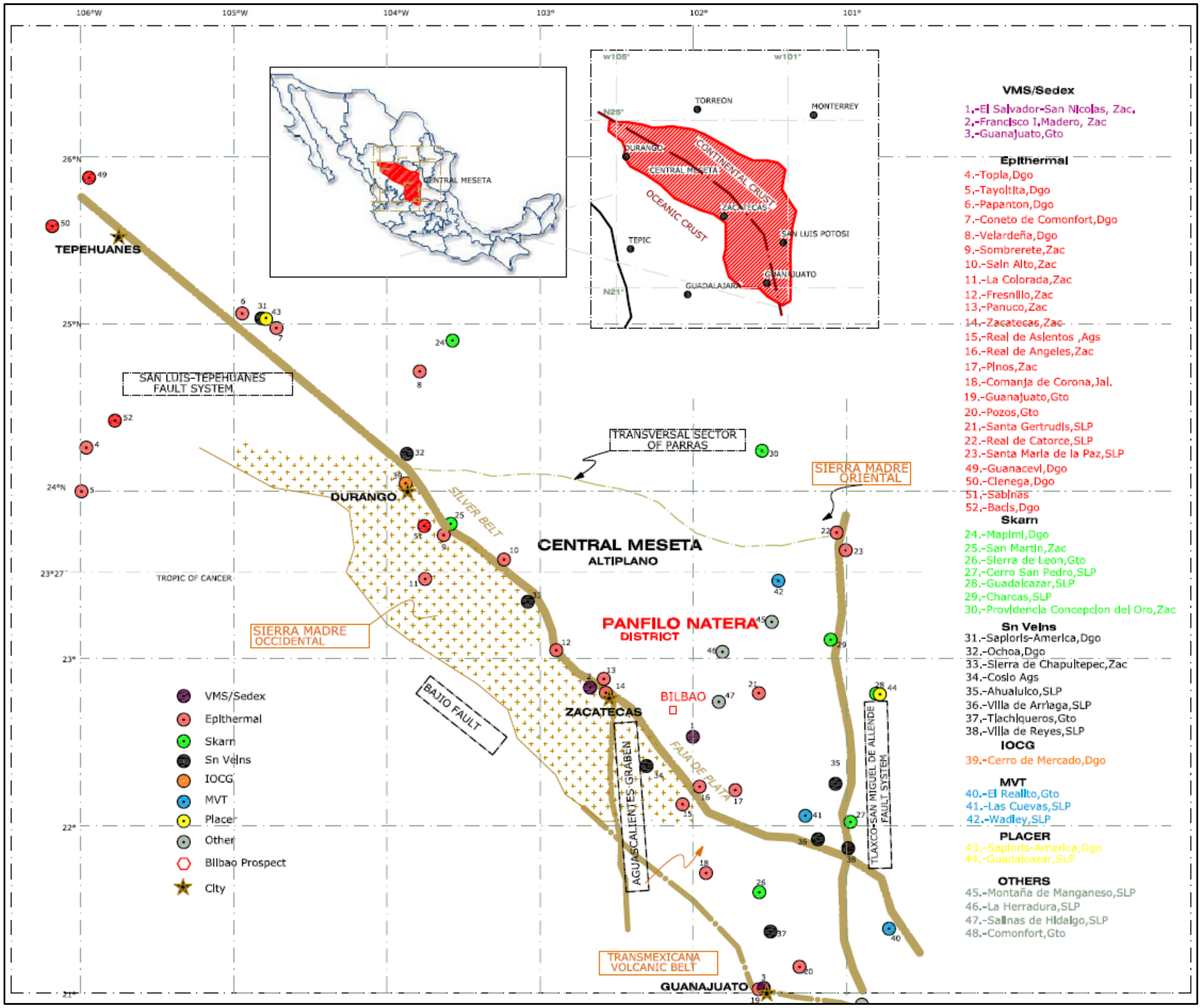
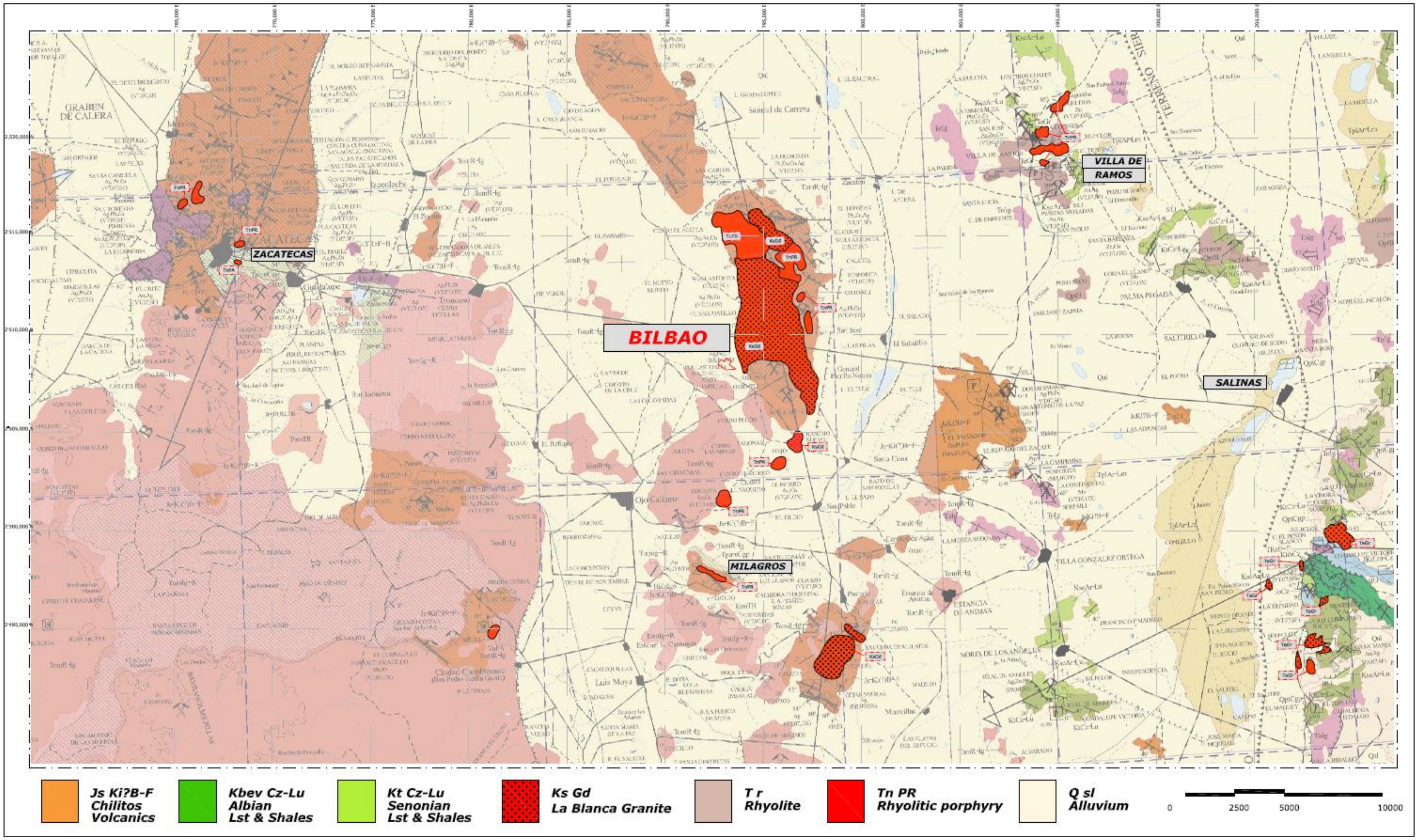


Figure 7-3 Regional Geology of the Pánfilo Natera District



Most of the sedimentary rocks outcropping in the district are roof pendants over the intrusive body. They form a northeast trending anticline. In places skarn zones are present at the contact of the intrusive and sedimentary rocks.

The dominant mineralization in the area is in the form of silver-lead-zinc fracture filling veins that cross both the intrusive bodies and the sedimentary rocks. The longest and strongest of these is the San José vein, 3.5km north of the town of General Pánfilo Natera, which has been mined until recently by a subsidiary of Minera Fresnillo and is now held under option by Arian Silver Corporation. Volcanogenic massive sulphide mineralization occurs at the San Nicolás (Teck-Cominco) and Real des Angeles (Frisco) deposits. Hydrothermal solutions that circulated through the fracture system, have deposited silver, lead, zinc and copper mineralization as limestone replacement deposits (such as Bilbao) in which there is a combination of skarn mineralization, carbonate replacement and fracture filling. The massive limestone is strongly silicified in the replacement zones. The primary ore minerals are galena, argentite and sphalerite, and their corresponding oxidized products.

Wollastonite deposits are related to the skarn zones and are present in lenses within marbled limestones that are close to the sedimentary-igneous contact.

Most of the mineral occurrences and deposits discovered to date have surface expressions at outcrop. Soil geochemistry is likely to play an increasingly important role in finding new deposits in the Pánfilo Natera district because a large proportion of the district is covered by alluvial infill and soil overburden.

7.4 Geology of the Bilbao Property

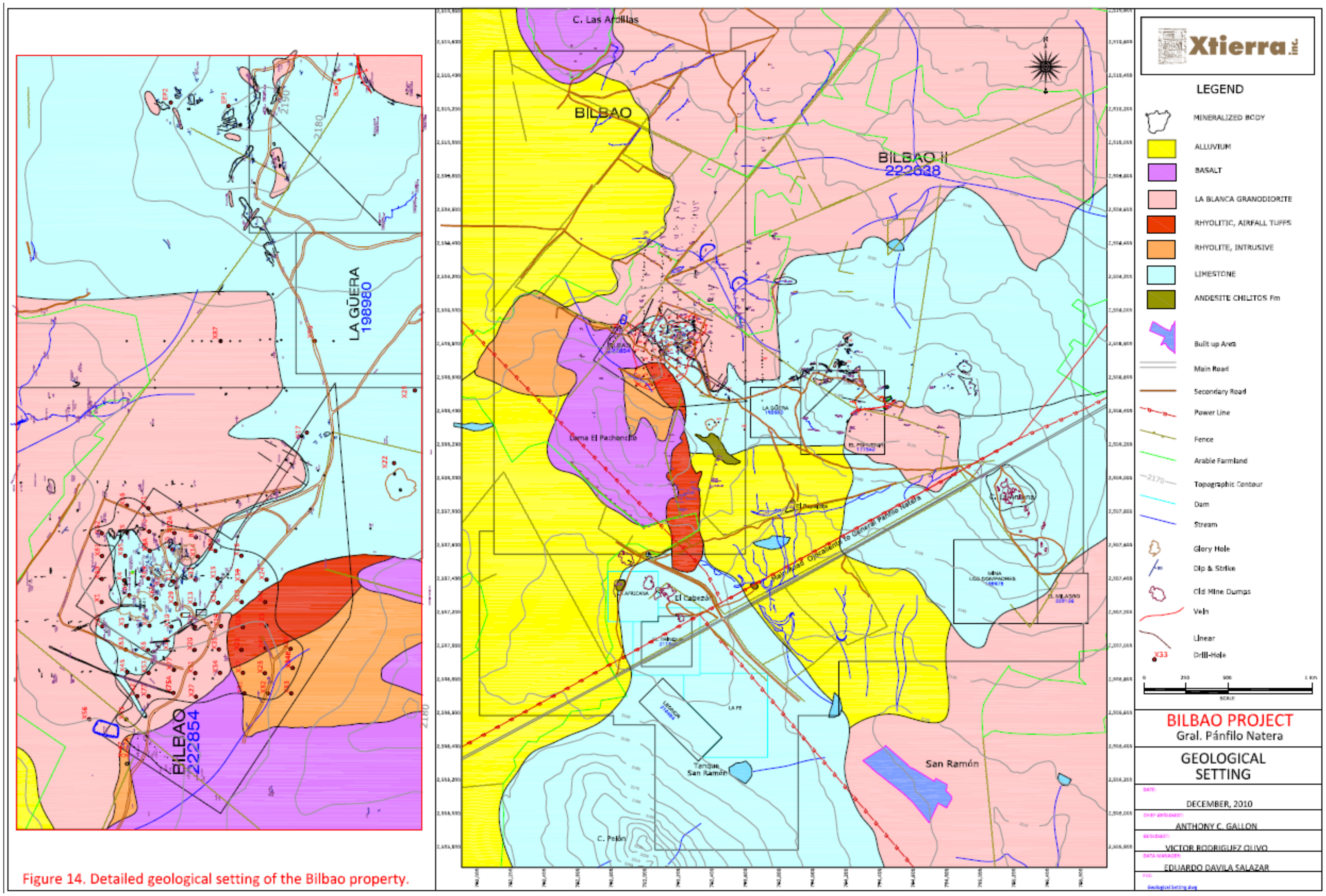
Figure 7-4 shows the geological setting to the Bilbao property. The oldest rocks within the Bilbao claims are Chilitos volcanics which are variously interpreted as of Late Jurassic-Early Cretaceous age. These volcanics occur in the lower ground in the south-east sector of the Bilbao I claim, near the old El Cabezón mine, and are only exposed in pits and road aggregate quarries. The rocks are mainly green chloritised andesites. No pillow lavas have yet been observed in these rocks within the Bilbao claims. This same rock sequence hosts the San Nicolás volcanogenic massive sulphide deposit some 18 km to the south-east. Presently there is no indication that the Chilitos Formation on the Bilbao claims hosts similar VMS mineralization although that possibility remains since most of the sequence is obscured by younger rocks and soil cover

Overlying the Chilitos volcanics is a sequence of Aptian-Albian limestones within which the Bilbao mineralization is hosted. These Lower Cretaceous limestones were deposited roughly 110 million years ago. The limestones are, for the most part, relative pure carbonates although there are interbedded sandy limestones within the sequence on the La Güera claim area, and carbonaceous limestones on the El Porvenir and Bilbao II claims. A notable sedimentary feature of the limestone sequence is the occurrence of rip-up clasts of bedded limestone embedded within an overall matrix of fine grained limestone which indicates that the sequence was deposited in a palaeo-environment that was periodically affected by strong current action. Where the limestones have been mineralized they are altered to brown iron oxides. These oxides crop out on the small hillock on the southern rim of Glory Hole 1.

After deposition of Lower Cretaceous limestones there was a long period of non deposition, since Upper Cretaceous sequences are absent, before Oligocene rhyolitic airborne tuffs were deposited in this erosion surface. These tuffs where then intruded by a series of Tertiary rhyolitic plugs which form prominent hills (e.g. Cerro El Morro) in the general vicinity.

Of paramount importance, as concerns emplacement of the Bilbao mineralization is the intrusion of the La Blanca granite which probably intruded the older rock sequences in late Oligocene (30Ma ago) times. There are no

Figure 7-4 Bilbao Property Geology



specific radiometric ages for the La Blanca granite intrusive in the literature so the age its intrusion may be Upper Cretaceous since the nearest dated granite at Zacatón has a K/Ar date of 77Ma according to Solé et al (2007). The composition of the La Blanca ―granite‖ suggests that biotite-hornblende granodiorite would be a closer petrographic description.

Residual metal-bearing fluids, concentrated during the final crystallization phase of the granite caused mineralization of the limestone. The resultant Bilbao deposit developed as a mineralized skarn body which selectively replaces the more porous limestone bedding of the roof pendant.

Typical contact metamorphic minerals such as garnets and diopsidic-pyroxenes are developed in the skarn zones and marbleized limestone also occurs. Where the limestones were originally siliceous, wollastonite and other meta-silicate minerals are developed as can be seen in the La Güera and El Porvenir claims some 600m to the east of the Bilbao shaft. The wollastonite there has been sporadically worked for that mineral on the El Porvenir claim.

The Pb-Zn-Cu-Ag replacement mineralization preferentially follows the more porous horizons within the limestone sequence as stratabound lenses as can be clearly seen in Figure 7-5 of Glory Hole 2 and on occasion transgresses and replaces the limestone as bodies of irregular morphology.

In addition to the main stratabound mineralization there are cross-cutting mantos as well as a pervasive skarnoid envelope along the main batholithic contact and its larger granitic sills.

Mineralization also occurs within the outer parts of the granite itself as an endoskarn which is sporadically developed and is accompanied by thin silver-rich veinlets.

The original sulphide mineralization was subsequently altered by percolating rainwater so that at surface the Bilbao prospect is typified by brown iron oxides.

Following intrusion of the granite an extensional tectonic regime ensued with development of horsts and grabens bounded by normal faults. Regionally the main grabens have a NNE-SSWtrend. The fault-bound grabens resulted in the development of a type of Basin and Range topography not dissimilar to that found in the western part of the USA. Bilbao occupies a horst between two such grabens

A late stage silver-rich vein mineralization, emplaced in the fractures and typified by the El Cabezón mineralization, is likely to have been caused by this extensional tectonic phase. In like manner the mineralization recently discovered at the Ardillas prospect on the Bilbao II claim follows, and forms a replacement off, WNW-SSE faults which probably developed during the same tectonic event.

The rifts within the grabens were then filled with Pliocene (5Ma) piedmont fans of conglomerate and sandstone derived from the surrounding horsts. Indeed the Bilbao oxide body itself was partially eroded during this period and fragments of the oxidic ore were incorporated in the arenaceous infill sediments north-west of the outcrop of the Bilbao mineralization. This is the reason why there is a strong ―false‖ Pb-Zn soil geochemical anomaly over the sub-outcrop of these infill sediments in the flat farmlands in that area.

The lower parts of the main graben, some five kilometres to the east of the Bilbao prospect, are occupied by a series of alkaline lakes (El Sapo, El Tule, Las Pilas and El Salado) some of which have been sporadically worked for salt. As mentioned elsewhere in this report, these are also enriched in lithium and boron as soluble salts occurring with the halite brines. Report #174 entitled “A preliminary assessment of water resources available to the Bilbao Project” gives details on these lakes; the waters of which are considered too saline to be used as a source for the processing plant envisaged at Bilbao.

Extrusive onto these older sequences is a very late stage Pleistocene amygdaloidal basalt flow emanating from the vicinity of Loma El Pachoncito. These basaltic lavas overlie the extension of the Bilbao mineralization in the southwestern sector of the deposit.

The youngest rocks are recent alluvial sediments, soils and calcrete/caliche cover which are particularly strongly developed in the southern and western parts of the Bilbao prospect

Figure 7-5 Glory Hole 2 Illustrating Moderately Dipping Massive Limestone Beds

Rock exposure in the Bilbao prospect area is dominated by the mineralised oxide outcrops surrounding Glory Hole 1. The sulphide body does not crop out and is a separate entity to those represented by the oxides in the glory holes. The Company and others have surveyed the bedrock at Bilbao, which is mostly overlain by caliche, and so outcrop is sparse and extensive trenching, or short-hole drilling, would be required to significantly improve the detail of the surface mapping.

7.5 Geologic Setting and Mineralization at Bilbao

The Bilbao mineralization is developed within the contact zone of the La Blanca granodiorite batholith and its irregular sill-like apophyses, where it occurs as a sulphide replacement of a roof pendant of marbleized limestone, commonly in conjunction with skarn minerals such as hedenbergite (pyroxene) and other calc-silicates. Subsidiary sulphide mineralization is developed as an endoskarn within the granodiorite.

Sulphide and oxide mineralization occurs as stacked tabular lenses up to 50 metres thick that preferentially follow the bedding of the limestone or are transgressive to the bedding close to the granite contact where they develop as a contact skarn.

Drilling has intersected at least seven strongly mineralized bodies representing oxide, mixed and sulphide mineralization. In addition there are numerous thinner intersections of lower grade material representing partial sulphide replacements and their oxidized equivalent.

The mineralised horizons have been defined over an area of approximately 400m by 300m (see fig 12) and are best developed along a NE trending axis centred on hole X13 in which massive and semi-massive sulphide replacement is developed over a thickness of 40 metres. This trend may reflect a cross-fold at right-angles to the primary NW-SE structure. The intensity and thickness of the various mineralized horizons shows a general tendency to diminish laterally from this trend, thus defining a ―Core Zone‖ that includes the majority of the mineral resource. The oxide-sulphide transition zone ranges between 130 metres and 150 metres in depth depending on the location within the deposit. Lenses occurring above the transition are predominantly oxidized whereas those below it are predominantly sulphide.

7.6 Metal Distribution

Silver grades are generally closely correlated with sulphide replacement mineralization and particularly with lead grades. However other types of silver mineralization occur in the SW & SE sectors of the deposit including massive sulphide, veins in the granite endoskarn and as a breccia body, (probably a fault breccia) in drillholes X85 & CG4. High grade silver veins and veinlets occur in Hole X26 which intersected stringers of native silver with stromeyerite from 381 to 385 metres, close to the granite contact. Silver grades here reflect the observed mineralization with several samples exceeding 1kg/t Ag. The stringers are at a high angle to the core so the true thickness is exaggerated; nevertheless they are of potential interest since several of the drill holes in the western sector have also intersected similar vein mineralization. Intersections of these vein occurrences have not been taken into consideration in the resource estimate since their geometry and distribution are not adequately known. The distribution of these higher-grade silver zones is the subject of on-going study in the expectation that vein-hosted silver mineralization may be found in the future.

Gold and tin assays have been performed for only a minority of drill holes and in view of this limited assay data neither metal has been included in the resource estimate. Gold values exceeding 1g/t Au were recorded in at least ten 1m intersections in six separate drill-holes.

Table 7-1 Significant Gold Intersections

DDH	Fom	To	Width	Au g/t
B6A	233.15	239.25	6.10	1.32
including	233.15	234.70	1.55	2.70
X19	247.00	263.00	16.00	0.31
X30	74.00	80.00	6.00	0.53
including	79.00	80.00	1.00	2.02
X32	191.00	194.00	3.00	0.82
X32	213.00	218.00	5.00	0.73
X33	171.00	260.00	89.00	0.22

The distribution of the higher gold grades is sporadic but there is a correlation between elevated gold and higher copper values.

Tin, occurs as a component of the Bilbao ore as discrete grains of cassiterite. Tin grades are generally less than 500ppm, but there is a general enrichment of tin in most of the higher grade intersections, which commonly report values of 0.1% to 0.2% Sn. The more significant tin intersections are listed Table 7-2 in below.

Table 7-2 Significant Tin Intersections

DDH	Fom	To	Width	Sn %
X29	86.00	114.00	28.00	0.51
including	97.00	100.00	3.00	2.57
X30	60.00	85.00	25.00	0.20
X32	183.00	200.00	17.00	0.62
X33	49.00	63.00	14.00	0.24
X33	161.00	169.40	8.40	0.29

Neither gold nor tin has been considered in the resource estimation but there is a potential that both may be recovered as by-products in any processing scheme.

Other metals that might be of commercial interest include cadmium, which is a common associate of zinc minerals in some base metal skarn deposits, iron oxide and tungsten.

7.7 Mineralization Types

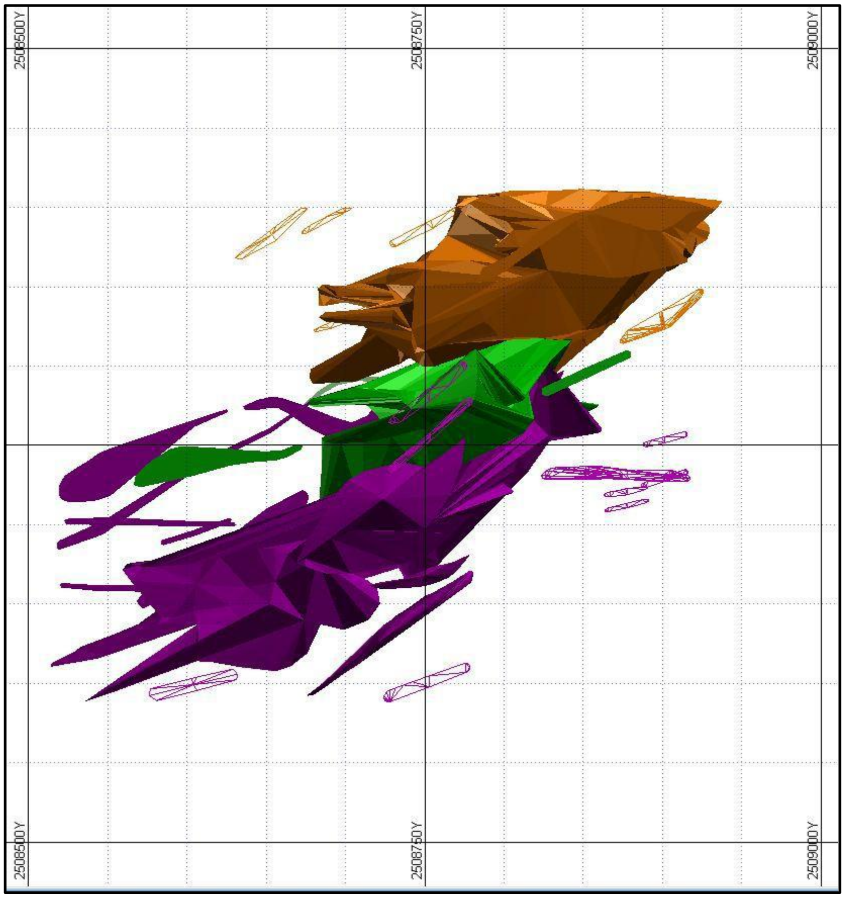
The Bilbao mineralization is classic contact metamorphic skarn type developed by irregular replacement of carbonate. However, there are marked constraints on this irregularity in that the mineralization selectively replaces certain horizons in the stratigraphy of the limestone and so assumes lenticular form. Furthermore, there is a strong tendency for mineralization to occur at the granitic contact. Because the mineralization is related to fluids emanating from the (hot) granite this impinges on the mineralogical zoning away from the heat source and results in higher temperature minerals being deposited closer to the main granite contact. This has implications on primary mineralogy, especially for zinc, which in this case appears to have formed several mineral species in congruence with the above zonation. Thus at the higher temperatures prevailing close to the granite franklinite, willemite and hemimorphite have developed whereas sphalerite appears to have formed further away from the contact.

The recognition and delineation of oxide and sulphide mineralization is of importance in relation to their metallurgical recoveries and to the overall economics of any future mining operation at Bilbao. It is therefore important to classify and segregate resources according to mineralization type.

Discrimination of sulphide, mixed and oxide mineralization for the purposes of resource estimation has been carried out principally by examination of descriptions of the narrative logs. Mineral zones have then been classified as sulphide, oxide and mixed according to the predominance of ore types within the zone. Within the central core of the deposit sulphide and oxide zones are thus depicted on section as separated by a mixed zone up to 60 metres thick.

Detailed examination of drill core and logs indicates that in reality the distribution of ore types is considerably more complex than this simple pattern and it appears that oxidation has preferentially affected the thinner zones and the margins of the thicker zones, leaving the deeper and thicker zones unaffected. Insufficient data is available to depict the distribution of ore types with accuracy.

Figure 7-6 Bilbao: Vertical Distribution of Ore Types
Brown = Oxide, Purple = Sulphide, Green = Mixed



7.7.1 Sulphide Mineralization

The principal sulphide minerals present in the primary mineralization are pyrite, sphalerite, galena and chalcopyrite. Examination of these sulphides in drill core indicates that they occur mostly as separate grains frequently from 1-5mm in size, and can be easily recognized. The sphalerite occurs as the iron-rich black jack variety, which is relatively easy to recognize in comparison to pale or fine-grained varieties that have not been recognized at Bilbao. Minor or rare sulphide minerals recognized in the Bilbao core include pyrrhotite, marcasite, arsenopyrite, bismuthinite, bornite, molybdenite and guanajuatite.

Accompanying these sulphides are the primary oxides franklinite and minor cassiterite. Furthermore willemite, the zinc silicate, (and rare larsenite its Pb analogue) complete the skarnoid mineralogical suite. The gangue hosting these sulphides comprises limestone, or more usually calc-silicate rich marbleized limestones.

The presence of minor tin and tungsten values indicates a granitic affinity and an association with other Mexican manto-type deposits.

Sulphide mineralization has been recorded at a minimum depth of 58 metres and to a maximum depth of 298 metres below surface.

Figure 7-7 Drill Core Illustrating Oxide-Sulphide Contact
(Oxide - reddish core in top half of core box Sulphide - bottom half of core box)

7.7.2 Oxide Mineralization

Most mineralization within 100 metres of surface has been completely oxidized by reaction with meteoric waters. The oxide part of the body is directly derived from weathering of the primary sulphides, and because of this the grades are more or less the same as those occurring in the sulphides

Historically, the oxide material was mined in the open pits and underground with zinc and copper recovered by leaching with dilute sulphuric acid or directly shipped to the plant to recover all metals. (Note: historical production was shipped directly to a smelter in Texas and not treated on site).

The oxidized fraction of the deposit has an altered mineralogy compared to that in the primary mineralization in that silicates, sulphates, phosphates and carbonates predominate. The bulk of the gangue in the oxide ore comprises a mixture of quartz, iron-oxides/hydroxides, carbonates, phosphates, sulphates, arsenates and silicates. Through scanning electron microscopy (SEM) work undertaken by the University of San Luis Potosí and mineralogy undertaken by Grammatikopoulos et al (2008) of SGS, the mineralogical composition of the Bilbao oxide is much better known than when originally studied by Kilborn. The main points emerging from these studies of the mineralogy are listed below:

1. The composition of the whole is made up chiefly of iron oxides and quartz with about 7% metals principally lead, zinc and copper together with roughly 3 ounces of silver.
2. The mineralogy of the gangue is governed by two factors. Firstly, it is derived from a primary contact skarn and secondly it formed from oxidation of a primary pyrite-rich sulphide body; the whole being weathered and altered by secondary acidic reactions generated by decomposition of pyrite into dilute sulfuric acid solutions by meteoric water. This has resulted in resistate minerals such as franklinite, cassiterite, willemite, pyroxenes, and quartz being intermixed with secondary alteration products such as oxides, hydroxides, carbonates, sulfates, phosphates, arsenates and clays. Complicating the issue is

that, in part, the oxidation is incomplete so that some remnant sulphides persist in the oxide part; this is especially so with respect to sphalerite.

3. The gangue minerals identified in the mineralogical study are:
   i. Fe-oxides: hematite, goethite Clays: nacrite, kaolinite.
   ii. Sulphates: anhydrite, barite, gypsum, basaluminite,
   iii. Carbonates: calcite
   iv. Arsenates: ogdensburgite
   v. Silicates: pyroxene, quartz and aerinite
4. Lead occurs in the oxide, predominantly as the unusual mineral hedyphane Ca2Pb3(AsO4)3Cl. Formerly this was considered a type of mimetite - the yellow lead mineral found on the dumps at Bilbao, but is now classified as an end-member of the apatite group along with phosphohedyphane)
5. Minor amounts of galena, lead oxides and lead/ manganese-oxides are also found in the oxide.
6. Trace amounts of other lead minerals such as phosphohedyphane Ca2(Pb,Ca)3(PO4)3Cl and corkite PbFe3+(PO4)(SO4)(OH)6 also occur.
7. Zinc in the oxide fraction occurs as sphalerite, willemite and hemimorphite.

Oxide mineralization has been recorded to a maximum depth of 215 metres from surface (in hole X20). Most mineralization above 100 metres depth (2050m elevation) is completely oxidised, with the degree of oxidation diminishing with depth so that mineralization below 150 metres (2000m elevation) is largely unaffected by oxidation. The distribution of oxidation appears to be controlled by the permeability of the rocks, and is controlled largely by faults and other fractures. Oxidation has preferentially affected the outer surfaces of the sulphide lenses, so that the thinner lenses are completely oxidised, whereas thicker lenses retain a sulphide core surrounded on the margins by an oxide shell.

To the west of the main Bilbao body thick tuffs and basalt flows fill a Tertiary palaeovalley below which oxidation extends to greater depths which may be related to the former palaeo-topography rather than fault controlled.

The most obvious criterion for discriminating oxide and sulphide mineralization types is the abundance of iron oxide and absence of sulphides. Oxide mineralization is usually deep orange to reddish-purple, but discrimination of oxide ore from sulphide on the basis of colour alone can be misleading. In some cases remnant sulphides persist in the oxide and zinc can remain as resistate primary willemite in the oxide zone.

The predominant iron oxide occurs as hematite and has a significant manganese component with manganese “trees” common on flat surfaces. Geochemically anomalous manganese levels (>1000ppm Mn) occur around the mineralized core and extend throughout the Property.

The mineralogy of the oxide fraction of the Bilbao deposit is known in its general aspects but has not been studied in detail. A mineralogical study on specifically chosen oxide samples will be necessary to better describe the textures, grain-size, aggregation, associations and particularly the mineral species involved in the economic minerals of lead, copper, zinc/cadmium, silver and tin. Moreover a study of the arsenic and mercury contents and problems involved in acid liquors emanating from wet jarosite waste would be advisable to avoid any potential environmental challenges with the tailings.

7.7.3 Mixed Mineralization

The interface between oxide and sulphide mineralization is not always sharp and easily defined. Some intersections comprise many metres of partly oxidized material within which sulphide remnants occurring as lenses or patches from a few centimetres up to several metres wide may alternate with oxidized or partly oxidized material to form mixed mineralization.

Mixed mineralization has been recorded at a minimum depth of 28 metres and to a maximum depth of 229 metres below surface

7.7.4 Petrographic and Mineralogical Studies

Petrographic and mineralogical studies were carried out on a series of samples provided by the Company to Dr. Maria del Carmen Ojeda of the Instituto de Metalurgia at the Autonomous University of San Luis Potosí.

The principal oxide minerals identified were smithsonite, anglesite and limonite. Other secondary minerals which have been identified and are important components include pyromorphite, $Pb5(PO4)3Cl$, and mimetite, $Pb5Cl(AsO4)3$. Zinc sulphate, goslarite, is very soluble whereas lead sulphate (anglesite) is insoluble; as a consequence the zinc minerals in the surface oxide are absorbed in complex iron-manganese-zinc oxides or occur as carbonates such as smithsonite, $ZnCO3$.

The principal minerals that are known to occur within the ore at Bilbao are listed in Table 7-3 below.

Table 7-3 Bilbao Deposit: List of Recorded Minerals

Metal	Elements	Sulphides	Oxides	Silicates	Sulphates	Phosphates	Arsenate	Carbonate
Copper	*Native copper*	**Chalcopyrite** Chalcocite Bornite Emplectite		Chrysocolla		pseudo-malachite		Malachite
Lead		**Galena**	Minium	Larsenite	Anglesite	Pyro-morphite Phospho-hedyphane Corkite	Hedyphane Ogdens-burgite	Cerussite
Zinc		**Sphalerite**	Franklinite Zincite	Willemite Hemimorphite				**Smithsonite**
Silver	Native Silver	*Stromeyerite*						
Gold	Native Gold							
Iron		Pyrite Marcasite	**Hematite** **Goethite** **Limonite** Magnetite		**Jarosite**			
Tin		Stannite	Cassiterite					
Other		**Arsenopyrite** *Bismuthinite* *Guanajuatite*	Silica Ilmenite Wad *Uraninite*	Ca Garnets K feldspar Clays;Nacrite and Kaolinite Diopsidic Pyroxenes Hedenbergite	Barite Gypsum Anhdrite Basaluminite	Apatite Xenotime		Dolomite Calcite Rhodochrosite

note: Major components in bold, minerals in italics are rare

8. Deposit Types

The Bilbao deposit is characterised by skarn minerals (eg. hedenbergite (pyroxene), wollastonite and other calc-silicates) and can be classified as a polymetallic skarn deposit.

Base metal skarn-type deposits commonly form a continuum with Carbonate Replacement Deposits (‗CRD') that are generally developed distally from intrusions. In Mexico such deposits are known as manto type deposits when the mineralization forms a coherent massive sulphide sheet.

Polymetallic skarn deposits are typically developed within carbonate rocks close to the contact with intrusive plutons, which are frequently granodiorite or leucogranite in composition. Typical sulphide minerals include sphalerite ± galena ± pyrrhotite ± pyrite ± magnetite ± arsenopyrite ± chalcopyrite ± bornite. Other trace minerals reported include scheelite, bismuthinite, stannite, cassiterite, tetrahedrite, molybdenite, fluorite, and native gold. Proximal skarns tend to be richer in Cu and W, whereas distal skarns contain higher amounts of Pb, Ag and Mn. Other examples of skarn deposits in Mexico are San Antonio, Santa Eulalia and Naica. Large, well-known examples of this deposit type are also found in the western United States at Leadville, Colorado and in Utah at Park City and Tintic.

According to British Columbia Geological Survey Mineral Deposit Models, *"Pb-Zn skarns tend to be small (<3Mt) but can reach 45Mt, grading up to 15 % Zn, 10 % Pb and > 150 g/t Ag with substantial Cd. Cu grades are generally < 0.2 %. Some deposits (e.g. Naica (Mexico) and Falun (Sweden)) contain Au. CRD deposits account for roughly 4 billion ounces or 40% of the 10 billion total silver ounces produced in Mexico. They are second only to Mexico's epithermal veins in historical silver production. As sources of base metals, manto deposits are overshadowed on a world scale by the giant syngenetic classes such as sedimentary exhalative and volcanogenic massive sulphides. However, because of their high precious metal contents, they provide exciting targets for small producers."*

Polymetallic skarns are commonly oxidised at surface. During weathering, the original primary ore minerals, sphalerite (ZnS) galena (PbS) and pyrite (FeS2) are oxidised generating an acid metal-saturated solution. Buffering by the limestone results in decalcification of the host rocks and the precipitation of metals as zinc-carbonate and lead carbonate mineral species. Iron, being more mobile in slightly acidic to neutral solutions, is more widely distributed as goethite, jarosite and hematite. In general, it is thought that the oxidation process causes grade dilution, especially for zinc. Examples of oxidized manto-type zinc-lead deposits include Torlon, Guatemala.

9. Exploration

Exploration on the Bilbao claims comprised conventional geological, geophysical and drilling methodologies to investigate the resource on surface and in depth.

9.1 Geology and Prospecting

Surface mapping has been completed over almost the entire Property, including areas away from the main mineralization. A full understanding of the underlying geology is hindered by extensive calcrete development at surface. Notwithstanding, the outcrop is sufficient to allow an appreciation of the overall geological setting.

9.2 Geophysics

An airborne survey (magnetic/radiometric surveys) was carried out by the government in 1986 unfortunately the survey line spacing makes it suitable only as reconnaissance tool.

During 2006, the Company conducted limited induced polarization (IP) and ground magnetic (MAG) surveys in the southern part of the Property. The objective of these surveys was to determine whether there were blind targets south of the currently known mineralization.

Since the mineralization at Bilbao has a significant magnetic component it had been anticipated that the magnetic survey over the Bilbao deposit would have a diagnostic fingerprint that could be used to locate additional mineralization. The effectiveness of the survey was impaired by the strong magnetic response of the overlying basalt flows which masked any signature from the underlying mineralization apart from a general NW-SE magnetic trend that probably parallels that of the mineralizing fault structures. The magnetic survey was therefore of limited use in guiding any of the scout drilling done in search of extensions to the core sector of the mineralization.

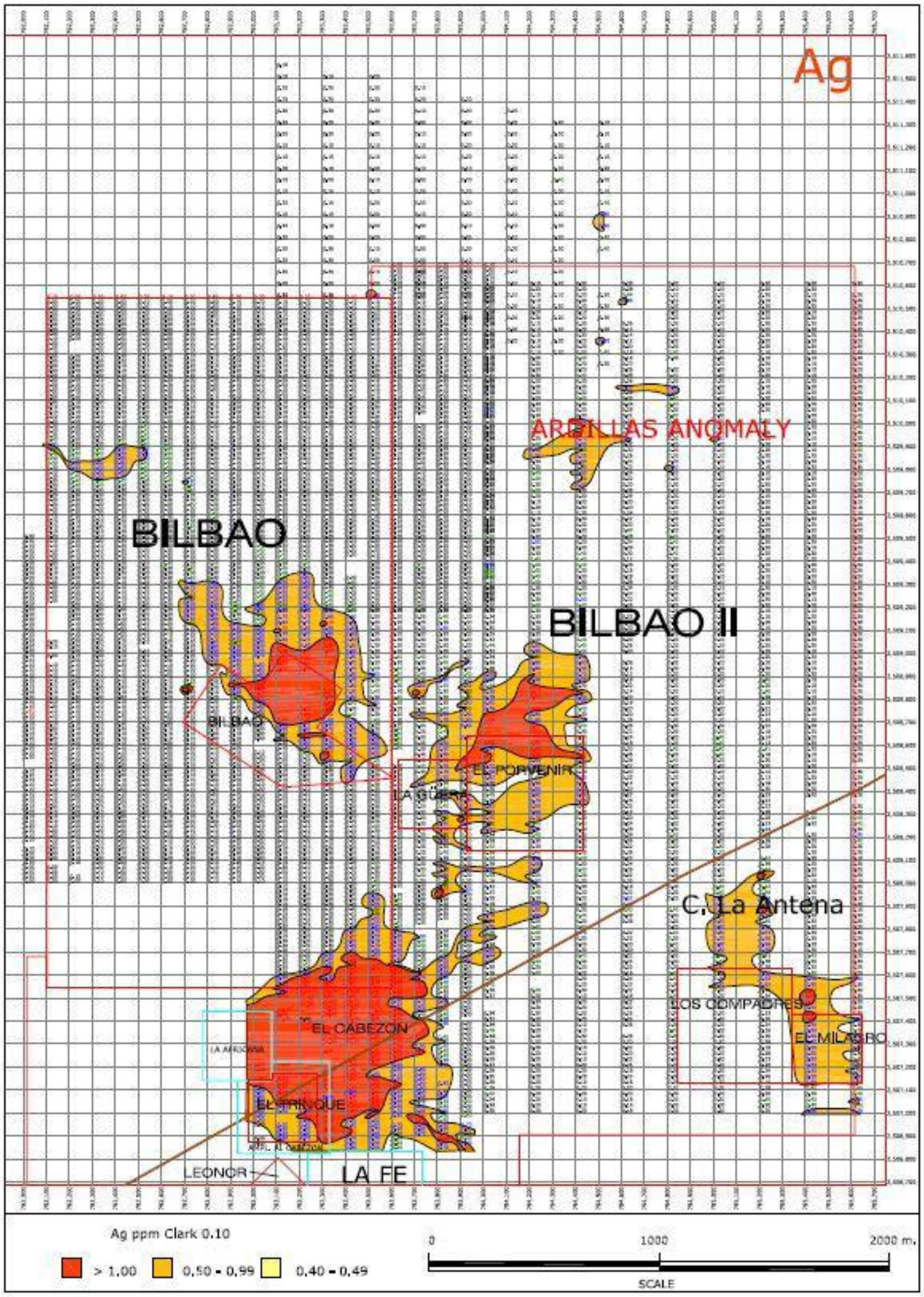
9.3 Geochemistry

In 2001, Phelps Dodge completed a soil geochemical survey over part of the project area on a 200m by 50m grid. The results of this survey outlined a large Cu, Pb and Zn geochemical anomaly over the historic Bilbao workings.

Xtierra have since undertaken a more comprehensive soil sampling program (see Figure 9-1) which has included sampling over all the component claim blocks in the Bilbao area. In the southern part of the Property a large geochemical anomaly revealed a strongly anomalous contamination plume emanating from the old El Cabezón mine and mill complex but the survey did not detect other soil anomalies which might reflect bedrock mineralization. Notwithstanding soil sampling within the Bilbao II claims has shown two distinct soil geochemical anomalies namely:

- Ardillas, (794.400/2509.950)—a Pb-Ag anomaly with visible hedyphane mineralization occurring as dissemination within granite adjacent to a fault complex trending WNW-ESE (ie parallel with the San José mineralized trend). This has not so far been drilled but is scheduled for examination at a future date,
- El Porvenir, (794.200/2508.800)--a weak Cu-Au soil geochemical anomaly at the contact of granites and limestones to the east of the Bilbao mineralization. Two drill holes (EP1 & EP2) tested this anomaly and found sparse disseminations of pyrite and chalcopyrite suggesting that the La Blanca granite has porphyritic affinities.

Figure 9-1 Bilbao Soil AG Geochemical Anomolies

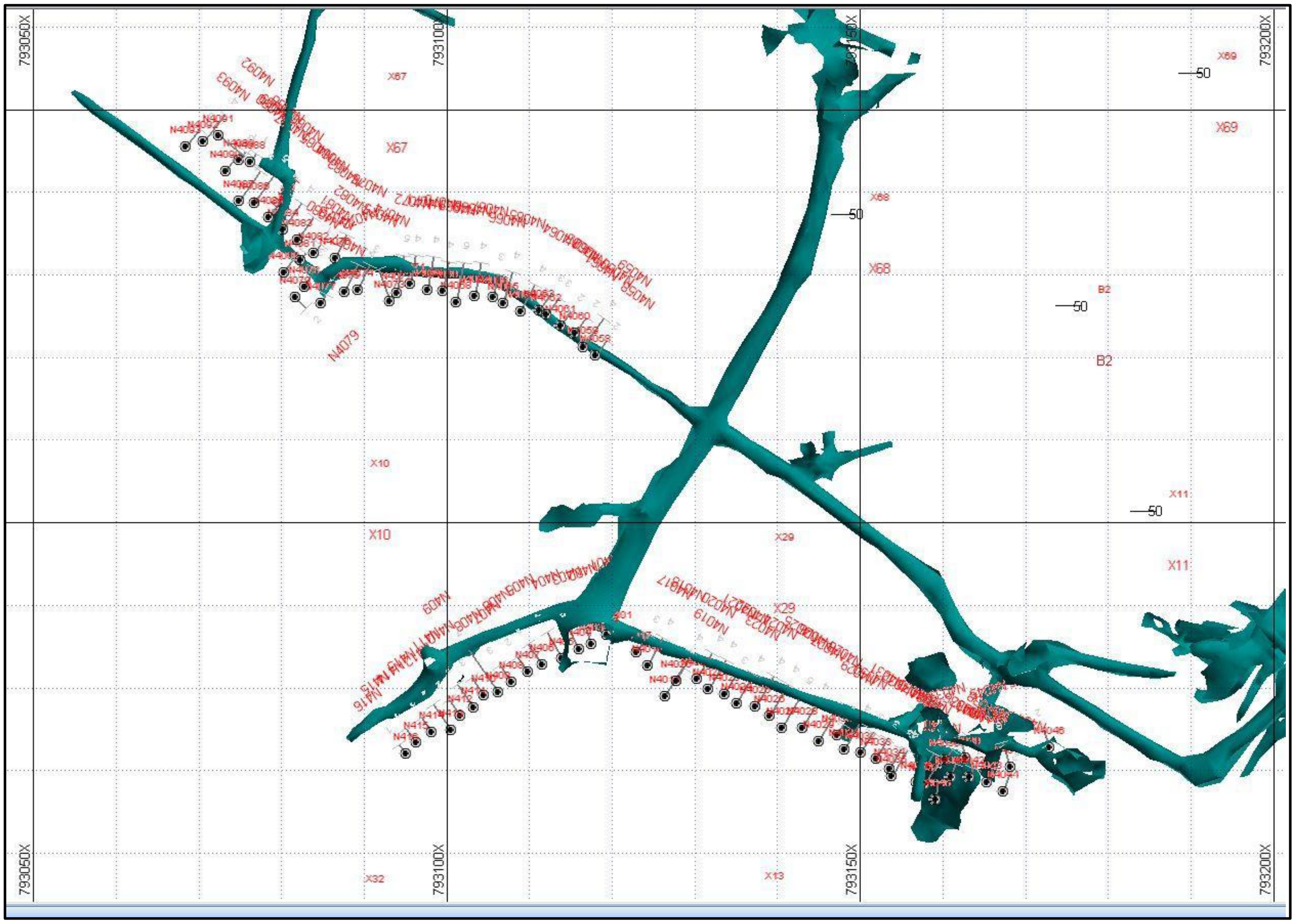


9.4 Surface and Underground Sampling

Comprehensive channel sampling of the oxide zone has been completed for the glory holes and some sampling has been conducted on both underground levels. A total of 324 channel samples were collected from three glory holes (127 samples) and underground drifts within oxide ore in the 40m level at 2112m elevation (197 samples). Sample length varied from 0.5 to 3.0 metres, averaging 1.40m. Underground surveying, sampling and geological mapping have been undertaken both on the 40 and 76m levels.

Figure 9-2 below shows the location of these samples in plan.

Figure 9-2 Plan Showing Bilbao Channel Sample Locations on 40m Level



9.5 Topographic and Underground Surveys

Topographic mapping was undertaken at the same time as geological and geophysical surveys were being done on the Property. In particular, altitudes were taken on all sampling lines, farm roads and all cultural features, including fences, were mapped by GPS. This enabled a detailed working topographic map to be made on which geological features were plotted. Conventional land surveying was undertaken in the area of the main shaft which included spotting all drill-hole collar elevations, this to enable inter-drill hole correlation.

A digital terrain model has been prepared from available high definition satellite imagery. The accessible underground workings have been surveyed by laser scanning and a digital terrain model (DTM) has been generated.

10. Drilling

10.1 Introduction

Since 2006, Xtierra has drilled 113 diamond drill-holes in the Bilbao deposit. This total includes five geotechnical holes and two metallurgical holes. One hundred and six holes were drilled to define resources and provide some condemnation information.

All of the drill-holes are diamond NQ-HQ core holes with most (104) being vertical. They have been drilled by the company through six campaigns since 2006 completing a general grid of 50 m by 50 m and a tighter drilling grid of 35 m by 35 m in the high grade core (Figure 10-1). After the last 2010 resource estimation, 26 infill holes were drilled to complete the tight grid in the central high-grade zone. The drilled zone extends over an area of 530m along north-south axis and 580m along east-west axis.

10.2 Sampling and Logging

Xtierra has collected the geological and assay data from the drill program and compiled it in an MS Access database and plotted it on a series of N-S sections.

Parker's report (April 2010) describes the core handling, logging and sampling procedures for the Xtierra campaigns. Summarizing, Xtierra has stored the cores in boxes holding 3 m of core and appropriately labeled. All the logging and sample descriptions were recorded on paper forms first, next transferred to digital forms on Microsoft EXCEL spreadsheets and then, exported to the MS Access database. Xtierra geologists logged RQD, recovery, and geology including lithology, alteration and mineralization (Figure 10-2). Geology was logged in a description column and in RPM's opinion it should be logged in separated columns for lithology, type of mineralization, oxide minerals and abundance of oxide minerals, sulphide minerals, and abundance of sulphide minerals.

The lithology table contains the codes for six sedimentary or volcanic rock units plus granite, fault, vein and non-recovery (Table 10-1). Mineral zone (oxidation state) information is not included in any table.

Xtierra reduced the amount of lithology codes in the last database taking off the codes "sand" and "rhyolite". RPM recommends clarifying the equivalence of these codes in the last database version.

Bedded rocks were modeled in two groups, upper and lower unit (Table 10-1); exoskarn is wholly included into the lower unit. Endoskarn and granite were modeled together.

Table 10-1 Lithology Codes and Geological Units

Geological Model	Codes in 2013 DB
Upper Sedimentary Unit	Alluvium_Soil
	Basalt
	Lithic_Arenite
	Piedmont_Breccia
Lower Sedimentary Unit	Limestone
	Exoskarn
Intrusives	Granite
	Endoskarn
Others	Fault
	Vein
	No_Recovery

Figure 10-1 Drill Hole Location

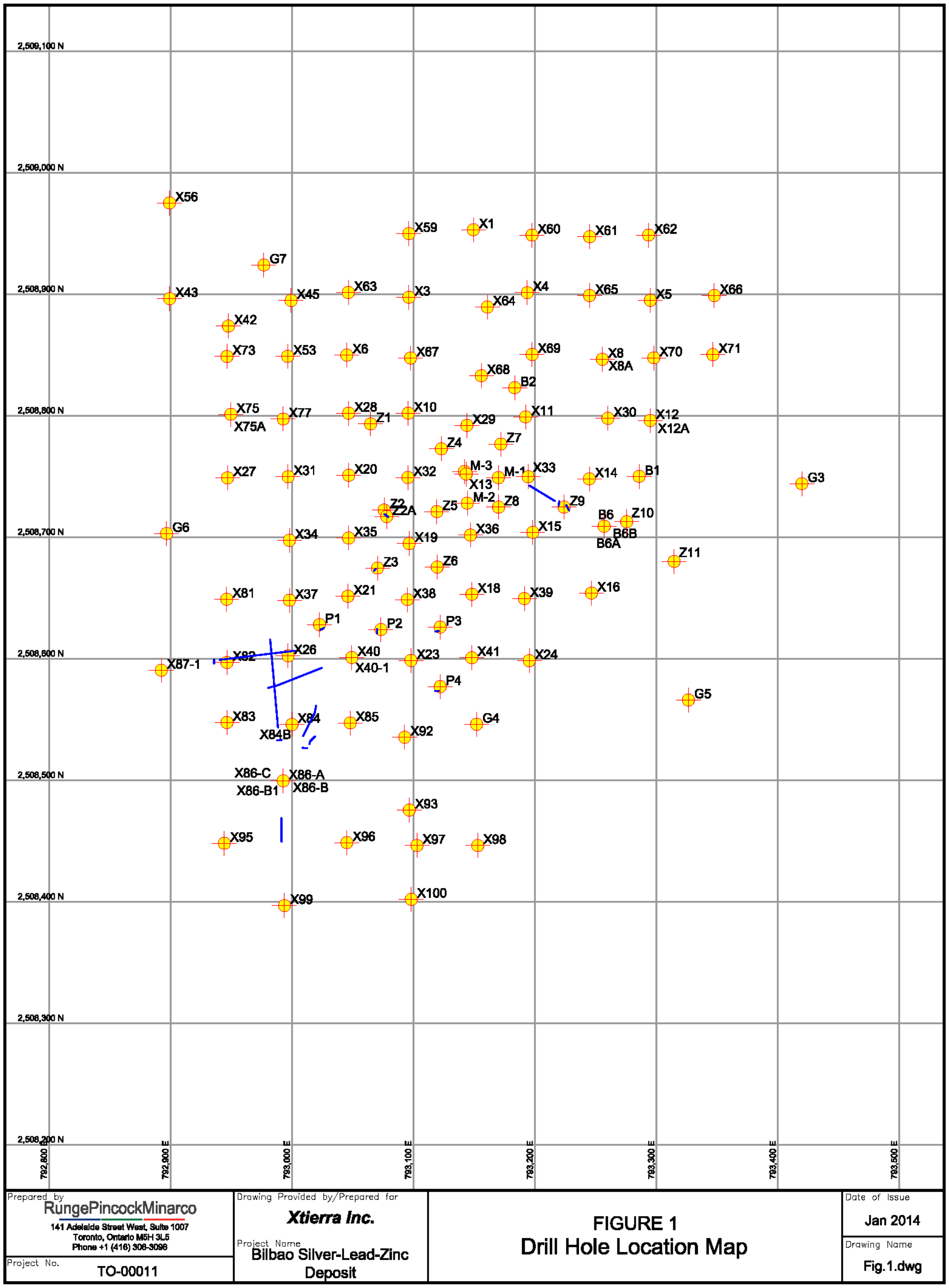


Figure 10-2 Log Sheet Example

Borehole No	X-45	CORE SIZE	CASING	FROM	TO	REPRESENTING m	RECOVERED m	Core Recovery %	RQD from	RQD to	Sum of core lengths >0.1m	RQD %	BOX NUMBER	Depth from	Depth to	Description of Core
				FROM	TO	REP	REC	C/REC	FROM	TO	SUM	RQD	BOX #	From	To	Description
		NQ	HQ	0.00	7.60	7.60	0.15	2%	0.00	2.00	0.00	0%	1	0.00	10.65	Granito muy delesnable y fracturado esteril.
LOCATION	BILBAO			7.60	10.65	3.05	2.15	70%	2.00	4.00	0.00	0%	1			
AREA				10.65	13.70	3.05	2.60	85%	4.00	6.00	0.10	5%	1	10.65	11.70	Caliza color rojiza ,con una ligera oxidacion con oxidos de fierro , vetillas
13K	792995.69			13.70	16.75	3.05	2.00	66%	6.00	8.00	0.25	13%	1			de calcita y una estratificacion de 45ª respecto al eje del barreno.
UTM	2508899.76			16.75	19.80	3.05	1.40	46%	8.00	10.00	0.55	28%	1			
COLLAR Elevation m	2145.69			19.80	22.85	3.05	2.00	66%	10.00	12.00	0.40	20%	1	11.70	26.70	Granito muy delesnable y fracturado esteril.
LINE	793000			22.85	25.90	3.05	2.90	95%	12.00	14.00	0.00	0%	1			
PEG	2508900			25.90	29.00	3.10	3.10	100%	14.00	16.00	0.00	0%	2			
ANGLE OF HOLE	90°			29.00	32.05	3.05	2.55	84%	16.00	18.00	0.00	0%	3			
DRILLED TOWARDS	Vertical			32.05	35.10	3.05	2.90	95%	18.00	20.00	0.00	0%	3			
Tropari Surveys				35.10	38.20	3.10	3.10	100%	20.00	22.00	0.00	0%	4			
EOH	264.50			38.20	41.25	3.05	3.05	100%	22.00	24.00	0.00	0%	4			
DATE STARTED	10-nov-08			41.25	44.35	3.10	3.10	100%	24.00	26.00	0.00	0%	5	26.70	27.75	Caliza oxidada , brechada con oxidos de fierro y vetillas de calcita.
DATE FINISHED	15-nov-08			44.35	47.40	3.05	2.95	97%	26.00	28.00	0.50	25%	5	27.75	39.65	Granito muy fracturado con baja oxidacion
DRILLER				47.40	50.45	3.05	3.05	100%	28.00	30.00	0.10	5%	6			
LOGGED BY	vro			50.45	53.20	2.75	3.05	111%	30.00	32.00	0.00	0%	7			
				53.20	54.20	1.00	0.75	75%	32.00	34.00	0.15	8%	7			
				54.20	56.60	2.40	1.20	50%	34.00	36.00	1.50	75%	8			
				56.60	59.65	3.05	0.40	13%	36.00	38.00	1.65	83%	9	39.65	40.90	Caliza fracturada y oxidada de color rojizo ,con vetillas de calcita y un
				59.65	62.75	3.10	1.35	44%	38.00	40.00	1.15	58%	10			50° de oxidos de fierro.
ZACATECAS ZAC.				62.75	65.80	3.05	0.75	25%	40.00	42.00	1.45	73%	10	40.90	42.80	Caliza de color gris claro masiva esteril y de textura sacaroide.
				65.80	68.85	3.05	1.80	59%	42.00	44.00	1.70	85%	11			
				68.85	71.90	3.05	0.20	7%	44.00	46.00	1.45	73%	12			
				71.90	74.95	3.05	0.30	10%	46.00	48.00	1.80	90%	12	42.80	51.20	Granito muy compacto con velillas de calcita de 1mm y con ligera oxidacio
				74.95	78.00	3.05	3.00	98%	48.00	50.00	1.40	70%	13			rellenando los fracturas.
				78.00	81.05	3.05	2.50	82%	50.00	52.00	1.55	78%	13			
				81.05	84.10	3.05	1.00	33%	52.00	54.00	1.25	63%	14			
				84.10	86.80	2.70	1.00	37%	54.00	56.00	1.80	90%	15	51.20	57.65	Granito muy compacto de color gris claro masivo esteril,con vetillas de cal
				86.80	90.20	3.40	0.60	18%	56.00	58.00	1.40	70%	16			cita de un mm. Y una estratificacion de 35° con respecto al eje de barreno
				90.20	93.30	2.70	2.70	100%	58.00	60.00	1.00	50%	16	57.65	60.50	zo- oxiCaliza muy fracturada y oxidada con diseminaciones de pirita del 1
				93.30	96.35	1.00	1.00	100%	60.00	62.00	0.80	40%	17	60.50	62.15	Granito muy fracturado y oxidado ,45° de contacto con la caliza contacto
				96.35	99.40	2.40	2.40	100%	62.00	64.00	0.90	45%	18			superior con respecto al eje del barreno.
				99.40	102.30	3.05	3.00	98%	64.00	66.00	1.30	65%	18	62.15	63.10	zona de oxido. Encajonado en calizas con una fuerte oxidacion con una
				102.30	105.60	3.10	3.10	100%	66.00	68.00	0.50	25%	19			fuerte silicificacion con oxidos de fierro ,oxido de plomo 45° de oxido
				105.60	108.70	3.05	3.05	100%	68.00	70.00	0.35	18%	19	63.10	65.60	Granito muy fracturado y alterado ,argilizado con moderada oxidacion .
				108.70	111.75	3.05	1.80	59%	70.00	72.00	1.40	70%	20	65.60	66.05	Zona de sulfuros masivos de plomo,zinc,cobre 70° de sulfuros.
				111.75	114.80	3.05	3.05	100%	72.00	74.00	0.15	8%	21	66.05	74.95	Granito de color verde ,muy cloritizado con diseminaciones de oxidos de fe
				114.80	117.85	3.05	2.60	85%	74.00	76.00	0.22	11%	21	74.95	77.50	Zona de oxidos ,con vetillas de calcita rellenadas por oxidos de fierro
				117.85	120.90	3.05	2.90	95%	76.00	78.00	0.75	38%	22			plono zinc.
				120.90	123.95	3.05	3.00	98%	78.00	80.00	1.90	95%	23	77.50	148.40	Granito muy fracturado y oxidado color rojizo con fracturas rellenas de cal
				123.95	127.00	3.05	3.00	98%	80.00	82.00	1.80	90%	23			cita de un mm. Y una estratificacion de 70° con respecto al eje de barreno

The geologist defined the sampling based the visible mineralization. Intervals were marked by drawing a line parallel to the core axis. All samples were one meter long. The sample was split using a rock saw. Half of the core was sent to the laboratory for assay. The remaining half was returned to the box to be kept for reference. RPM, in general, agrees with the procedures. During the 2013 drill campaign, Xtierra assayed internal intervals previously not assayed because there was no visible mineralization. This was done to improve the interpolation into unmineralized areas.

Parker's report (April 2010) indicated that "company geologists logged drill core using a narrative style that has incorporated most of the significant features of the mineralization, lithologies, alteration and structures, which allows for a reasonable geological interpretation on section. However, inspection of the logs and core showed that it was not always possible to discriminate the three mineralization types (sulphide, oxide and mixed) with sufficient accuracy for a reliable estimate of the resource." RPM took the mineralization limits from the interpreted sections to achieve the resources model. As the mineral types are the basis for metallurgical processing and the resources are defined for each of the mineral types, RPM strongly recommends logging as necessary and transferring the sulphide and oxide mineral contents to the database in order to accurately define each mineral zone for a Pre-Feasibility/Feasibility Study.

10.3 Drill Hole Database

Xtierra transferred all logging data from paper and the sample descriptions into Microsoft EXCEL. RPM received EXCEL files with collar coordinates, drill-hole trajectories, assays, and lithology. There are 15,450 Ag assays, 15,554 Zn, Pb and Cu assays, 136 survey points and 933 lithology registers.

Historical data, pre-2006, was not used to estimate resources because it lacks industry standard support for sample type, sampling protocol, sample security, QA/QC and assay type.

Xtierra underground channel samples could be incorporated to modelling - estimation processes to improve the model if there is sufficient support for sample locations and assay and QA/QC protocol.

10.3.1 Drill Hole Inventory

Xtierra provided RPM with geological logs for 113 drill holes. The ASCII database provided to RPM contained assay data for 108 holes. The geological model was generated using 113 holes (all the logged drill holes). Of the 113 holes with geological logs, 6 were not included in the database (X47, X47A, X57, X87, X90, and X91) and 2 did not have assays (G3 and Z11). In the database one hole had assays but no geological log and one hole (X100) was duplicated. The block resource model was estimated using 105 holes which had assays. Table 10-2 lists the 105 holes by drill campaign.

Table 10-2 Drill Hole Campaigns

Campaign	Year	N	(m)
Phase I	2006	28	7222
Phase II	2008	15	4138
Phase III	2008 - 2009	7	1900
Phase IV	2010 - 2011	31	7688
Phase V	2011 - 2012	18	4875
Phase VI	2013	6	1785
Total		**105**	**27609**

10.3.2 Drill Hole Trajectories

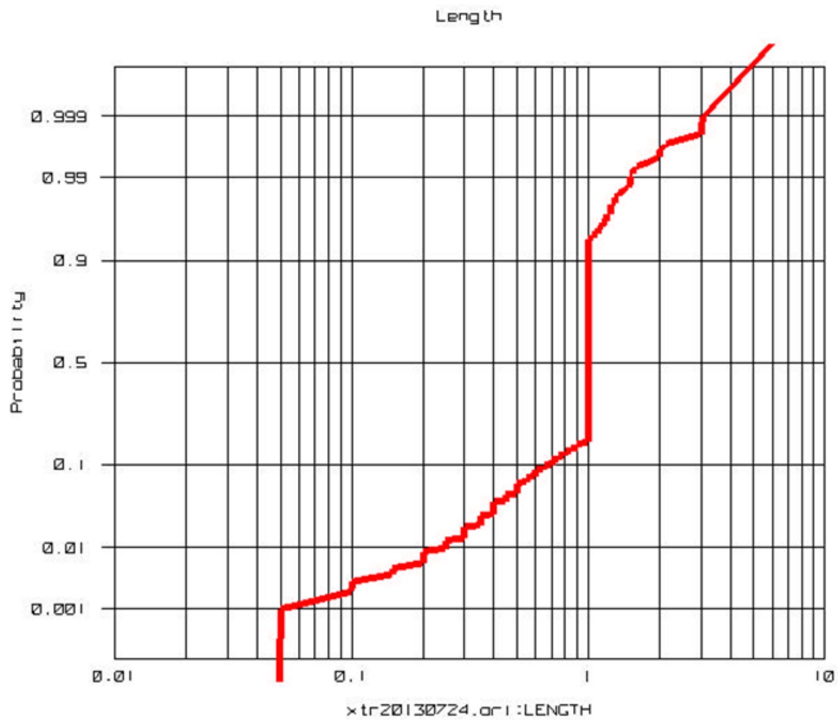
Because Xtierra believes that short vertical diamond drill holes have a minor deviation, the database contains the theoretical azimuth and inclination of the holes drilled in the first five campaigns. The hole depths, in the first five campaigns, vary between 95 and 449 m, with 23 hole depths greater than 300 m.

The down-the-hole surveys for the 2013 drilling (last campaign) were measured by reflexometer every 50 m in the six holes. RPM observed that drill-holes P1, P2 and P3 have between 4.3 to 4.65 m of horizontal deviation at about 300 m depth. For the resource estimation RPM assumed that vertical drill-holes without down-hole surveying were vertical. While RPM used the unsurveyed holes to classify indicated resources, all future drill campaigns should include down-hole surveys. The lack of down-hole surveys limits the accuracy of mine planning and the definition of measured resources will likely require sufficient drilling with down-hole surveying to better define the exact location of the mineralization in space.

10.3.3 Assays

The sample database contains 15,450 samples with Ag assays and 15,554 with Pb/Zn/Cu assays. Sample lengths vary a few centimeters to five meters, but most of the sample lengths are 1 m (78%) and 94% of them are less than or equal to 1 m. Figure 10-3 shows the sample length distribution.

Figure 10-3 Length Samples Frequency



The assay table contains zero values; Parker's report (April 2010) indicated that Ag zero values were assigned to values below detection limit. RPM assumed all the zero values as below detection limit values, and therefore, they were included in the composites. There are 2,747 Ag zero values, 2,820 Pb zero values, 2,390 Zn zero values and 8,065 Cu zero values. RPM would recommend reporting the detection limit by campaign/laboratory/method and assigning half detection limit value to assays under detection limit and negative codes to non-sampling and non-recovery intervals. When the geologist considered intervals to not be sampled because they were barren, a different code had to be assigned.

11. Sample Preparation, Analyses and Security

11.1 2006-2011 Campaigns

11.1.1 Sample Preparation

Parker (April 2010) checked the QA/QC of the first five (2006-2011) drilling campaigns, where half of the core samples were crushed, pulped and prepared at the analytical laboratories of SGS in Durango, Mexico and later by Stewart Group laboratory in Zacatecas, Mexico, following procedures similar to those used by SGS for previous drilling phases. Sample pulps were then sent to Stewart Group's dedicated certified laboratory, Eco Tech Lab in Kamloops, Canada. The Stewart Group analyzed for 38 elements using the ICP-MS technology. If the initial analysis showed values exceeding 10g/t Ag, 1% Pb, 1% Zn or similar over-limit values for the ICP-MS methodology these samples were automatically checked with further analysis using fire assay and atomic absorption spectrographic methodologies (BAUFG-14 and/or BM511).

Parker's report (April 2010) described that core sample preparation followed SGS PRP89 procedure, in which samples are dried at 100°C and then crushed to pass a 10mesh/2mm screen. 250g sub-sample, obtained via riffle splitter, is pulverized to 85% passing 75µm /200mesh for analysis. All samples were analyzed according to SGS ICP14B procedure in which a 0.25g pulp sample is digested in a mixture of nitric and hydrochloric acid (Aqua Regia). This digested sample is aspirated into the Inductively Coupled Plasma Optical Emission Spectrometer (ICP-OES). This light is recorded by optical spectrometers and, when calibrated against standards, provides a quantitative analysis for 33 elements. For core samples reporting base metals at percentage level by this method, sample pulps were fused with sodium peroxide prior to analysis for copper, lead, zinc and silver by ICP-OES according to SGS procedure ICP90Q. Many of the core samples (approximately 56%) were also analyzed for gold by fire assay with gravimetric finish (SGS procedure FAG323). This demonstrated sporadic anomalous gold values.

11.1.2 QA/QC

The laboratory's QA/QC results were shown in Parker's report (2011) for these drill campaigns (11,034 core samples). The SGS Minerales-Durango QA/QC protocol included blanks, duplicates, and reference samples. Additionally, Xtierra sent 80 pulps to Inspectorate America Corporation laboratory at Sparks, Nevada, USA for a second laboratory check. RPM opines the appropriate industry practice guidelines were generally followed by the insertion of project's (laboratory blind control insertion) blanks, coarse and pulp duplicates, and reference samples. Due to the project's lack of QA/QC for the first five Xtierra campaigns, RPM recommends second laboratory checks of at least 10% of the pulps, plus inserting low and high grade reference and blank samples for drilling completed for a feasibility study.

The charts shown in Figure 11-1 and Figure 11-2 were reported in the 2011 report.

Duplicates show consistently good repeatability in the 2011 scatterplots. RPM recommends indicating the nature of duplicates, coarse or fine, and incorporating the relative error – data percent graphs. The maximum error, currently accepted by industry, is 10% and 20% for 90% data for fine and coarse duplicates, respectively.

Although the bias of the standards reported in 2011 is between ± 5%, RPM observes that many reference samples are under the lower confidence limit in Ag, Pb and Zn and recommends showing the proportion of references sample results inside/outside both limits.

11.2 2013 Campaign

During 2013, 1319 samples from 6 new drill holes and 754 samples from previous campaigns that had not been assayed were sent to ALS at their Kamloops facility in Canada. The project's QA/QC associated with the 2073 samples included 40 blanks, 108 standards, and 115 duplicates, which are 263 samples inserted by Xtierra, equivalent to 12.6% of the total samples.

After the core was logged and density determinations were completed the core was sawed in half and in the case of duplicates, the remaining half was, again, sawed in half sending one quarter of the core as a coarse duplicate. Internal blanks (from a quartz vein) were inserted systematically throughout the analytical sample stream, as were the duplicates and standards.

11.2.1 Sample Preparation

The methods utilized for analysis were as follows:

1. Sample Prep—dry, crush, and pulp. Core Sample preparation followed SGS PRP89 procedure, same as the first drilling campaigns, in which samples are dried at 100°C and then crushed to pass a 10mesh/2mm screen. The sample was then split via a riffle splitter to produce a 250g sub-sample for analysis with the remainder stored as a reject/back-up sample. The 250g split is pulverized to 85% passing 75µm /200mesh.

2. ICP-MS for 51 elements, [ME-MS41] 0.25g pulp sample is digested in a mixture of nitric and hydrochloric acid (Aqua Regia). This digested sample is aspirated into the Inductively Coupled Plasma Optical Emission Spectrometer (ICP-OES) where the atoms in the plasma emit light with characteristic wavelengths for each element. This light is recorded by optical spectrometers and when calibrated, against standards, provides a quantitative analysis for 51 elements.

3. If Ag exceeds 10g/t, and/or Au 200ppb (detection limit) then the samples are assayed by Fire Assay with gravimetric finish. [ME-GRA21]

4. If Pb, Zn or Cu exceed 10,000 ppm (i.e. 1%), then the samples are checked by ore grade analysis using ICP-AES with sodium peroxide fusion (ICP-81).

While the principal estimated elements Cu, Pb, Zn & Ag were of paramount importance the use of ICP-MS as a broad spectrum element scan has enabled an appreciation of the minor elements such as W, Sn, Sb, Bi as well as the distribution of Fe and Mn within the mineral zone.

11.2.2 QA/QC

11.2.2.1 Blanks

Forty blank samples were inserted in 2013 campaign; results were under the values of 0.03% of Zn (see Figure 11-3), 0.008% of Pb, 0.0012% of Cu, and 3 ppm of Ag. These results are considered by RPM within acceptable limits.

11.2.2.2 Reference Samples

A reference material (CDN-ME-1204) from CDN Resource Laboratories Ltd. was inserted into the sample batches. The certified recommended values are shown in Table 11-1 below.

Table 11-2 and Figure 11-4 summarize the 2013 campaign results for the standard reference sample. Zn, Pb, and Cu biases are all within the accepted 5% limits, however, Ag has a bias is of -9.1%, which is greater than the accepted limit.

Figure 11-1 2006-2011 Duplicates (from 2011 report)

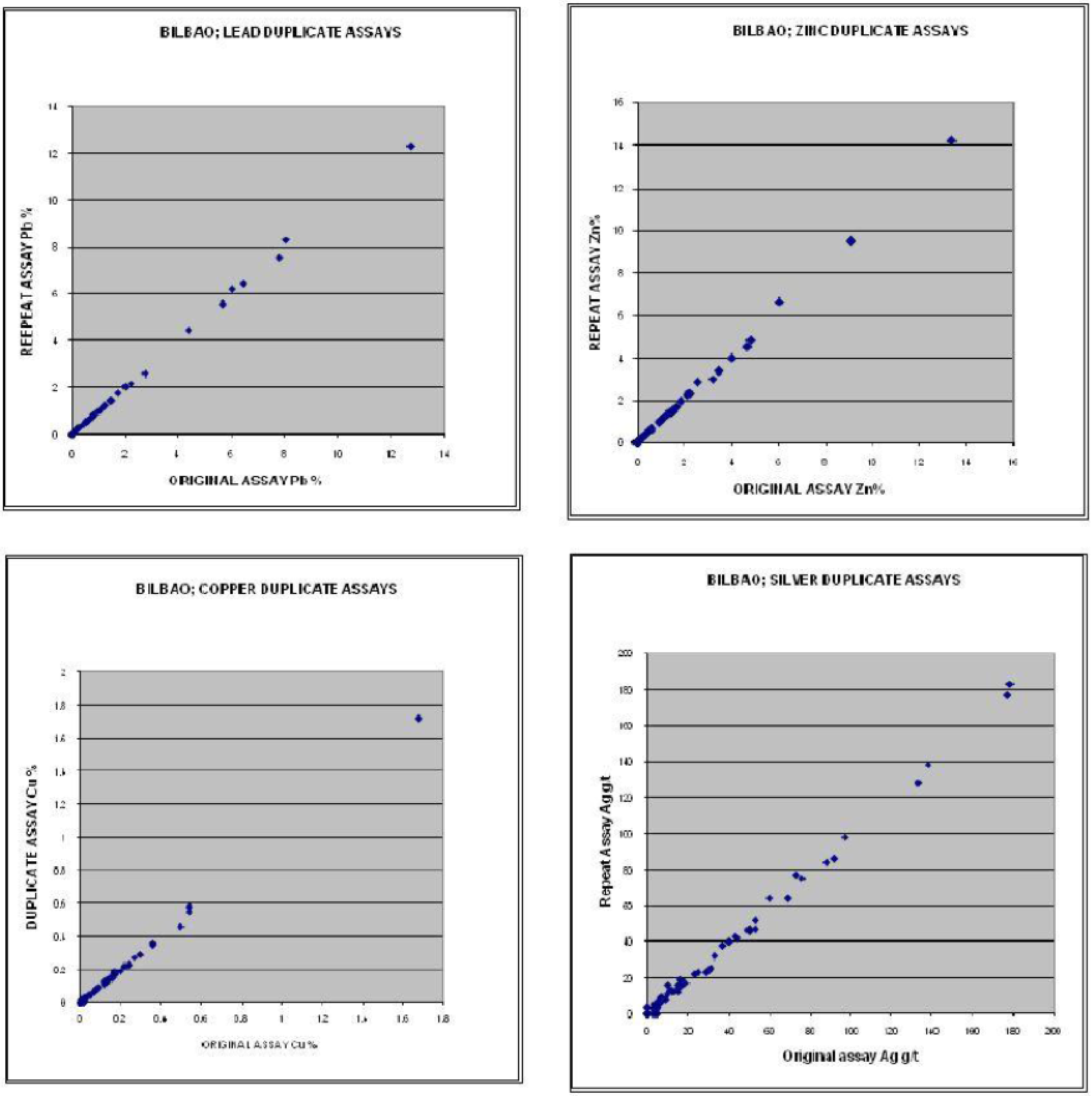
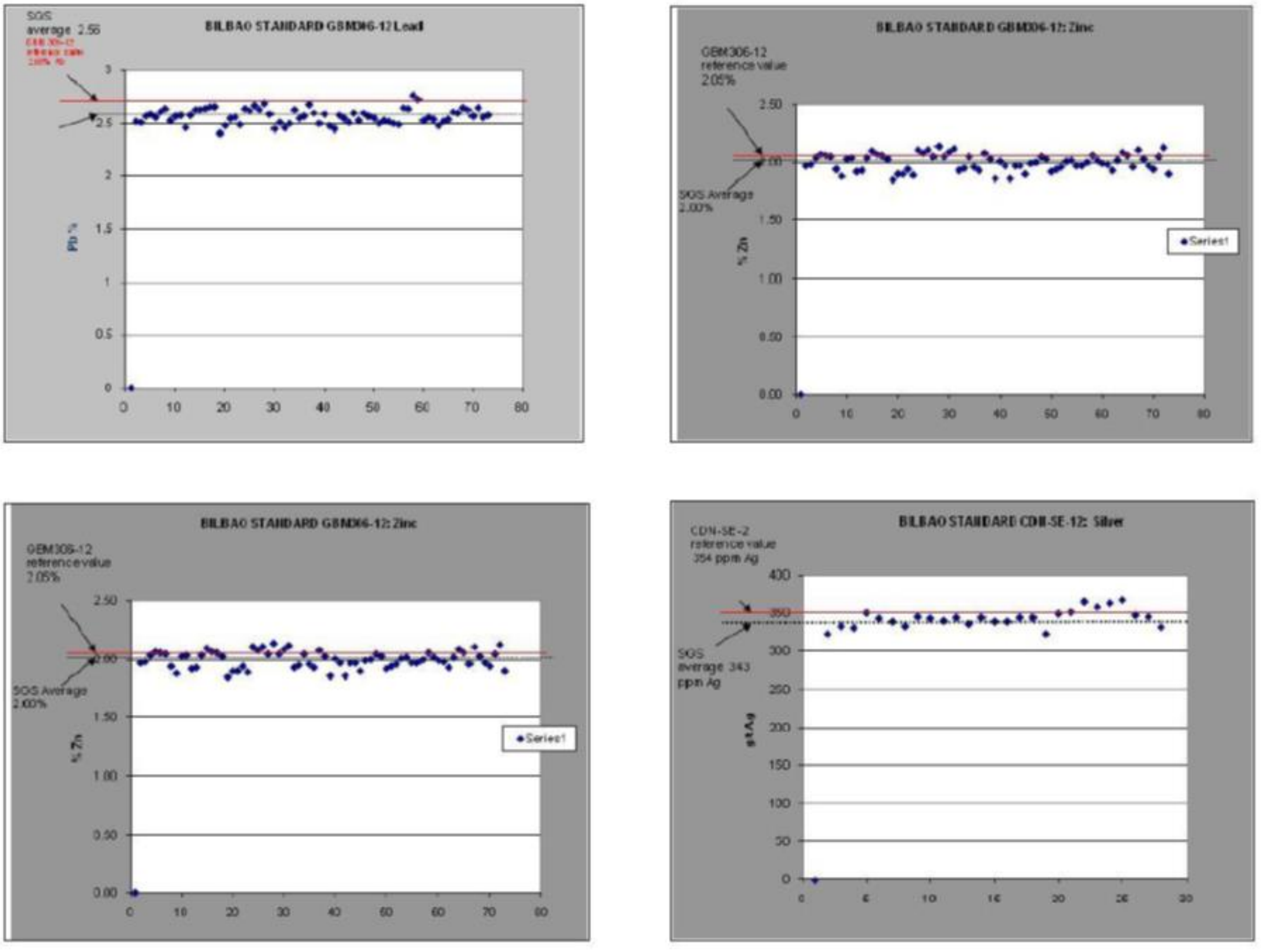


Figure 11-2 2006-2011 Reference Samples (from 2011 report)



Results out of the 2 STD lower and upper limits are greater than the industry accepted results of ±5%. Zn has 12%, Pb 7%, Ag 65%, and Cu 30% of the results out the limits. RPM strongly recommends researching the source of these poor reference sample results. If these out-of-limit results are confined to certain assay batches, RPM would recommend reassaying those batches along with the appropriate QA/QC samples. If the out-of-limit results are random with all batches, RPM would recommend sending out at least 10% of the pulps along with the appropriate QA/QC samples to a second lab for a check. If the biases of the assays of the standard samples are representative of the laboratory accuracy and the results from the core samples are similarly biased, the estimation of grade from these samples would be conservative.

Figure 11-3 2013 Campaign Blanks

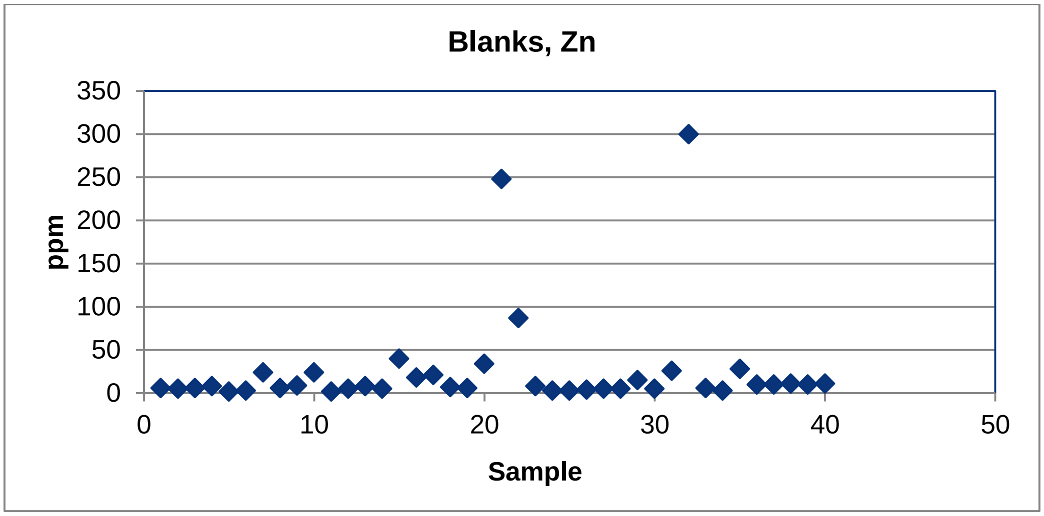


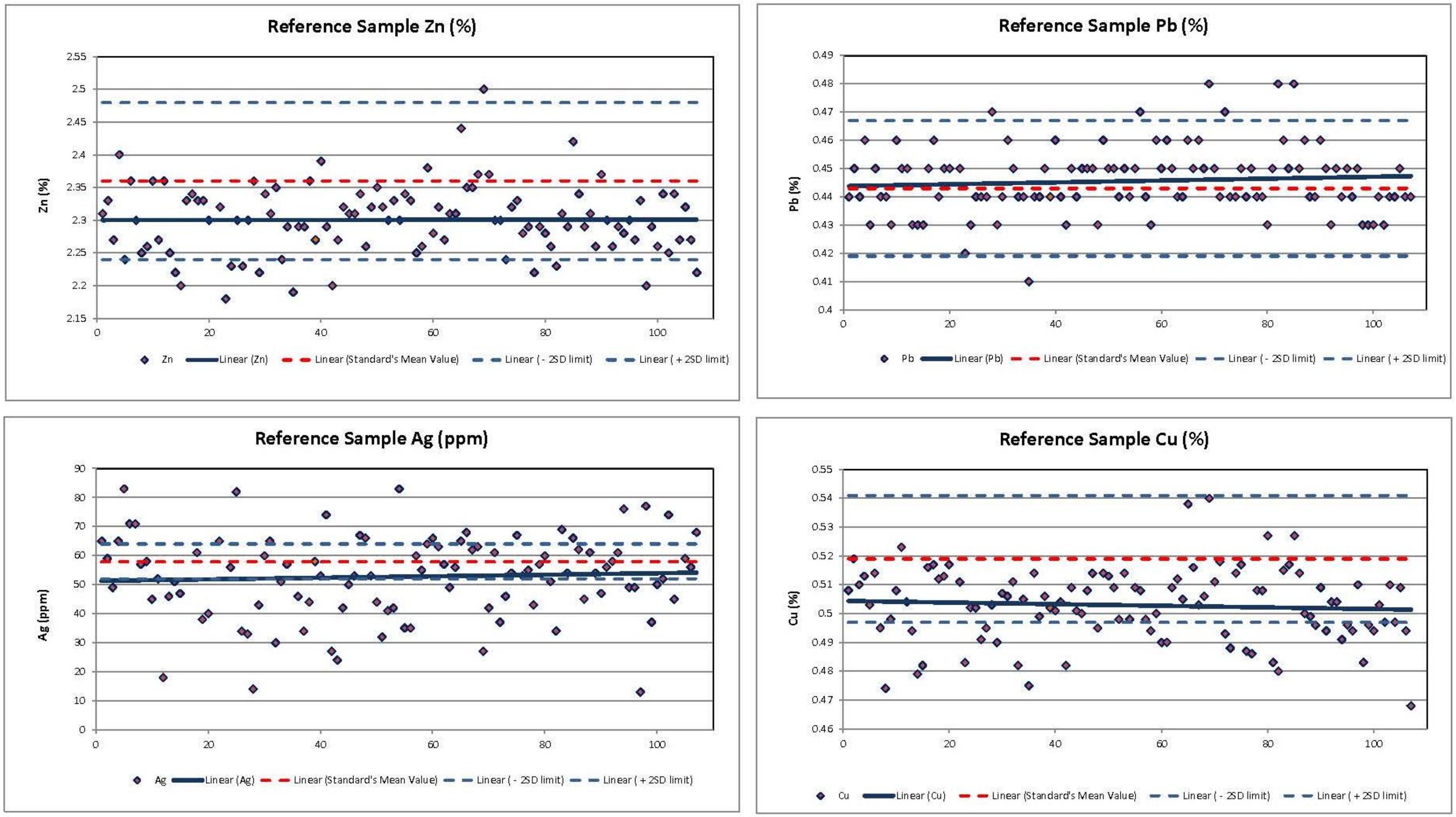
Table 11-1 Reference Values of Standard CDN-ME-1204

	Mean value	2 STD	Unit
Au	0.975	0.066	ppm
Ag	58	6	ppm
Cu	0.519	0.022	%
Pb	0.443	0.024	%
Zn	2.36	0.12	%

Table 11-2 Standard Results of 2013 Campaign

	Cu (%)	Pb (%)	Zn (%)	Ag (ppm)
Average	0.503	0.446	2.3	52.7
Expected value CDN-ME-1204	0.519	0.443	2.36	58
Bias	-3.10%	0.60%	-2.50%	-9.10%
# samples	106	106	106	101
% below Lower Limit	30%	1%	11%	43%
% above Upper Limit	0%	6%	1%	22%

Figure 11-4 2013 Campaign Reference Samples



11.2.2.3 Duplicates

Duplicates were taken cutting lengthwise a quarter of core (half of the half reject) of the same interval; these samples correspond to twin samples and not to duplicates, however RPM agrees that that is a good approach to get coarse duplicates using core samples. This report is using duplicates or twin samples to reference to the checking of half and quarter core assays. Xtierra inserted the twin samples regularly for each sample batch submitted.

Twin sample analyses are shown in scatter plots, max – min plot and relative differences chart in Figure 11-5.

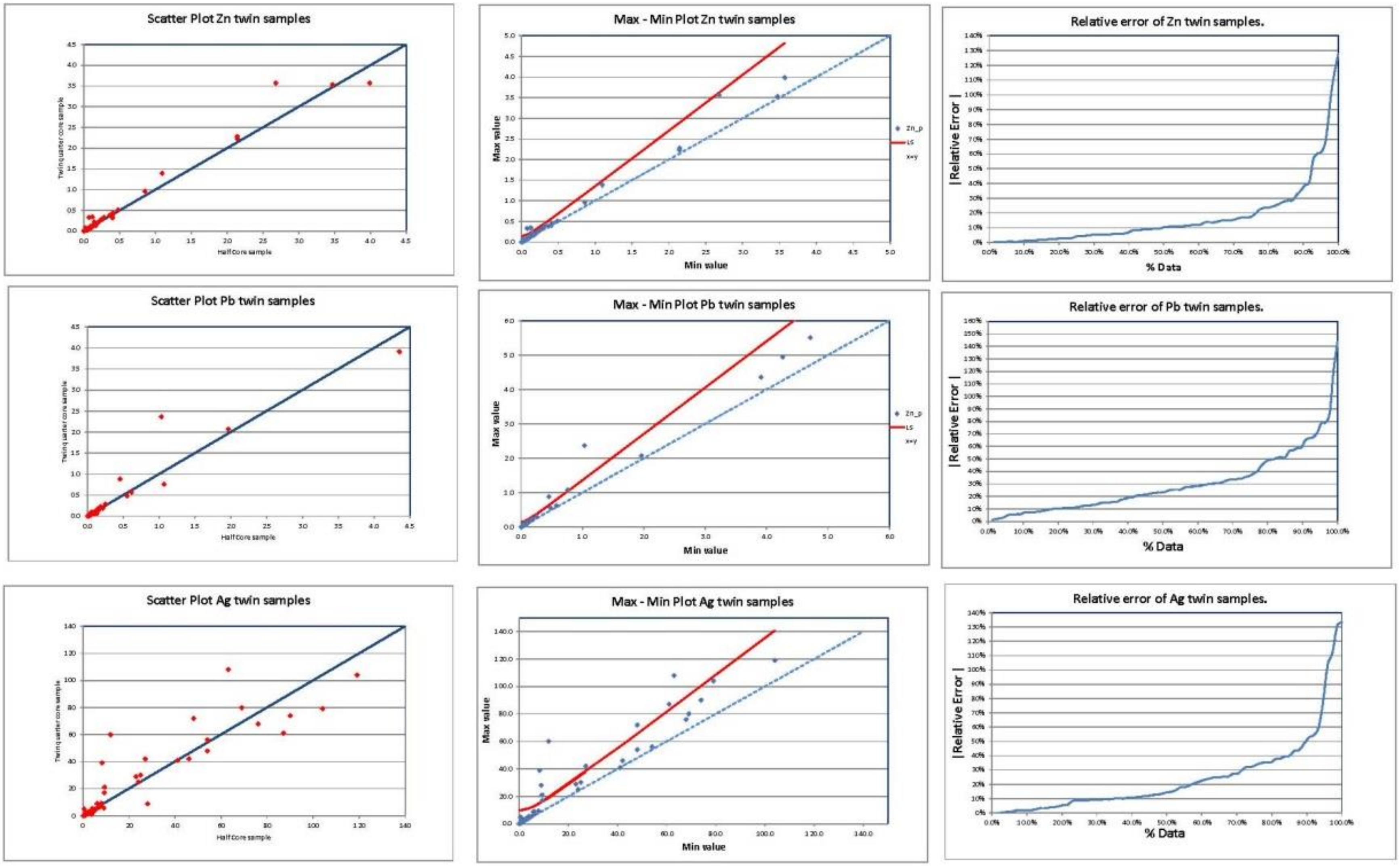
Zn duplicates show the best correlation. The Max – Min analysis indicates that two pairs of duplicates have a difference greater than 30% error (above the red line). Zn relative differences for values greater than 0.01% (84 of 115 duplicates), were plotted against percent of data to determine the percent of data with an error greater than 30% which is the accepted limit for 90% confidence in this control. RPM observed that 12% of data have an error greater than 30%, which is close to the 10% allowed for twin samples. RPM considers this to be within an acceptable range.

Pb twin samples show good correlation between pairs; just two pairs have a high difference. The Max – Min comparison graph indicates that just these two pairs of duplicates have a difference greater than 30% error (above the red line). Pb relative difference graph for values greater than 0.005% (87 of 115 pairs of duplicates), show that 36% of data have an error greater than 30%, which significantly exceeds the 10% allowed by the industry standard guidelines.

Finally, Ag twin samples have poorer correlation than Zn and Pb. There are nine pairs with high difference. The Max – Min graph indicates that these nine duplicates are out of the 30% allowed error (above the red line). Ag relative difference chart, using Ag values greater than 1 ppm (74 of 115 duplicates), show that 28% of data have an error greater than 30%, and which like Pb, exceeds the 10% allowed by industry standard guidelines.

RPM considers that Pb-Ag results may be acceptable given the low grade where the comparison is being made and the nugget effect existing in this high variability type of deposit. However, RPM would recommend checking the sampling protocol for every pair of duplicates. Further RPM would recommend assaying coarse rejects from the original core sample (which should be a “true” duplicate) to determine if this variability is a characteristic of this deposit.

Figure 11-5 2013 Campaign Duplicates



12. Data Verification

Parker in 2011 completed spot checks to ensure that the assay data had been accurately transcribed from the assay certificates; no errors were detected. There is no information about the methodology used and the amount of drillholes (data) checked in the Parker's report. RPM recommends documenting the database checking.

RPM verified the logs of three diamond drill holes during April, 2013 in the 3 day project visit. The holes reviewed were X13, Z1 and Z4; these holes intersected significant mineralized endoskarn in the high grade core. RPM verified the logs matched with core in terms of intervals and lithology/mineralization. RPM visited the site checking mineralized outcrops in and around the pits and the mantos in the two upper levels of the underground mine. RPM concluded the mantos are as continuous as the drift dimensions and the grade had high variability, but in general the mantos had visible Zn-Pb oxide mineral.

Afterwards, RPM spot checked three lab certificates, ICP certificate 2010 – 4529, 4523 and 4523-2 of the drill hole X-71. RPM detected differences in the rounding of the third decimal in Zn-Pb; this discrepancy is irrelevant for resource estimation, however, RPM recommends completely matching the database with the lab certificates. Zn, Pb, and Ag grade database mistakes were not found by RPM. Since database verification was not part of the original scope, RPM carried out limited data verification. However, RPM considers it is essential to complete data verification of at least 10% of holes prior to a feasibility study (FS). This data verification should include:

1. Field check of drill hole location;
2. Logging review;
3. Coordinate-log-assay certificates to database comparison.

During the modelling and estimation processes, RPM detected eleven overlaps in the assay table between the drill holes X12 and X12A which have the same location and down-the-hole survey. These overlaps are between the elevations 2064.94 and 2083.94. RPM arbitrarily corrected the overlaps, deleting the intervals of the drill hole X12. Also, the density table contains overlaps between the holes Z1 (143.7-143.25 m), Z3 (259.3-259.1 m and 276.37-276.35 m), and Z4 (213.16-213.15 m).

RPM concludes that the data collection and database development are accurate enough to support resource estimation for a PEA or PFS level study.

13. Mineral Processing and Metallurgical Testing

13.1 Introduction

Several metallurgical studies have been carried out on samples from the Bilbao deposit over the last 20 years, beginning in 1994-1995 with a bench program at Lakefield Research on behalf of Minera Portree. Much of the focus for the testwork has been directed towards metal recovery from the oxide zone of the deposit by various processes including gravity, froth flotation, and whole ore leaching. In contrast, work on samples from the sulfide zone is limited to three principal programs conducted in 2007, 2011, and 2012, as summarized in the following sections.

13.2 2007: SGS Durango, Mexico

A test program of seventeen flotation tests and two BBWI tests was carried out at the SGS laboratory in Durango, Mexico in the spring of 2007. Feed to the program consisted of 10 composite samples prepared from split core. The flowsheet for the flotation tests consisted of a selective approach to produce first a silver-lead concentrate followed by a zinc concentrate. In general, the silver-lead recoveries improved with the downhole depth of the samples, reaching 74% silver and 90% lead in sample 10 (236-245m). At the same time, zinc recovery was less successful, ranging from 6% to 57%. The report suggests that poor zinc recoveries were due to the presence of significant quantities of willemite (zinc silicate) in the ore.

Final concentrate lead grades varied considerably during the testwork, from 4% to 65%, and in general followed closely with lead recovery. Zinc grades in the final zinc concentrate were poor, with no test reaching 28% Zn. Tests were run on each sample at two primary grind sizes, 80% passing 100 mesh (150 microns) and 80% passing 200 mesh (75 microns). Grind size within the range tested did not appear to significantly affect the metallurgy reported.

13.3 2011: Universidad Autonoma de San Luis Potosi

Drill core samples from the Bilbao deposit were received in the fall of 2009 at the Instituto Metalurgia of the Universidad Autonoma de San Luis Potosi (UASLP). Samples identified in the core register as part of the sulfide zone were used to generate three sub-composites, labeled as Sulfide I, Sulfide II, and Sulfide III. Similar composites were produced representing the oxide and transition zones.

An overall Master Sulfide Composite was formed by combining equal parts of the three sub-composites. The Master Composite graded 2.37% lead, 3.29% zinc, 0.28% copper, 102 g/t silver, and 16.9% iron. Sulfur grade was not measured during the program.

A series of bench-scale, rougher and cleaner flotation tests were conducted with the objective of identifying the optimal process conditions for generating saleable lead and zinc concentrate products at maximum metal recovery. The starting point for these tests was a conventional lead-zinc flowsheet consisting of a sequential rougher float followed by regrinding and cleaning of the respective lead and zinc rougher concentrates.

Initial tests focused on the primary grind size and indicated improved zinc flotation to the rougher zinc concentrate at a grind size P_{80} of 74 µm as compared to a coarser P_{80} of 150 µm. Lead recovery was not significantly affected by grind size, although grade of the lead rougher concentrate improved with finer grinding due to lower zinc recovery in the lead float.

Good recoveries of lead and silver to the lead rougher concentrate were achieved with the addition of collectors A3418, A211, and X-343. In addition, some fast-floating zinc was observed to report to the lead rougher concentrate and required the addition of depressants, in the form of sodium metabisulfite, zinc sulfate, and sodium sulfide. Optimal activation and flotation of the zinc was achieved through the addition of copper sulfate and A211 at a pH of 10.5.

Mineralogical characterization was carried out on the rougher lead and zinc concentrates by SEM X-ray microanalysis. Results indicated that the relatively low zinc grade observed for the zinc concentrate was largely due to the predominance of iron rich marmatite zinc sulfide, rather than sphalerite. The average zinc grade of the contained zinc sulfide was found to be 54.9%, compared to stoichiometric ZnS at 67.1%. Dilution in the lead concentrate was found to be the result of association with sulfide and non-sulfide gangue minerals for galena particles of 20µm or less.

Batch cleaner flotation of the rougher concentrate focused on adding a regrind step to improve sulfide liberation and iterative cleaning stages to reject entrained gangue. Baseline conditions consisted of a target regrind for both the lead and zinc concentrates to a P_{80} of 20 µm. The reground concentrate was then floated in three stages with the tailings collected for assay after each stage.

Batch cleaner testwork focused on upgrading of the lead concentrate through regrinding and fine tuning of the iron and zinc depression scheme. Moderate success was achieved, with lead concentrate grades of up to 61.4% Pb at varying recoveries of up to 77%. Upgrading of the zinc concentrate through regrinding and three stages of cleaning resulted in final concentrate grades in the 44-48% Zn range, with a maximum zinc recovery of 51.3%.

Three locked cycle tests were conducted on the Master Composite to evaluate the effect of recycle streams on the concentrate grades and recoveries. The tests consisted of five cycles each, with the cleaner tails streams reporting to the previous cleaner feed of the next cycle and the lead first cleaner tailings going to the zinc rougher feed of the next cycle.

The effect of “locking” the flowsheet resulted in improved zinc recovery to the zinc concentrate due to rejection of entrained zinc sulfide from the lead circuit back to the zinc roughers. Results of the final and most favourable, locked cycle test are presented in Table 13-1.

Table 13-1 Locked Cycle Test Results from the UASLP Metallurgical Program

Product	Weight	Grade					Distribution				
		Ag	Pb	Zn	Cu	Fe	Ag	Pb	Zn	Cu	Fe
	%	g/t	%	%	%	%	%	%	%	%	%
Lead Concentrate	3.23	**1595**	**46**	2.17	2.26	6.3	**64.3**	**81.4**	2.2	45.9	1.2
Zinc Concentrate	6.23	143	0.94	**40.5**	0.86	13.5	9.3	2.7	**66**	28.3	4
Combined Tailings	91.54	23	0.32	0.04	0.04	1.11	26.3	15.9	31.8	25.8	94.9
Calculated Head		*80*	*1.83*	*3.21*	*0.16*	*17.77*					

Lead and silver recoveries to the final lead concentrate are comparable to those in the open circuit work. Overall zinc recovery is improved due to the recycling of the lead circuit first cleaner tail to the zinc roughers. However, both lead and zinc final concentrate grades are low, and in the case of the zinc concentrate not all of the dilution can be attributed to the iron content of the marmatite.

13.4 2012: SGS Minerals Services

Based on the positive flotation results achieved in the UASLP study a more comprehensive program was undertaken at SGS Minerals Services in Lakefield, Canada. Approximately 525 kg of split core samples representing the sulfide and transition zones of the Bilbao deposit were received at SGS on August 23, 2011.

13.4.1 Sample Selection

The samples were taken from metallurgical drill holes M1 and M3, as shown in Figure 13-1. The green region represents the transition zone of the deposit, whereas the sulfide zone is shown in purple. Interval samples were combined to form composites based on downhole depth. In total, seven zone composites were generated: two from the transition zone; and five from the sulfide zone.

Based on the assay data for the intervals, the metal grades of the composites were estimated. Table 13-2 provides a summary of these estimates and the elevation of the composites in the deposit.

Table 13-2 Summary of Projected Head Assays and Elevations for the Seven Zone Composites

Composite	Weight	Ag	Pb	Zn	Cu	Elevation, M		Hole
	kg	ppm	%	%	%	From	To	
Transition A	50.1	3.22	0.14	3.04	0.03	2036.6	2020.0	M3
Transition B	69.4	196.7	4.59	5.61	0.35	2003.4	1992.0	M1
Sulphide A	58.9	63.6	3.11	2.89	0.16	2012.0	1999.2	M3
Sulphide B	86.6	61.4	1.24	0.82	0.34	1992.0	1982.3	M1/M3
Sulphide C	99.7	26.1	1.71	1.44	0.12	1982.3	1970.9	M1/M3
Sulphide D	85.2	62.1	3.34	4.04	0.23	1970.9	1959.6	M1/M3
Sulphide E	74.8	48.3	2.26	2.79	0.14	1959.6	1946.7	M1/M3

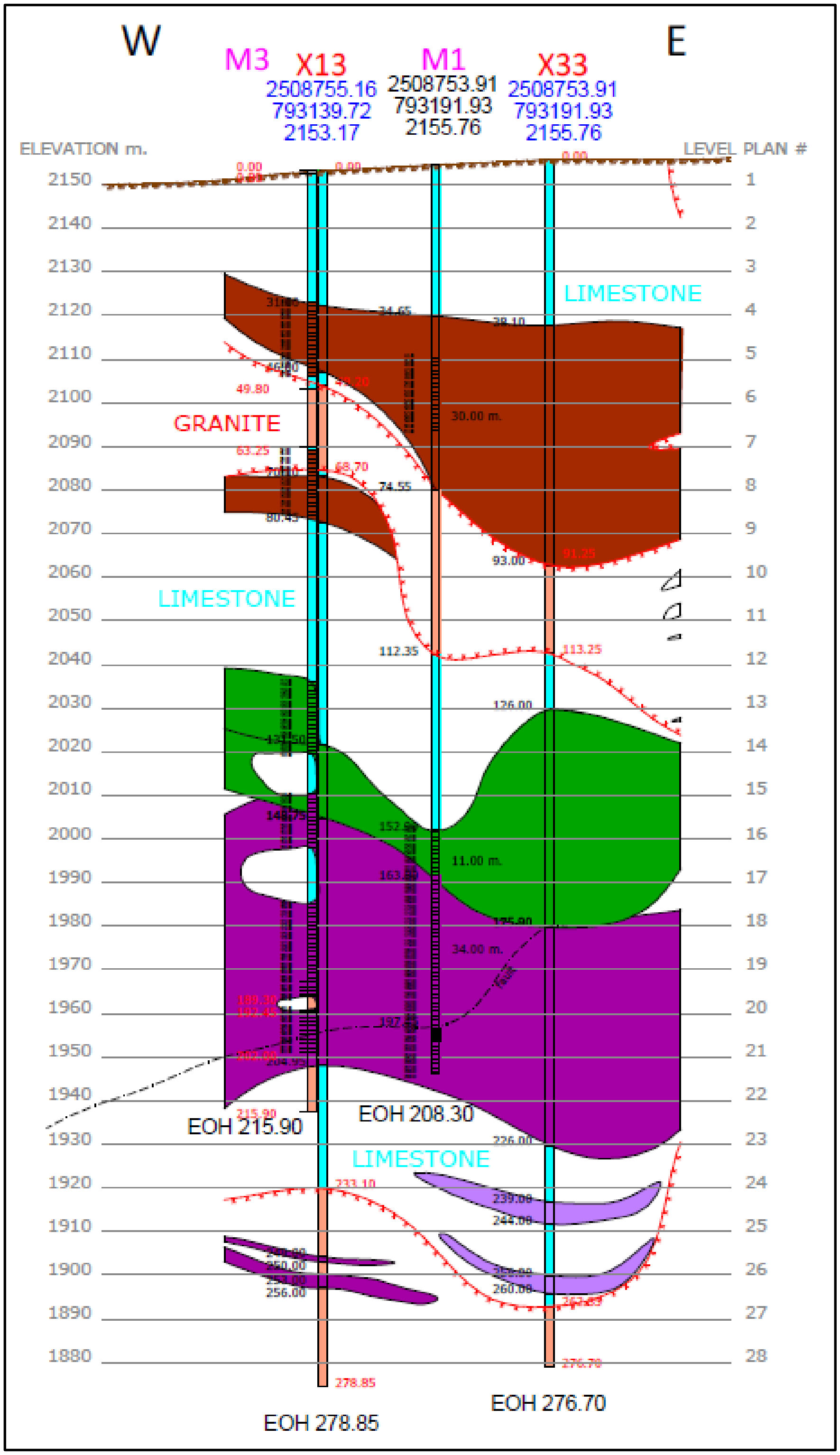
Composite samples were crushed and blended, and grindability test charges were removed, as required. The remaining composite was crushed down to minus 10 mesh and split into charges for flotation testwork. Two master composites were generated, one each for transition and sulfide zones, by combining the individual composites in the respective area.

Gold grade of the composite samples did not exceed 0.03 g/t and were below detection limit in several instances. Silver grades for the sulfide composites ranged from 24 g/t in Comp C to 89 g/t in Comp B. For the two transition composites a large variation in silver grade was observed: from 4 g/t in Comp A to 210 g/t in Comp B.

13.4.2 Mineralogy

Head samples of the Sulfide Master Composite (SMC) and Transition Master Composite (TMC) were ground to a P_{80} of ~100 µm and submitted for quantitative mineralogical characterization by QemSCAN. The samples were screened into three size fractions and prepared as polished sections for particle mapping and liberation study. The TMC was found to include a major component of iron oxides, 28.5%, and the calcium-iron silicate hedenbergite, 18.8%. Whereas the SMC was characterized by Ca-Mn carbonates, 16.2%, hedenbergite, 15.6%, and pyrite, 12.3%.For both composites, lead was predominantly found as galena, while zinc was present as sphalerite/marmatite. Microprobe point analysis of zinc mineral grains in each sample revealed an average grade of ~49% Zn.

Figure 13-1 Section Drawing of Metallurgical Drill Holes M1 and M3



13.4.3 Grindability Testwork

Grindability testing consisting of SAG Mill Competency (JK Parameters, DWI), Bond Rod Work Index (BRWI), Bond Ball Work Index (BBWI), and Abrasion Index (AI) tests were conducted on the zone composite samples, and the results were fed into a grinding circuit simulation. Table 13-3 presents a summary of the grindability testing on the sulfide and transition zone composites.

Table 13-3 Summary Grindability Testwork Results

Composite	Relative Density	JK Parameters A x b	JK Parameters t_a	DWI kWh/m^3	BRWI kWh/t	BBWI kWh/t	AI
Transition A	**2.42**	**131**	**1.40**	1.85	-	11.8	-
Transition B	**3.38**	**48.2**	**0.37**	7.1	-	12.2	-
Sulfide A	**3.05**	**49.3**	**0.42**	6.21	-	14.8	-
Sulfide B	**3.31**	**54.8**	**0.43**	6.06	-	16.2	-
Sulfide C	**3.01**	**53.7**	**0.46**	5.60	-	15.2	0.11
Sulfide D	**3.04**	**64.5**	**0.55**	4.71	-	14.1	-
Sulfide E	**3.29**	**38.8**	**0.31**	8.47	-	14.6	-
Sulfide B+C+D	**-**	**-**	**-**	-	14.3	-	-

The results indicate that the sulfide zone is of average hardness for this type of material and that the transition zone is slightly softer. The milling design proposed based on the results presented here consists of a conventional SAG mill in closed circuit with a trommel screen followed by a ball mill in closed circuit with a cyclone cluster.

13.4.4 Flotation Testwork

Rougher Flotation Tests

Initial rougher flotation testwork was carried out on the Sulfide Master Composite with the objective of optimizing lead and zinc recovery to the respective rougher concentrates. Starting test conditions were based on the conventional sequential flowsheet explored in the UASLP program. Sodium cyanide and zinc sulfate were added to the primary grind to depress zinc in the lead circuit. Reagent conditions for the first rougher flotation test are presented in Table 13-4.

Due to the high pyrite content of the composite the use of xanthate in the roughers was avoided. Instead, a diothiophosphate promoter (A211) was used along with the dialkyl dithiophospinate 3418A. Following the lead flotation the pH of the pulp was raised to 10.5 with the addition of hydrated lime and copper sulfate was added to activate the zinc.

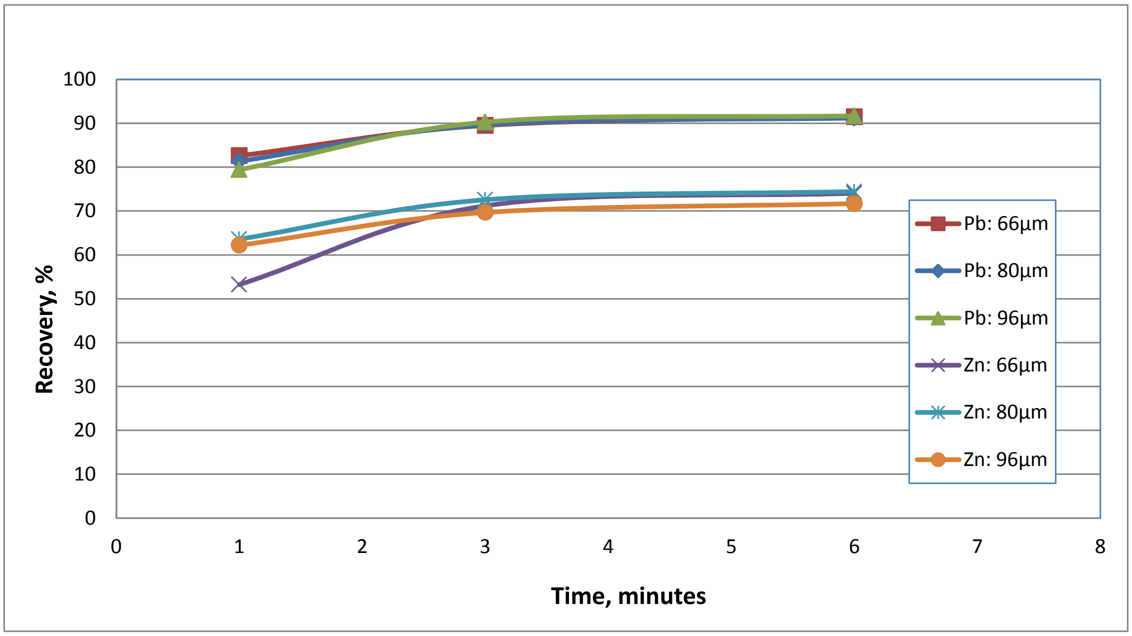
An initial grind size P_{80} of ~80 µm was selected based on historical testwork on the deposit. The first three rougher flotation tests examined the effect of primary grind size on the kinetics of lead and zinc recovery.

Figure 13-2 presents the recovery data from these tests. For lead, no noticeable difference in recovery was observed for the P_{80} size range tested: 66 µm to 96 µm. The results are similar for zinc, with a slight decrease in final recovery at the coarsest grind, although this is probably within the margin of error for this type of test. Based on the results of this series a grind size P_{80} target of 100 µm was selected for all subsequent testwork.

Table 13-4 Initial Conditions for the Rougher Flotation Tests

Stage	Reagents added, grams per tonne								Time, minutes		
	Lime	NaCN	ZNSO4	CuSO4	A211	A3418A	MIBC	DF250	Grind	Cond.	Froth
Grind	50	50	350						35		
Collectorless Rougher	90						10			1	3
Condition						15				1	
Pb Rougher 1					5		5			1	1
Pb Rougher 2	15				5		10			1	2
Pb Rougher 3					5		5			1	3
Condition	1510			500						2	
Zn Rougher 1	230				10			5			1
Zn Rougher 2	275				10			5			2
Zn Rougher 3	175				10					1	3
Total Rougher	2345	50	350	500	45	15	30	10	35	8	15

Figure 13-2 Effect of Grind Size on Rougher Flotation Kinetics



The effect of reagent changes in the rougher circuit was investigated in three tests on the Sulfide Master Composite. The changes consisted in variations in collector and copper sulfate addition, and are summarized in Table 13-5. Test F3 represents the optimized conditions from the grind size series of tests.

Table 13-5 Reagent Additions for Rougher Flotation Reagent Screening Tests, F3-F6

Test	Reagent Dosage, g/t								
	Lime	NaCN	$ZnSO_4$	CuSO4	A211	3418A	AP3894	MIBC	DF250
F3	2195	50	350	500	45	15	0	30	10
F4	2090	50	350	250	30	30	0	25	15
F5	2275	50	350	500	30	7.5	0	25	15
F6	2175	50	350	500	30	15	30	20	15

In Figure 13-3 and Figure 13-4 the effect of these reagent changes on the lead and zinc grade/recovery curves is presented. For the lead concentrate, overall lead recovery was fairly consistent between tests at ~91%. However, lead grade of the concentrate was significantly different in test F5, where the dosage of collector 3418A was cut in half, as compared to the other tests in this series. The apparent effect of the lower reagent addition is to improve selectivity of galena flotation over other sulfides, particularly pyrite, without adversely affecting the recovery of lead.

Figure 13-3 Effect of Collector Addition on Lead Flotation

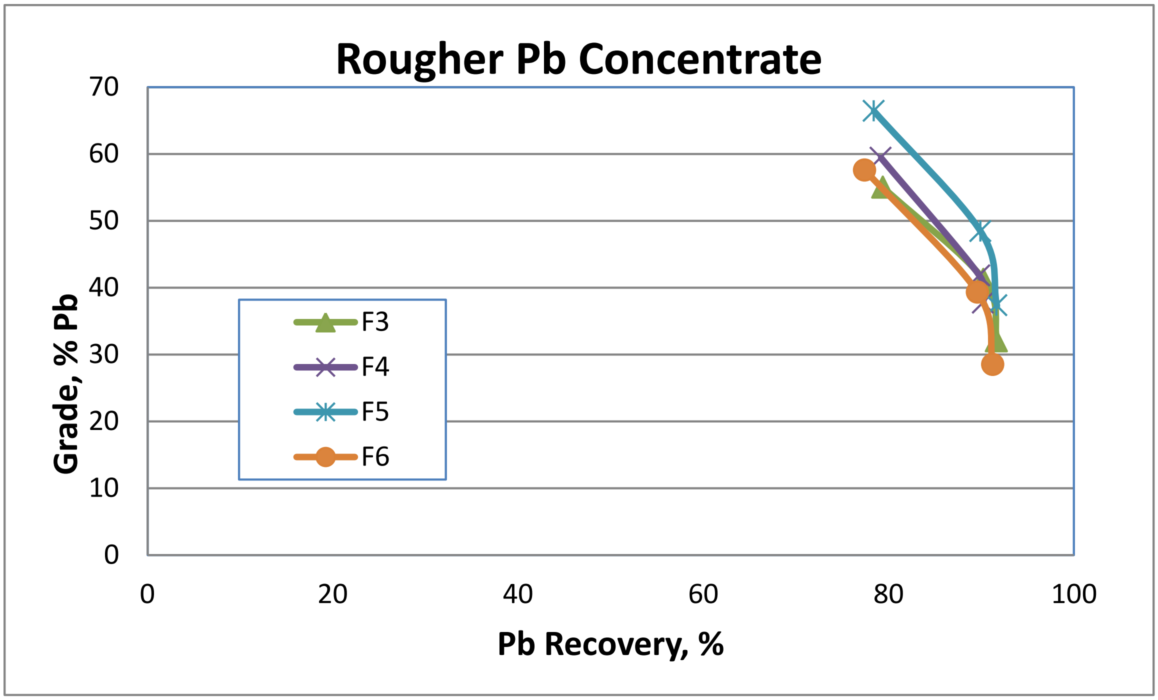
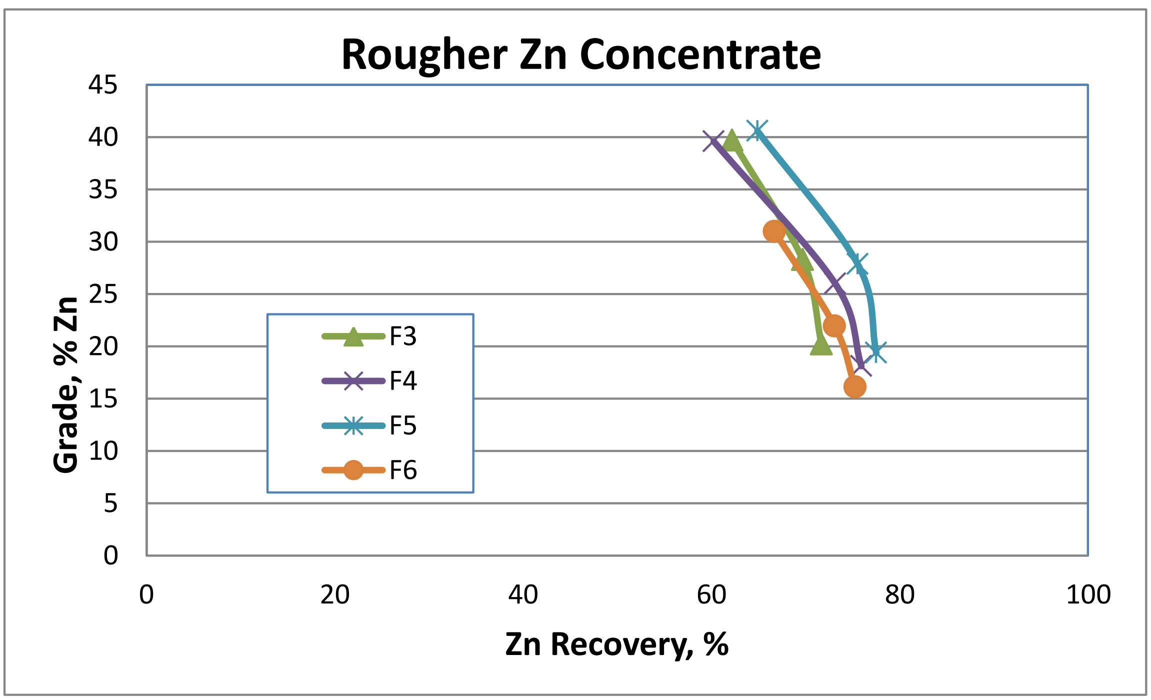


Figure 13-4 Effect of Collector Addition and Copper Sulfate Dosage on Zinc Flotation

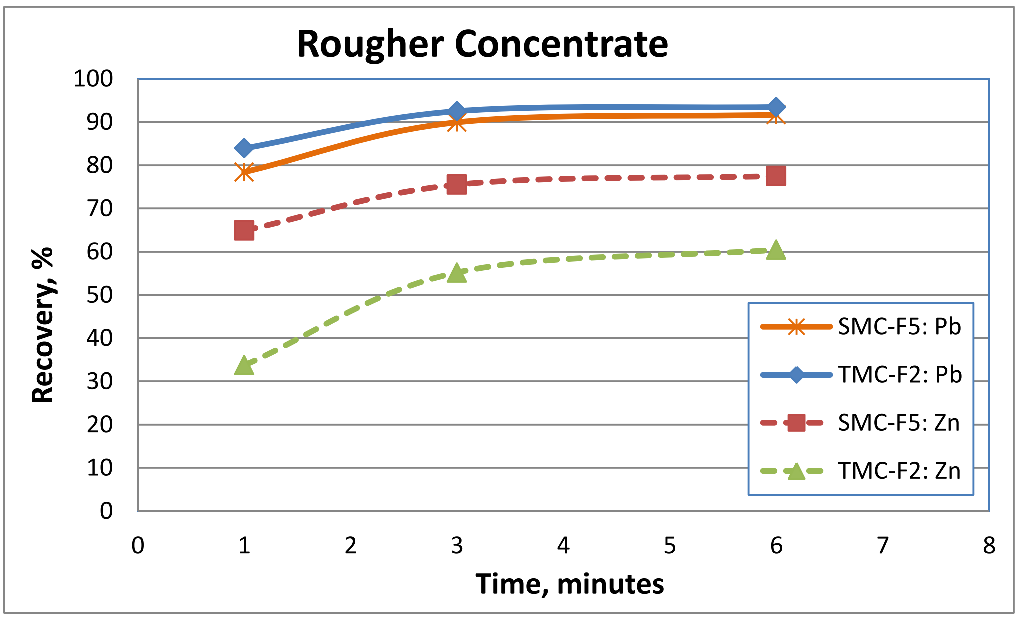


For the zinc concentrate, a similar distribution of results was observed, with Test F5 demonstrating the best results in terms of both grade and recovery. This effect is most likely due to the improved selectivity of galena over sphalerite/marmatite in the lead flotation stage of this test. More floatable zinc reported to the lead rougher tails in F5, and subsequently this improved the grade/recovery relationship in the zinc circuit.

The effect of PAX (potassium amyl xanthate) addition on zinc recovery was investigated at the end of test F3 to evaluate whether additional floatable zinc was not being recovered under the chosen reagent scheme. A dosage of 50 g/t PAX resulted in an increase in zinc recovery of only 1.4%, while at the same time recovering an additional 48% of the contained sulfur. Given the poor selectivity of sphalerite/marmatite over pyrite, as compared to A211, no further testwork using xanthates was conducted.

The optimized rougher flotation conditions (F5) from the Sulfide Master Composite were applied to the Transition Master Composite and a comparison of the recovery results is presented in Figure 13-5. Lead rougher recovery for transition composite was ~4% higher than for the sulfide composite, despite having a slightly lower head grade. Conversely, zinc recovery from the transition composite was much lower than for the sulfide composite under the same test conditions.

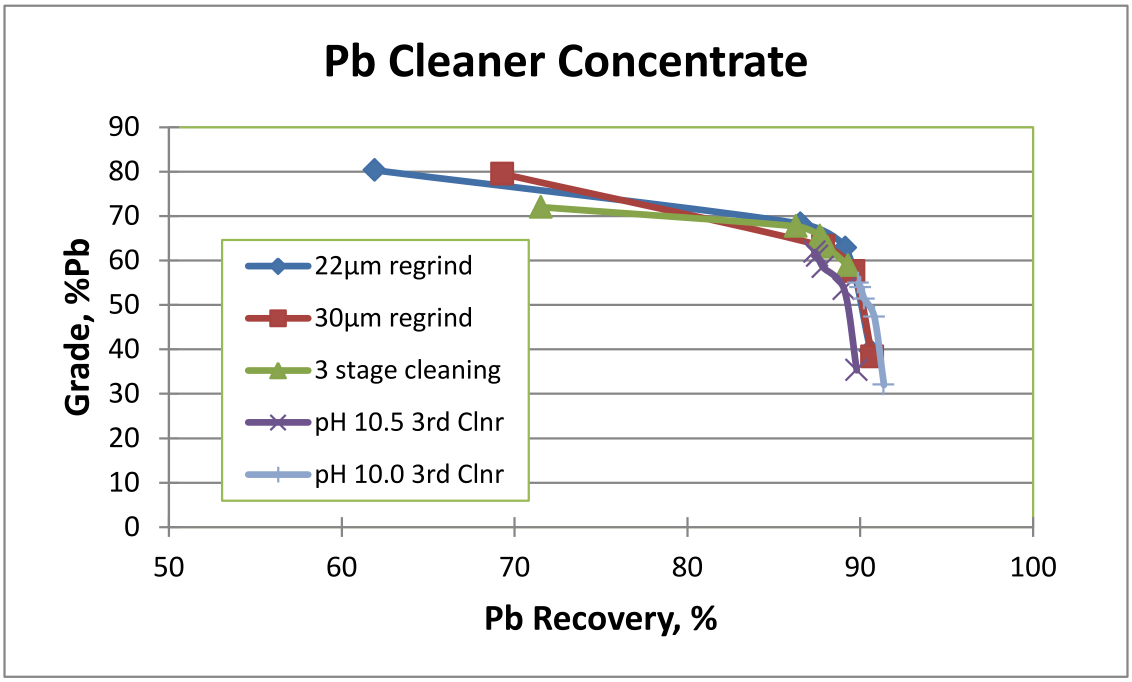
Figure 13-5 Comparison of Rougher Kinetics for the SMC and the TMC



Cleaner Flotation Tests

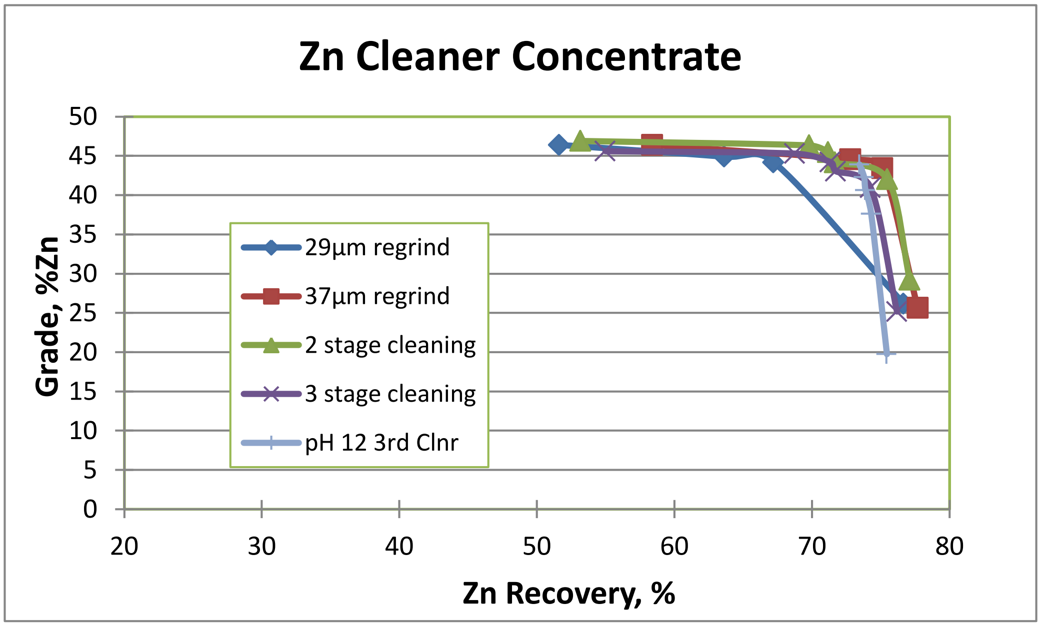
Bench-scale, batch flotation tests were carried out on the Sulfide Master Composite to develop the cleaner conditions necessary to optimize grade and recovery for the lead and zinc concentrates. Preliminary tests in this series consisted of 1st cleaner kinetics tests to optimize regrind size and flotation time. Figure 13-6 illustrates presents some of the grade-recovery curves for the tests on the lead cleaner circuit. The effect of finer regrinding to a P_{80} of 22 µm did not have a significant effect on grade or recovery. Additional cleaning stages and lower pH in the final cleaning stage were found to have only a small influence on the lead recovery. Overall, the lead circuit performance was good, with lead concentrate grades in the low 60's, and stage recoveries over 95%.

Figure 13-6 Comparison of Grade-Recovery Curves for the Lead Cleaner Tests on the SMC



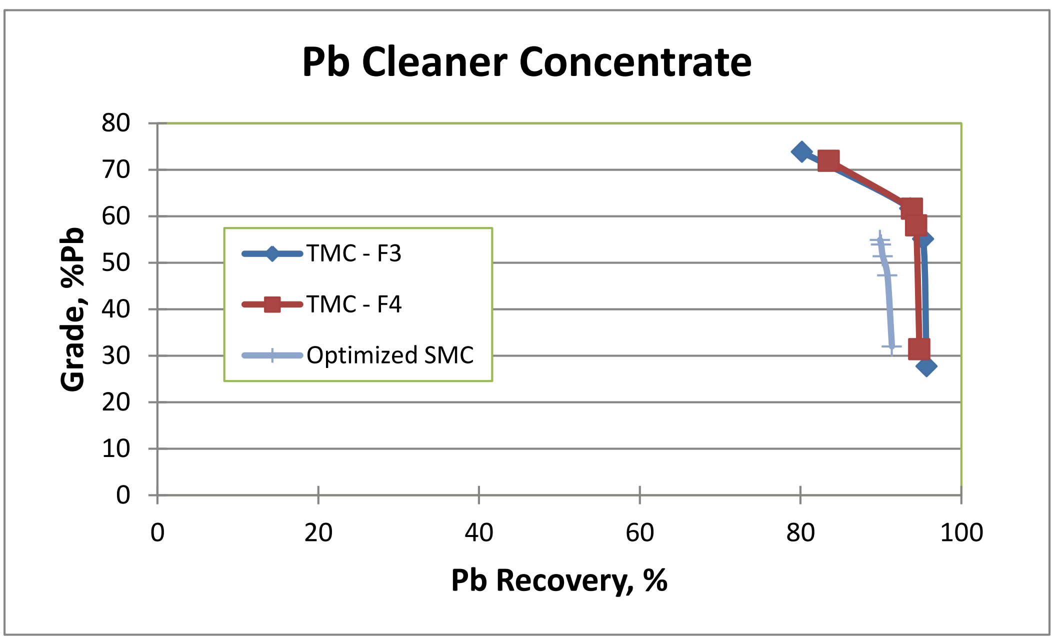
Zinc stage recoveries in the cleaner circuit were also good. Selected cleaner test results are presented in Figure 13-7. For the zinc circuit finer regrinding was found to have a negative effect on recovery to the first cleaner concentrate, probably due to difficulty in floating zinc slimes. At the coarser regrind good cleaning efficiency was observed, although recovery began to drop off sharply in all tests as the concentrate grade approaches 45%. The highest concentrate grade observed in these tests was 47.2% Zn and this is consistent with the findings of the microprobe analysis summarized earlier, which identified the zinc grade of the sphalerite/marmatite mineral as ~49%.

Figure 13-7 Comparison of Grade-Recovery Curves for the Zinc Cleaner Tests on the SMC



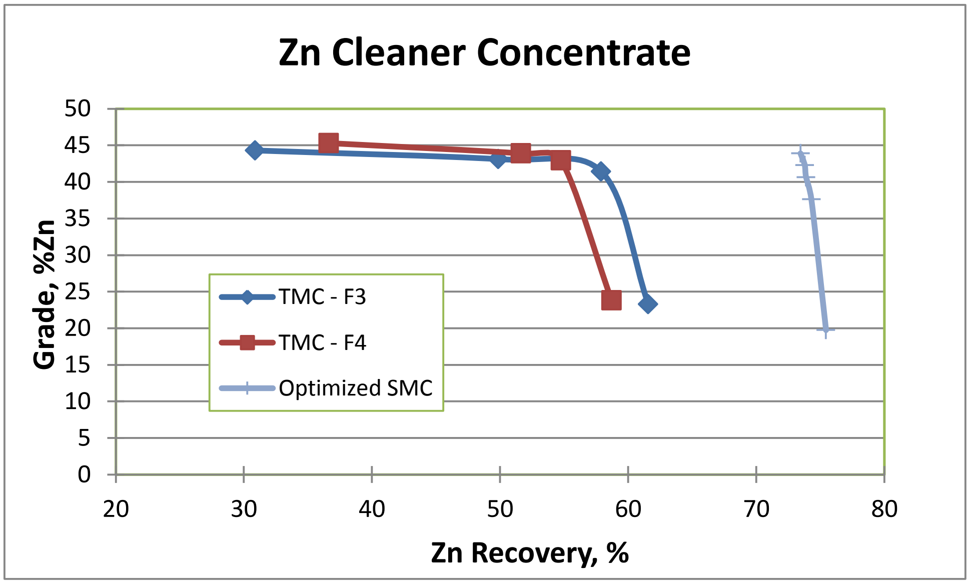
The optimized cleaner flotation conditions developed for the Sulfide Master Composite were applied to the Transition Master Composite. Figure 13-8 and Figure 13-9 present the grade recovery curves for two such tests and the optimized results from the SMC. Cleaning of the lead concentrates followed a similar response to the sulfide samples, except that the TMC tests started at a higher rougher lead recovery of close to 95%.

Figure 13-8 Comparison of Grade-Recovery Curves for the Lead Cleaner Tests on the TMC



The zinc response was also comparable, again with the main difference being the lower rougher recovery starting point. As with the SMC samples, the grade-recovery curve changes sharply as the concentrate grade approaches 45%.

Figure 13-9 Comparison of Grade-Recovery Curves for the Zinc Cleaner Tests on the TMC



Rougher lead concentrates for the Sulfide Master Composite were found to typically grade 2-3% copper, and thus the opportunity to generate a separate copper concentrate product was investigated. Initial efforts in this area consisted of adding an additional cleaning stage to the lead rougher circuit and raising the pH in this step to 12.0 with and without the addition of dextrin as a lead depressant/dispersant. Results were disappointing: with copper concentrate grades of less than 5% being achieved.

Follow up tests examined the use of metabisulfite to depress lead and improve the copper grade, without success. Potassium dichromate was then tested and resulted in a copper grade of 17.8% at a copper recovery of just over 50%. Dilution of the grade was attributed to lead and zinc sulfides floating as well. Additional cleaner stages were expected to reduce the lead contamination, but mineralogical study of the concentrate indicated that the zinc was present as middlings and sphalerite with fine inclusions of chalcopyrite. As a result, a test was conducted using potassium permanganate in the final copper cleaner to depress the chalcopyrite and reverse float the zinc. While zinc flotation was achieved in the last step, the final copper concentrate grade improved only slightly, to just over 18%.

The difficulty in achieving a saleable copper concentrate product for the SMC is believed to be at least partially attributable to the low grade of the sample. The copper head grade was measured at only 0.21%, less than 1/10th of that for lead and zinc individually. In addition, rougher copper recovery was typically only in the 50-60% range, which further contributed to low concentrate grades in the cleaner circuit.

Despite the poor results observed with the SMC, a potential opportunity still exists for generating a copper concentrate product if zones of the deposit have elevated floatable copper grades can be identified, and if those zones can be campaigned separately through the mill.

Locked Cycle Testing

A bench-scale locked cycle test was conducted on the SMC to evaluate the effect of recycle streams on final concentrate grade and recovery. The flowsheet for the test was based on the optimized conditions from the batch testwork. A primary grind P_{80} of 100 µm was used followed by sequential lead and zinc flotation.

The lead concentrate was reground to a P_{80} of 43 µm followed by three stages of cleaning with the cleaner tails from each stage reporting to the previous stage in the next cycle. The first cleaner tails were sent to a scavenger stage; with the concentrate going to the lead regrind of the next cycle, and the tails going to the zinc roughers. The zinc concentrate was reground to a P_{80} of 37 µm followed by three stages of the cleaning with the cleaner tails from each stage reporting to the previous stage in the next cycle. The first cleaner tails were sent to a scavenger stage; with the concentrate going to the zinc regrind of the next cycle, and the tails as a final product. At the end of six cycles, the final products from each cycle, and the intermediate streams from the last cycle, were submitted for assay.

Final product assays and weights for the final four cycles of the test were used to generate the metallurgical projection summarized in Table 13-6. The results are consistent with the steep grade-recovery curves observed for both lead and zinc in the open circuit cleaner tests.

Table 13-6 Locked Cycle Test Metallurgical Projection for the SMC

Product	Weight	Assays, %, g/t					Distribution, %				
	%	Cu	Pb	Zn	S	Ag	Cu	Pb	Zn	S	Ag
Pb 3rd Cleaner Con	4.3	3.16	**54.0**	4.84	14.8	**1095**	58.9	**90.6**	7.9	7.7	**73.4**
Zn 3rd Cleaner Con	4.7	1.13	0.45	**42.9**	30.2	91.4	23.1	0.8	**76.7**	17.2	6.7
Zn 1st Cleaner Scav Tail	4.7	0.08	0.35	0.83	6.30	23.7	1.7	0.6	1.5	3.6	1.7
Zn Rougher Tail	86.2	0.04	0.24	0.43	6.95	13.6	16.3	8.0	14.0	71.6	18.1
Head	100.0	0.23	2.58	2.66	8.36	64.6	100.0	100.0	100.0	100.0	100.0

Concentrate Analysis

Final concentrate samples from the SMC locked cycle test were submitted for minor element analysis. A summary of the results is presented in Table 13-7. The analyses indicate the possibility of penalties for bismuth in the lead concentrate and for iron and cadmium in the zinc concentrate.

13.4.5 Concentrate Dewatering Tests

Static settling tests were carried out on lead and zinc concentrate products from the locked cycle testwork. Lead concentrate screening tests indicated favourable settling characteristics using a non-ionic flocculant, Magnafloc 333. Optimization of the feed density and dosage indicated that at 25% density, and 7 g/t of flocculant addition, an underflow density of 76% and a thickener unit area of 0.025 m^2/t/d could be achieved. Similar results were observed for the zinc concentrate. At a diluted feed density of 25% and a Magnafloc 333 addition of 4 g/t an underflow density of 76% and a thickener unit area of 0.029 m^2/t/d was realized.

Table 13-7 Minor Element Assays for the Locked Cycle Test Final Concentrates

Element	LCT-1		Element	LCT-1	
	Pb Conc	Zn Conc		Pb Conc	Zn Conc
Cu %	3.16	1.13	C (t) %	3.18	0.58
Pb %	54.0	0.45	Cl g/t	30	50
Zn %	4.84	42.9	F %	0.014	< 0.005
Ag g/t	1095	91.4	Hg g/t	1.9	3.9
S, %	14.8	30.2	Si g/t	24800	19200
			Acid Insol	10.47	5.47
Al g/t	1920	1550	Li g/t	< 8	< 8
As g/t	15	15	Mn g/t	4330	8530
Bi g/t	1330	15.6	Mo g/t	14	8
Cd g/t	1060	6200	Ni g/t	8	8
Ca g/t	12900	9740	P g/t	< 200	< 200
K g/t	259	202	Sb g/t	291	22.6
Mg g/t	1640	1390	Se g/t	480	24
Na g/t	67	90	Sn g/t	322	148
Ba g/t	27.4	22.6	Sr g/t	19.6	13.8
Be g/t	0.94	0.95	Ti g/t	148	117
Co g/t	3.5	14	Tl g/t	4.7	0.7
Cr g/t	46	86	U g/t	2.8	1.2
Cu g/t	32100	9710	V g/t	7	3
Fe g/t	81900	159000	Y g/t	1.8	1.7

Vacuum filtration tests were conducted on the thickened concentrates which indicated that for both concentrates, at a 21 mm cake thickness and 0.7 bar of vacuum, cake moistures of less than 8% could be achieved. Under these conditions filter capacities were on the order of 870 L/m^2/hr for lead, and 750 L/m^2/hr for zinc.

13.4.6 Variability Composites

In addition to the sub-samples that comprised the master composites discussed earlier, a series of variability composites were generated for variability testing. The material used for these composites consisted of split core samples from the infill drilling program conducted in the summer of 2012. In total, six composites were generated from the sulfide zone. The intersections used in each composite and the calculated heads based on the core assays are presented in Table 13-8.

Table 13-8 Variability Composite Sample Intersections and Calculated Head Grades

Zone/Composite	Hole #	From m	To m	Calculated Head Grade Cu %	Pb %	Zn %	Ag ppm	S %
Upper Sulphide								
US-1	Z-4	146.0	162.5	0.55	2.08	4.62	164	16.2
	Z-7	136.5	168.5					
US-2	Z-4	174.0	180.5	0.43	3.32	6.57	159	14.5
	Z-7	173.5	180.0					
	Z-9	173.0	183.5					
Lower Sulphide								
LS-1	Z-5	184.5	213.0	0.17	2.50	2.35	49.8	6.80
	Z-6	189.0	209.5					
LS-2	Z-4	184.0	212.5	0.27	1.82	3.46	53.5	13.4
	Z-7	187.5	210.0					
LS-3	Z-9	186.0	214.5	0.24	0.36	1.77	46.1	2.27
Basal Sulphide								
BS-1	Z-4	236.0	254.5	0.41	1.57	2.70	91.0	8.66
	Z-5	236.5	267.0					
	Z-9	231.5	244.0					

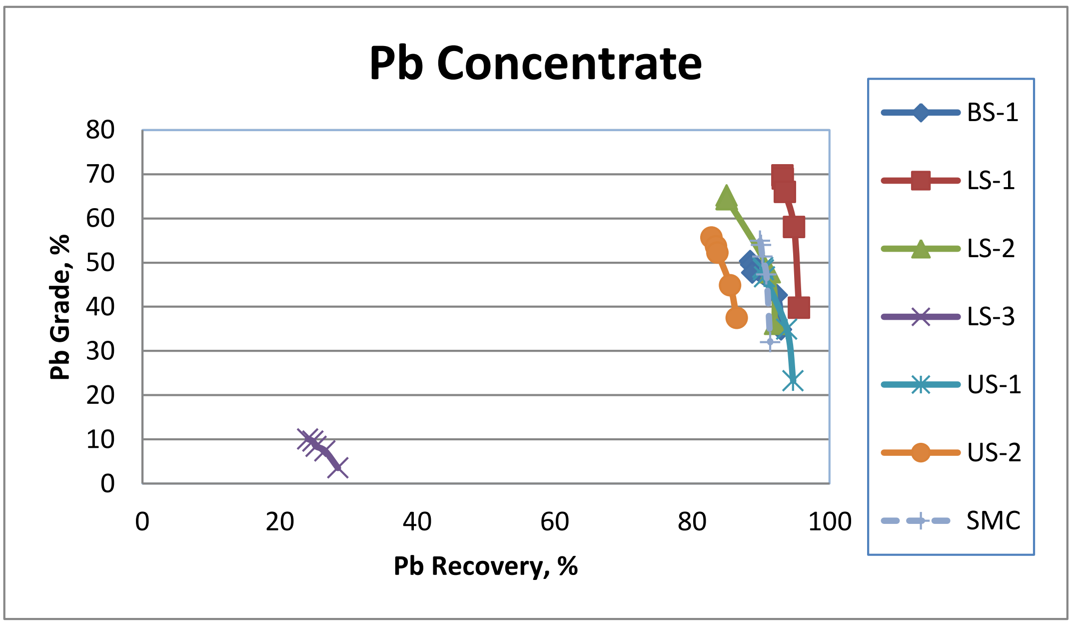
The designated intersections for each composite were crushed down to minus 10 mesh, blended, and split into test charges for flotation work. From the reject fraction a head sample was split out for assay. A summary of the assay results is presented in Table 13-9.

Cleaner flotation tests under the optimized conditions from the SMC testwork were carried out on each of the sulphide variability composites. Grade-recovery curves for lead from these tests are presented in Figure 13-10. The results indicate very similar cleaning characteristics between the Sulphide Master Composite and the variability composites in the lead circuit, with the sole exception of the LS-3 composite. (The low sulfur head grade of the LS-3 composite is more consistent with material from the oxide or transition zone, and the metallurgical performance of the sample supports this.)

Table 13-9 Assay Results for the Variability Composites

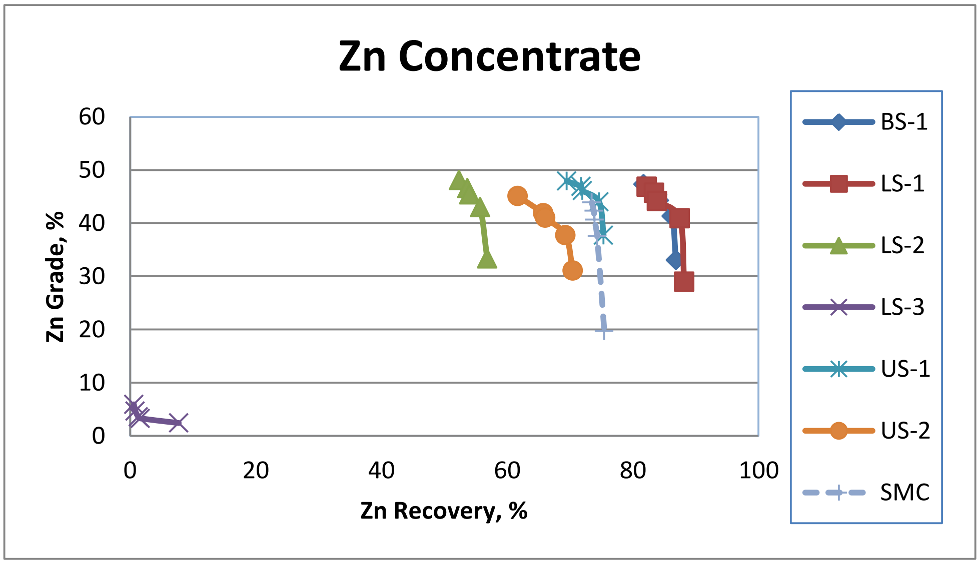
Sample ID	Pb %	Zn %	Cu %	Fe %	S_{TOT} %	Au g/t	Ag g/t
Basal Sulfide 1	1.50	2.59	0.35	19.1	8.29	0.03	96.3
Lower Sulfide 1	2.17	2.43	0.14	13.5	6.16	< 0.02	44.6
Lower Sulfide 2	2.06	3.30	0.25	27.0	14.60	0.19	52.6
Lower Sulfide 3	0.35	1.84	0.17	2.0	2.13	0.05	42.8
Upper Sulfide 1	1.64	4.51	0.41	24.8	16.20	0.03	146
Upper Sulfide 2	3.51	7.28	0.44	27.6	15.80	< 0.02	192

Figure 13-10 Lead Grade-Recovery Curves for Variability Cleaner Tests



Zinc grade-recovery curves for the cleaner variability composites are presented in Figure 13-11. Again, with the exception of LS-3, the curves are all comparable to that for the SMC. Final concentrate grades for the variability composites ranged from 45.1% to 46.9%, consistent with the results of the earlier mineralogical and flotation testwork, which indicated that the sphalerite/marmatite mineral has a fixed grade limit.

Figure 13-11 Zinc Grade-Recovery Curves for Variability Cleaner Tests



13.5 Metallurgical Projection

A metallurgical projection was developed based on the testwork results presented in the previous sections (see Table 13-10). The primary basis for this estimate is the results of LCT-1 which demonstrated the effects of recycle streams on the concentrate grades and recoveries. Head grades are based on the 2010 Datamine model generated by Rick Parker at a zinc equivalent cutoff grade of 3%.

Table 13-10 Metallurgical Projection for the Bilbao Sulfide Deposit

Product	Weight	Assays, %, g/t					% Distribution				
	%	Cu	Pb	Zn	S	Ag	Cu	Pb	Zn	S	Ag
Pb 3rd Cleaner Con	3.39	3.24	**54.0**	4.84	14.8	**1335**	52.3	**90.6**	5.7	7.7	**73.4**
Zn 3rd Cleaner Con	5.15	1.13	0.33	**43.0**	30.2	81.0	27.6	0.8	**76.7**	17.2	6.7
Zn 1st Cleaner Scav Tail	4.73	0.08	0.28	0.95	6.30	22.8	1.9	0.6	1.6	3.6	1.7
Zn Rougher Tail	86.7	0.04	0.19	0.53	6.95	12.8	18.2	8.0	16.1	71.6	18.1
Head	100.0	0.21	2.02	2.89	8.38	61.6	100.0	100.0	100.0	100.0	100.0

13.6 Conclusions and Recommendations

From the metallurgical testwork conducted on sulfide and transition samples from the Bilbao deposit, the following conclusions are drawn:

- Quantitative mineralogy by QEMScan identified that the lead was present predominantly as galena. Zinc was found to be present as sphalerite/marmatite with a mineral zinc grade of ~49%.
- Grindability testing consisting of Bond Ball Work Index, Bond Rod Work Index, Abrasion Index, and SMC tests was carried out on the sub-composites and indicated that the material was of average hardness with BBWI values ranging from 11.8 to 16.2 kWh/t.
- Conventional sequential lead-zinc flotation was found to result in optimum rougher circuit recoveries at a moderate grind size P80 of 100 µm.
- Upgrading of the rougher lead concentrate was achieved by regrinding to a P80 of ~45 µm and three stages of cleaning at pH 8-10.
- Upgrading of the rougher zinc concentrate was achieved by regrinding to a P80 of ~40 µm and three stages of cleaning at pH 10-11.
- Locked cycle testing of the SMC achieved good stability after six cycles of flotation and produced acceptable concentrate grades at lead and zinc recoveries of 90.8% and 76.7%, respectively. Silver recovery to the lead concentrate was 73.4%.
- Minor element analysis of the locked cycle test concentrates indicate the possibility of penalties for bismuth in the lead concentrate and for iron and cadmium in the zinc concentrate.

Recommendations for future work on samples from the Bilbao deposit include:

- Further scoping level testwork, mineralogy, and flowsheet development on composite and variability samples from the oxide zone to identify the potential for additional economic recovery of metal values. Bilbao contains a substantial in-situ oxide resource of 3.8 million tonnes (3 million inferred and 791,000 indicated) at a Zn equivalent grade of 6.5%, and opportunities may include new technologies for leaching and gravity recovery, or high-grading of the oxide zone to focus specifically on the zinc and/or silver minerals.
- Additional variability testing of samples from the transition zone to better characterise the extent of float recoverable mineralization in this area.
- Mineralogical characterisation of transition zone samples from different drill holes to develop correlations between lead and zinc deportment and core log data.
- Mineralogical characterisation of sulphide variability composite LS-3 to compare with the results of the transition zone samples and determine if this sample represents another area of altered material.

14. Mineral Resource Estimates

This resource model is an update of previous models incorporating twenty holes drilled during 2011 and 2013, which completed a total of 105 drill holes in the deposit. A lithology model was built and Indicator and Ordinary Kriging (OK) were used to estimate Zn, Pb, Ag and Cu resources. Density was updated using 224 new density determinations completed since the last 2010 model was constructed. The previous 2010 model (revised in 2011) had assigned a density of 3.6 g/cc to sulphide blocks based on the average of 14 measurements.

Previous resource models have been completed for Xtierra at Bilbao by Parker beginning in 2007. The last resources model completed was in 2011 and included 84 drill holes. The previous resource estimation was originally carried out by Bilbao geologists along with a modeling consultant and QP, Richard Parker Consulting Geologist. Lithology and a 3% equivalent zinc (Zneq) grade shell were the geological constraints used to complete an inverse distance estimation of Ag, Pb, Zn and Cu resources.

The last resources model completed was in 2011 and included 84 drill holes. The resources (including both oxide and sulphide) reported in 2011 are 10,617,891 tonnes @ 6.48% Zneq in the indicated category and 430,000 tonnes @ 5.19% Zneq in the inferred category, based on an estimation distance of 40 m.

In 2013, RungePincockMinarco (RPM) in Lakewood, Colorado was contracted to complete a resource estimation. RPM completed the wireframes, statistical analysis, block modeling, and mineral resource estimation. This resource model will be used in a Preliminary Economic Assessment (PEA).

For this study, the geological model was completed by Bilbao geologists based on core logging, surface geological mapping, and interpretation of geology on cross-sections spaced 50 meters apart. RPM created a lithological wireframe and 1% Zn, 1% Pb and 25 ppm Ag indicator kriging (IK) envelopes and Ag/Pb/Zn/Cu Ordinary Kriging estimation was executed to complete the block model.

The lithology model was initially created on paper cross sections, on generally 50m centres. This model is being used in the resources estimation for the first time. Using Vulcan, three-dimensional solids were generated, and then verified visually against the original data. The solid was then used in the block model construction where resource estimates were calculated.

The resource model for the Bilbao Deposit is based on assays from diamond drilling. The model has no constraints other than the surveyed topography.

This model provides the basis for the mineral resource estimates discussed herein.

Information contained in this report is based on information provided by Xtierra. RPM believes the resource estimates meet industry standards and the general guidelines for NI 43-101 compliant resources for Indicated and Inferred confidence levels as discussed herein.

14.1 Topography

For the Bilbao deposit, the topography was provided by Xtierra. This data was checked against the drill hole collar surveys and showed good correlation to the topographical surface. RPM generated a three dimensional surface using the same database and the result was similar. The Bilbao topographic surfaces were used to represent topography during the modeling process.

14.2 Modeling and Estimation

This Zn/Pb/Ag/Cu estimation of the Bilbao Deposit for Xtierra. in Zacatecas, Mexico, was executed by RPM to incorporate new drilling information acquired during 2011-2013.

The scope of this estimation started with compositing and ended with resources classification. Database and QAQC of 2011-2013 campaigns were checked by RPM. The historical database was assumed accurate.

The effective date for this resource estimate is July 24, 2013.

14.2.1 Geology and Modeling

The Bilbao Deposit is a contact metamorphic deposit, classified as skarn type. It is developed in the marbleized limestone at the contact with the La Blanca granodiorite (granite). The mineralization occurs as sulphide replacement bodies forming along the bedding in the limestone (mantos) and as minor replacement bodies in the intrusive (endoskarn). The highest grades are found in contact with the main intrusive body. Grades sharply decrease from the main intrusive body toward the west.

Previous resource estimates were based on a 3% Zneq iso-grade envelope which was interpreted using 50 m spaced sections. The 3% Zneq wireframe covered the highest grade zone and peripheral ore bodies. Where mineralized lenses do not continue onto adjacent east-west sections they were assumed to extend half way to the next section (approximately 25 m) and zones intersected on the margins of the deposit were projected for 12.5 m beyond the marginal drill-hole.

RPM reviewed the Ag/Pb/Zn grade distributions (Table 14-1) and found a strong lithological control on their localization. As well, RPM detected medium to low Ag/Pb, Ag/Zn and Pb/Zn correlations (Table 14-2). RPM concluded that Ag/Pb/Zn behaviors match in general but the zonations are not exactly the same. In a visual examination, RPM observed that highest Ag values extend shorter than highest Pb values and in turn, highest Pb values extend shorter than highest Zn values from the granite toward west. Zonation is typical in skarn deposits and Bilbao appears to demonstrate a typical skarn replacement zonation pattern.

Table 14-1 Statistics of 1 m Length Composites

	Litho	Basalt	Limestone	Sandstone	San	ExoSkarn	Granite	Vein	Fault
	N	58	7179	61	27	99	5816	5	4
Ag	Min	0.02	0.0	0.0	1.5	0.1	0.0	5.0	27.3
	Q1	0.5	1.4	0.4	1.5	2.5	0.6	5.0	27.3
	Median	1.5	4.0	1.5	1.5	12.0	1.5	20.5	27.3
	Q3	1.5	22.0	3.3	4.0	37.6	4.0	50.9	27.3
	Max	1.5	1434.5	43.0	8.0	269.0	2291.3	69.0	27.3
	Mean	1.2	25.0	4.9	2.8	32.4	8.0	28.1	27.3
	S.D	0.6	60.0	9.3	1.8	56.1	55.7	23.5	0.0
	CV	0.50	2.39	1.89	0.62	1.73	6.93	0.84	0.00
Pb	Min	0.000	0.000	0.000	0.020	0.000	0.000	0.100	4.80
	Q1	0.002	0.010	0.009	0.040	0.030	0.000	0.100	4.80
	Median	0.013	0.040	0.030	0.074	0.150	0.010	0.155	4.80
	Q3	0.020	0.495	0.090	0.108	0.707	0.038	0.308	4.80
	Max	0.059	39.375	1.450	0.620	8.900	7.562	0.490	4.80
	Mean	0.014	0.748	0.114	0.166	0.671	0.088	0.230	4.80
	S.D	0.01	1.80	0.28	0.22	1.37	0.41	0.16	0.00
	CV	0.88	2.41	2.47	1.31	2.05	4.68	0.70	0.00
Zn	Min	0.000	0.000	0.000	0.010	0.010	0.000	0.030	2.50
	Q1	0.010	0.010	0.010	0.110	0.030	0.010	0.030	2.50
	Median	0.013	0.048	0.020	0.120	0.213	0.030	0.136	2.50
	Q3	0.020	0.440	0.063	0.148	0.998	0.110	0.408	2.50
	Max	0.046	27.229	0.670	2.160	22.400	9.176	0.690	2.50
	Mean	0.015	0.781	0.062	0.485	1.461	0.183	0.225	2.50
	S.D	0.01	1.89	0.11	0.80	4.03	0.55	0.22	0.00
	CV	0.52	2.43	1.74	1.65	2.76	3.00	0.96	0.00

Table 14-2 Correlations Coefficients of 1 m Length Composites by Lithology

Litho	N	Ag/Pb	Ag/Zn	Pb/Zn
All	13553	0.30	0.27	0.51
AlluvSoil	0	-		
Basalt	58	0.68	0.46	0.39
Limestone	7179	0.43	0.39	0.50
Sandstone	61	0.85	0.45	0.47
Rhyollite	0	-	-	-
San	27	0.25	0.31	0.99
ExoSkarn	99	0.28	0.12	0.37
Granite	5816	0.28	0.21	0.58
Vein	5	0.12	0.99	0.07
Fault	4	-	-	-

To address the variable distributions of Ag, Pb, and Zn and their relation to the granite – limestone contact, RPM built a wireframe of the granite. RPM used the technique of creating surface increments until completing a closed solid. The final solid created was a “boolean” surface. The solid was visually checked and the original logged interval codes were compared to the granite solid, which is called back tag analysis. The back tag analysis showed the database has total length intervals of granite of 5,672.8 m and 5,502.1 m (97%) were inside the solid. The solid has 198 m of rock classified as limestone (3.3%). The granite database proportion inside the solid is well over the minimum 90 percent accepted by mining industry.

Indicator Kriging estimations (IK) were applied to create solids to constrain each of the highly mineralized limestone zones. The selected IK’s thresholds were 1% Zn, 1% Pb and 25 ppm Ag. Seven zones were created. They are:

1. The granite domain
2. Ag core domain (Ag≥ 25 ppm-domain Ag25)
3. Remaining Ag limestone domain (Ag<25 ppm – domain Ag0)
4. Pb core domain (Pb≥1% - domain Pb1)
5. Remaining Pb limestone domain (Pb < 1% - domain Pb0)
6. Zn core domain (Zn≥ 1%- domain Zn1)
7. Remaining Zn limestone domain (Zn<1% - domain Zn0)

The report will use the nomenclature of “core domains”, “domains 0” and “granite” to simplify the discussion taking account that core domains and domains 0 are different for Ag, Pb and Zn.

Table 14-3 shows the indicator variogram model for Ag, Pb and Zn. The estimations for the binomial distributions were executed by OK using variogram ranges as search distances. Twelve and one hundred twenty composites were defined as minimum and maximum to estimate the probability, fifteen maximum composites by octant and at least two drill holes were required to estimate a block. Finally, the final high grade domains were populated with the blocks with a greater than fifty percent probability removing the smallest non-continuous bodies.

Mineral zones were defined using the surfaces provided by the project. Surfaces were extended to the margins of the model and then they were cut by the Boolean surfaces representing topography and the contacts of oxide and sulphide mineral surfaces.

Table 14-3 Indicator Variogram Models

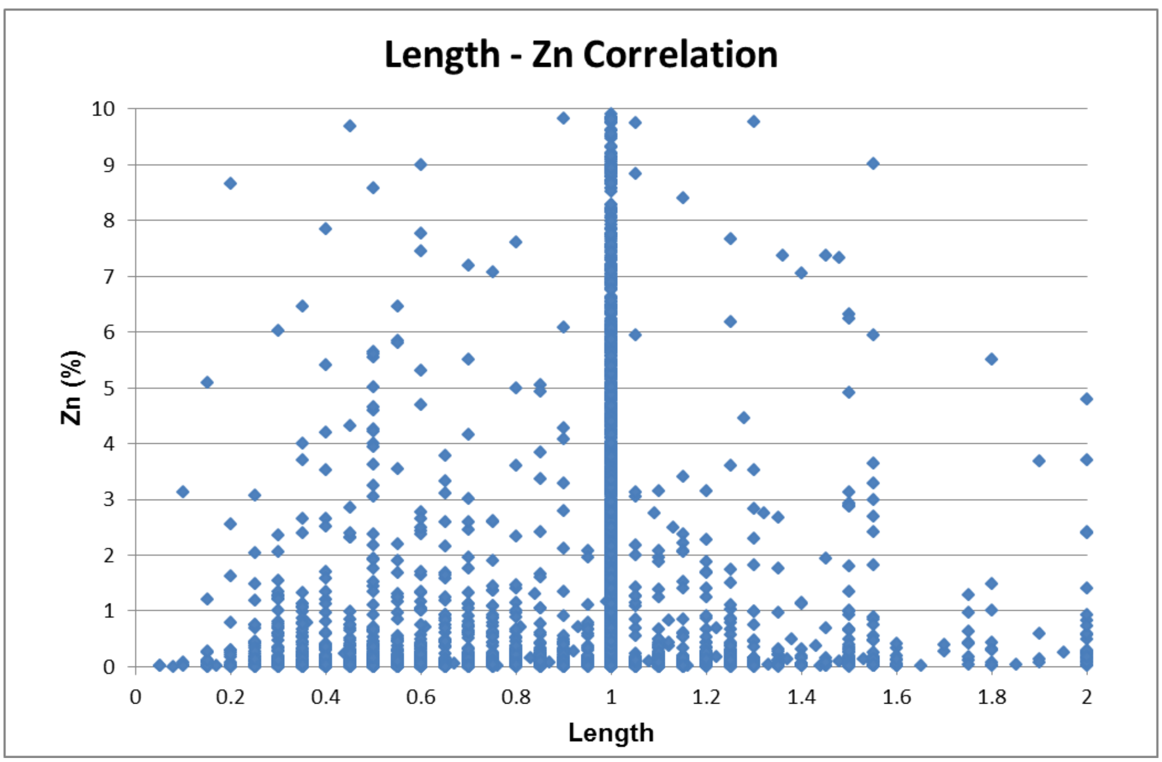
Variable		Ag (ppm)	Pb (%)	Zn (%)
Treshold		25	1	1
Bearing/Plunge/Dip		195/-30/-15	195/-30/-15	195/-30/-15
Nugget Effect (C0)		0.05	0.045	0.035
1st Structure	type	SPHERICAL	EXPONENTIAL	SPHERICAL
	C1	0.079	0.08	0.054
	Major	15	80	40
	Semi	15	10	30
	Minor	35	80	35
2nd Structure	type	SPHERICAL	EXPONENTIAL	SPHERICAL
	C2	0.05	0.03	0.06
	Major	130	130	90
	Semi	90	90	90
	Minor	35	80	35

14.2.2 Exploration Data Analysis

The samples database contains 15,450 samples with Ag values and 15,554 samples with Pb/Zn/Cu values. Sample lengths vary from few centimeters to five meters but eighty-three percent of lengths correspond to 1 m and 93% are lesser than or equal to 1 m. Figure 10-3 shows the sample length distribution.

Figure 14-1 shows that sample lengths and Zn grade have no correlation.

Figure 14-1 Lengths (m) versus Zn (%)



One meter composites were generated to explore the Ag/Pb/Zn behaviours defining the original variability after generating the final composite length.

Table 14-1 shows the statistics of 1 m length composites from the logged lithology. Limestone and granite make up 98% of the composites. Both lithologies have Ag/Pb/Zn coefficient variation greater than 2 which implies that non-lineal estimation would work better than the traditional simple and OK.

Correlation among Ag, Pb and Zn by lithology is shown in the Table 14-2. Ag-Pb/Ag-Zn correlations are 0.43/0.39 in limestone and 0.28/0.21 in granite. In turn, Pb-Zn correlations vary from 0.50 in limestone to 0.58 in granite. These correlations are considered low to medium and they indicate the way to estimate the metals is by defining individual domains for each.

Two meter composites were generated to estimate Ag/Pb/Zn indicator domains and grades using a 2 m height block size. IK was carried out using composites of 2 meters constant length to envelop the core domains higher than 1% Zn, 1% Pb and 25ppm Ag (see Section 14.2.2).

The statistics by domains are shown in the Table 14-4.

Table 14-4 Two Meter Composite Statistics.

	Ag				Pb				Zn			
	All	0	Gra	Ind	All	0	Gra	Ind	All	0	Gra	Ind
N	6835	3068	3128	639	6835	3175	3128	532	6835	3128	3128	579
Mean	16.91	11.45	8.42	85	0.438	0.305	0.104	3.19	0.507	0.228	0.213	3.60
Minimum	0	0	0	1.5	0	0	0	0.005	0	0	0	0.02
Maximum	1925	722	1925	985	34.94	17.3	7.23	34.94	26	10.42	22.2	26
Variance	2915	1106	2971	6242	1.70	0.87	0.21	7.08	2.08	0.49	0.647	8.02
Q1	1	1.05	0.51	37	0.005	0.005	0	1.47	0.01	0.01	0.01	1.66
Median	2	2.55	1.5	62	0.02	0.025	0.01	2.60	0.039	0.03	0.03	2.73
Q3	11	10.5	4	111	0.145	0.17	0.04	4.17	0.228	0.14	0.12	4.84
CV	3.2	2.9	6.5	0.9	3.0	3.1	4.4	0.8	2.8	3.1	3.8	0.8

Ag, Pb and Zn distributions have remarkable differences between the core domains and the surrounding 0 domains and granite domains. Coefficients of variations drop under 1 in the core domains and the Ag/Pb/Zn means drop ten to thirty times from cores to granite and 0 domains.

Figures Figure 14-2 through Figure 14-4 show the Ag, Pb and Zn distributions.

Contact profiles (Figure 14-5) were created to analyze the Ag, Pb and Zn grades contact behaviours and defining the samples configuration during the estimation. The three elements have a hard contact between the core domains and 0 and granite domains. Grades jump from 0.25% to 0.8% for Zn grades, 0.16% to 0.6% for Pb grades and 10 ppm to 25 ppm for Ag grades from the 0 and granite to core domains. These sharp jumps reinforce the decision about separating the high Ag/Pb/Zn grade zones.

RPM defined capping values by checking the log histograms to define the probability of high tails (Figures Figure 14-2 through Figure 14-4), determining the outlier's threshold (inflexions in the lineal distributions) and looking over the neighborhood of the highest grades (spatial examination) to determinate the correlation among them.

Figure 14-2 Ag Distributions by Core (pink), 0 (blue), and Granite (red) Domains

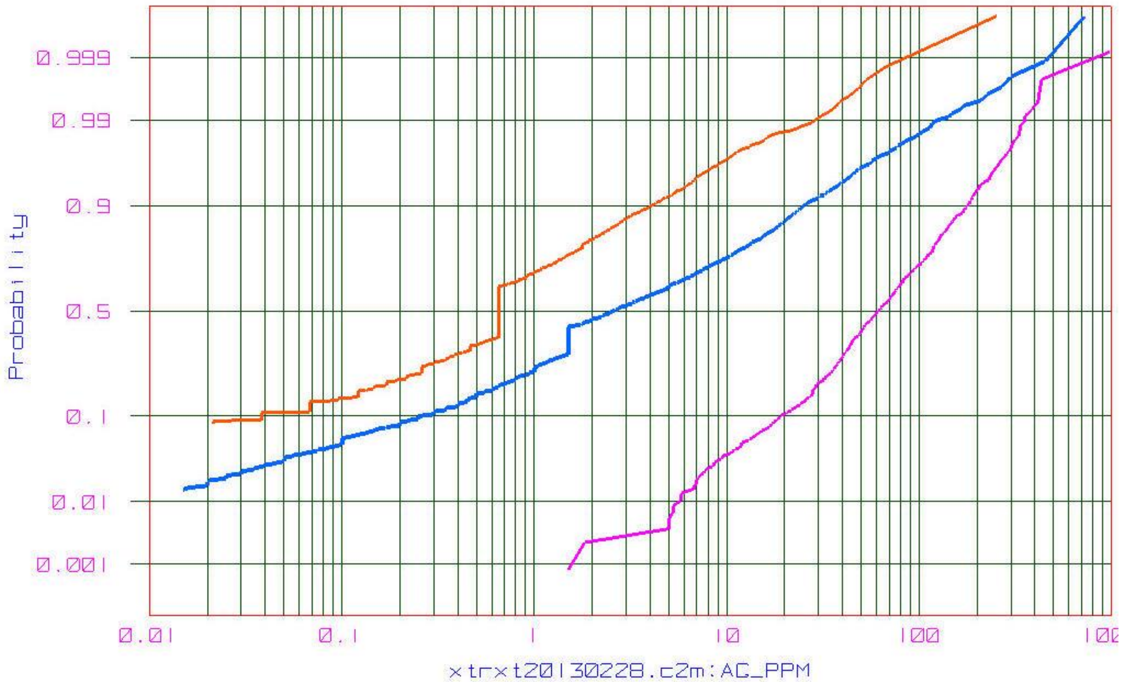


Figure 14-3 Pb Distributions by Core (pink), 0 (blue), and Granite (red) Domains

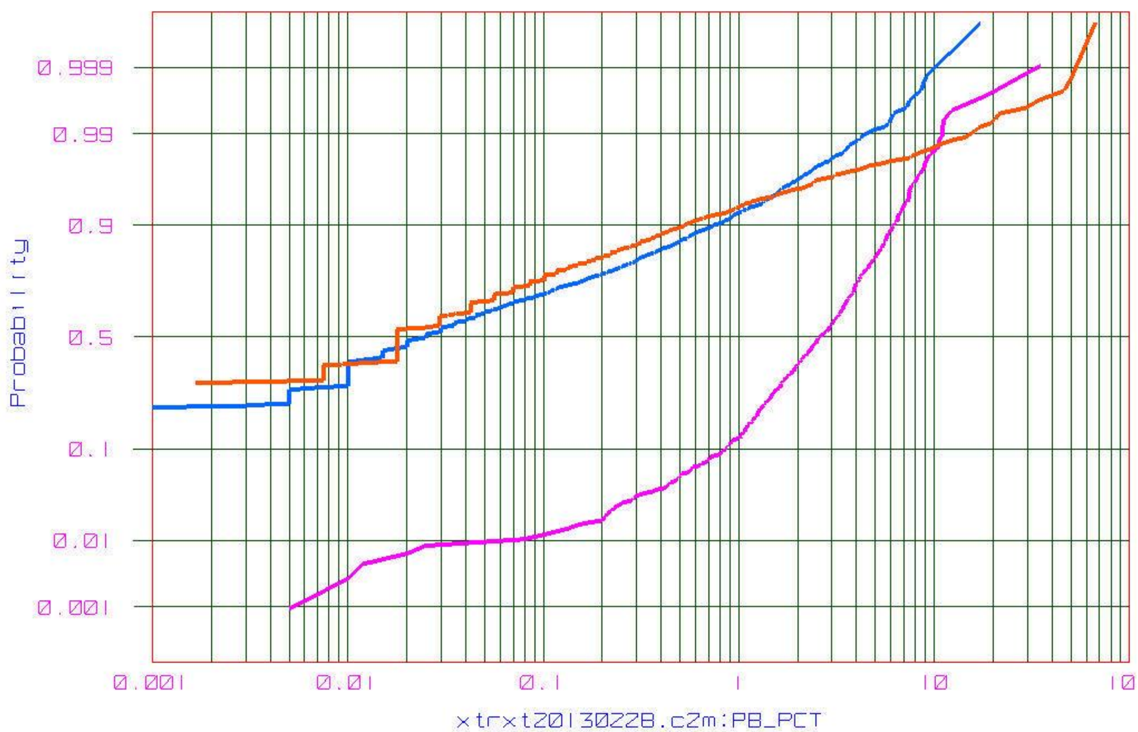


Figure 14-4 Zn Distributions by Core (pink), 0 (blue), and Granite (red) Domains

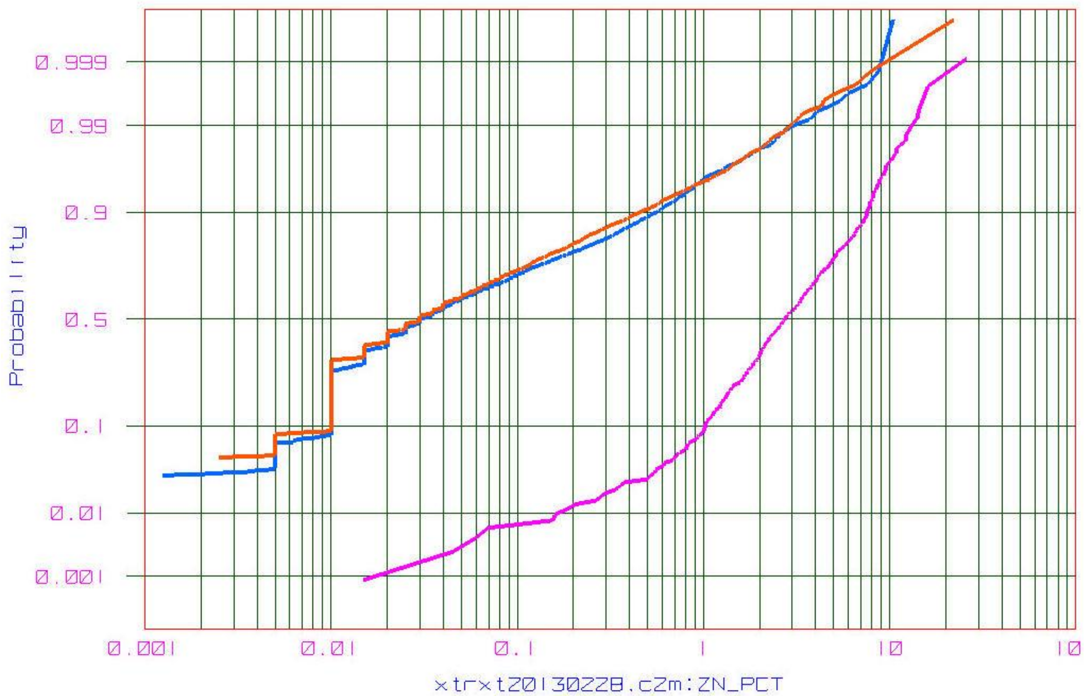


Figure 14-5 Contact Profiles

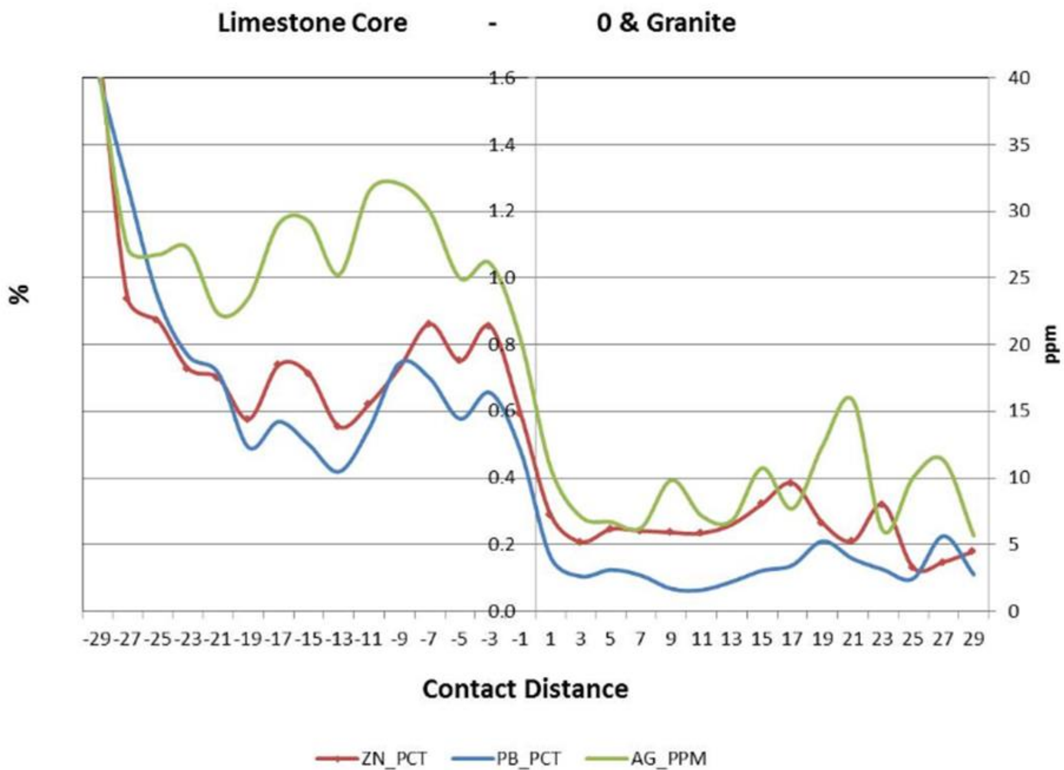


Table 14-5 summarizes the capping values definition (black numbers) and exclusion high grade distance values (blue numbers). The Ag core has a high tail greater than 450 ppm Ag (see pink line in Figure 14-2). The Pb core has a high tail greater than 11 % Pb (see pink line in Figure 14-3), and Zn core has a high tail greater than 15 % Zn (see pink line in Figure 14-4). Capping was applied on core domains to the above values. Zero domains have outliers of 450 ppm Ag (see blue line in Figure 14-2), 6 % Pb (see blue line in Figure 14-3) and 8 % Zn (see blue line in Figure 14-4). Capping was applied on the 0 domains to the above values. The granite domain has a lineal Ag and Zn high tails (see red line in Figure 14-2 and Figure 14-4). The value greater than 400 ppm Ag and 3% Zn were constrained to a single block (5m by 5m by 2m). The granite domain has an inflexion at 2.5 % Pb (see red line in Figure 14-3), capping was applied in this case.

Table 14-5 Outlier Summary

Outliers	Core	0	Granite
Ag (ppm)	450	450	400
Pb (%)	11	6	2.5
Zn (%)	15	8	3

Capping was applied to 2 m length composites slightly reducing the mean and the variance in the domains.

14.2.3 Variography

For the nine domains, directional correlograms of Ag, Pb and Zn were built using twenty five meter lag size and 22.5° cone angle. Directions were vertically and horizontally explored in 15° steps. Poor continuity was observed in all directions.

The correlograms were fitted taking the nugget effect from the down hole correlograms and due to the quite poor continuity found in directional correlograms, two structures were defined based on omnidirectional or omnihorizontal correlograms. Table 14-6 summarizes the correlogram models. Nugget effects vary from 0.2 to 0.65 and ranges of the first structures reach 10 to 25 m. Total ranges reach 100 – 120 m when the second structure is a small proportion of the sill. Correlograms of the core domains are shown in Figures Figure 14-6 through Figure 14-8.

Table 14-6 Correlogram Models

Variable		Ag			Pb			Zn		
Domain		0	1	Gra	0	25	Gra	0	1	Gra
Bearing/Plunge/Dip		285/-30/15	Hor / -90	Omni	Hor / -90	Omni	Hor / -90	Hor / -90	Hor / -90	0
Nugget Effect (C0)		0.5	0.65	0.6	0.2	0.5	0.2	0.45	0.32	0.2
1st Structure	type	SPHERICAL	SPHERICAL	SPHERICAL	SPHERICAL	SPHERICAL	SPHERICAL	SPHERICAL	SPHERICAL	SPHERICAL
	C1	0.3	0.35	0.3	0.65	0.5	0.3	0.4	0.55	0.5
	Major	30	25	10	10	10	10	5	10	10
	Semi	25	25	10	10	10	10	5	10	10
	Minor	10	15	10	10	10	10	5	10	30
2nd Structure	type	SPHERICAL		SPHERICAL	SPHERICAL		Exponential	SPHERICAL	SPHERICAL	SPHERICAL
	C2	0.2		0.1	0.15		0.5	0.15	0.13	0.3
	Major	60		50	100		50	100	70	70
	Semi	25		50	100		50	100	70	70
	Minor	10		50	60		40	40	10	120

Figure 14-6 Ag Core Correlogram

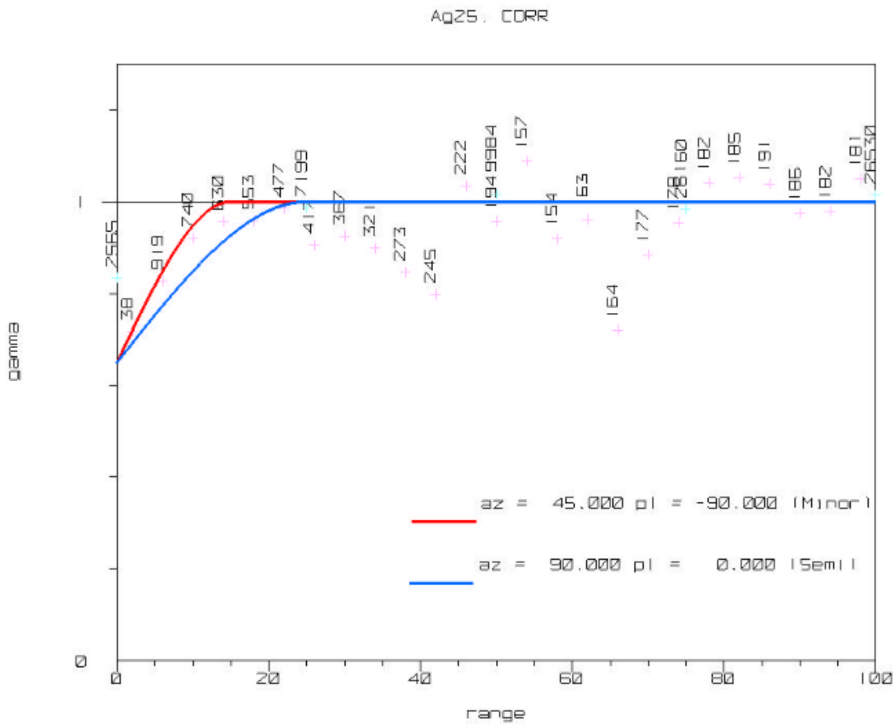


Figure 14-7 Pb Core Correlogram

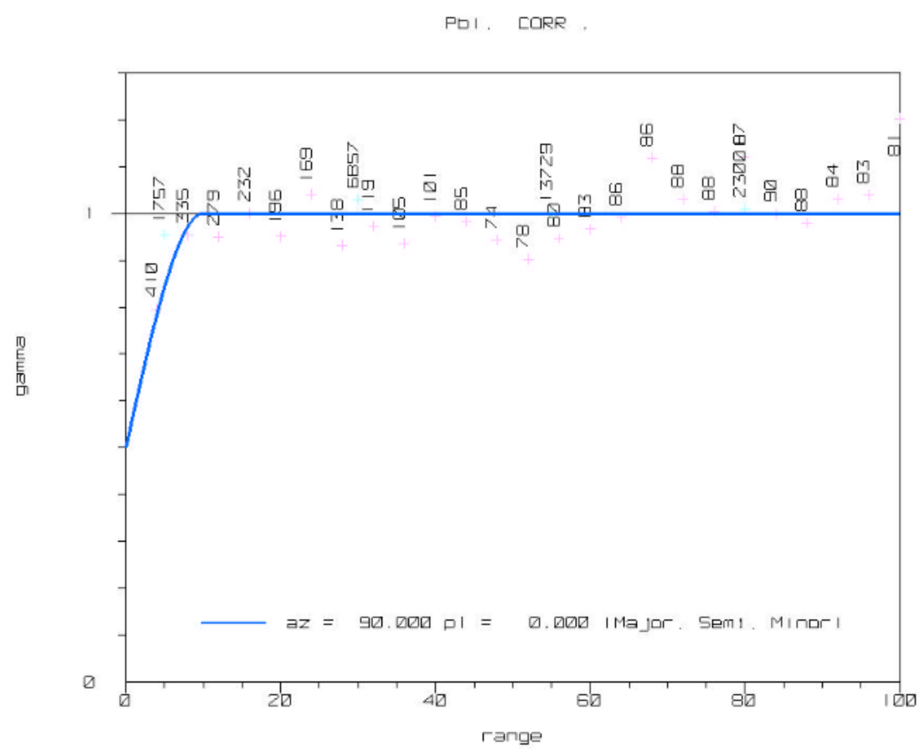
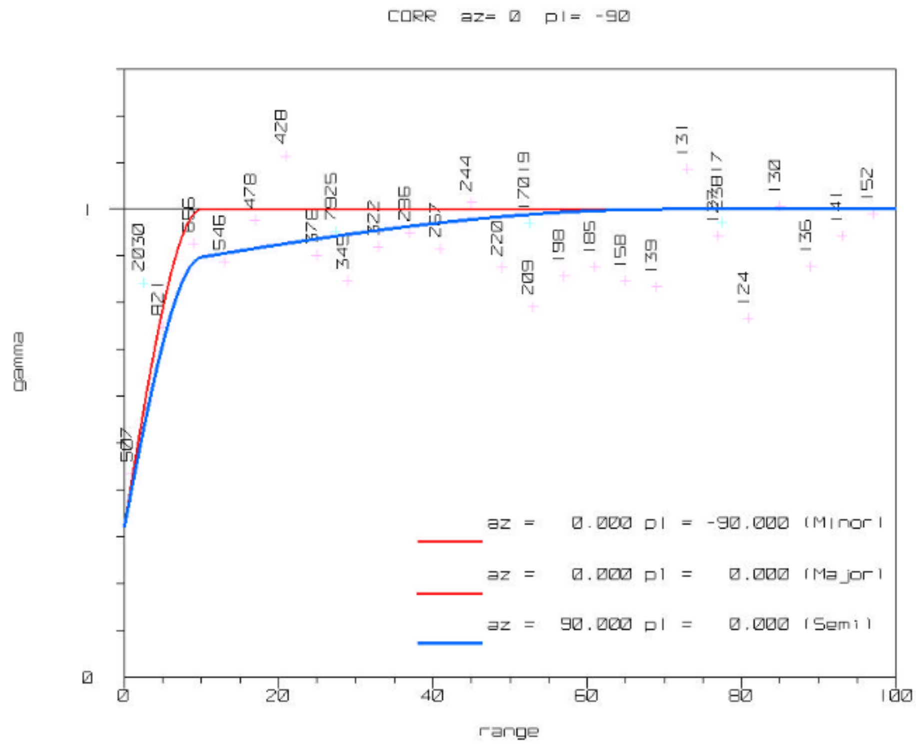


Figure 14-8 Zn Core Correlogram



Because this low continuity, the OK estimation will tend to be smooth and with high estimation variance which would suggest that the current drill hole grid spacing is still too large to get an accurate local estimation. This has a strong implication in resources classification.

14.2.4 Block Model

A sub-block model from 2.5 by 2.5 by 1 m to 5 by 5 by 2 m block sizes was built using the lithological wireframe. The block model was orientated 90°. The origin and dimensions of the block model are shown in Table 14-7.

Table 14-7 Block Model Parameters

		X	Y	Z
Offset	minimum	2508487.5	792887.5	1800
	maximum	2508982.5	793442.5	2200
Blocks	minimum	2.5	2.5	1
	maximum	5	5	2
No of 5x5x2 blocks		99	111	200

Domains were coded in the variables domag, dompb and domzn as seen in Table 14-8. The final mineral zone (minzon) in the sub-block model was assigned by the mineral zone solids code.

Table 14-8 Domain Codes

	Variable	Unit	Code
Ag	domag	Core	1
		Rest	0
Pb	dompb	Core	1
		Rest	0
Zn	domzn	Core	1
		Rest	0
Litho	litho	Core	10
		Rest	20
Mineral Zone	minzon	Oxide	1
		Transitional	2
		Sulphide	3

14.2.5 Density

In the previous 2010 block model, densities were established as 2.7, 3.1 and 3.6 for oxide, transitional and sulphide zones, respectively. These values were based on the few determinations available at that time (see Table 14-9).

Table 14-9 Densities

	In-house & AlS Densities			
Zone	N	mean	min	max
Oxide	72	2.744	2.04	3.28
Tansitional	12	2.835	2.44	3.44
Sulphide	163	3.250	2.25	3.9

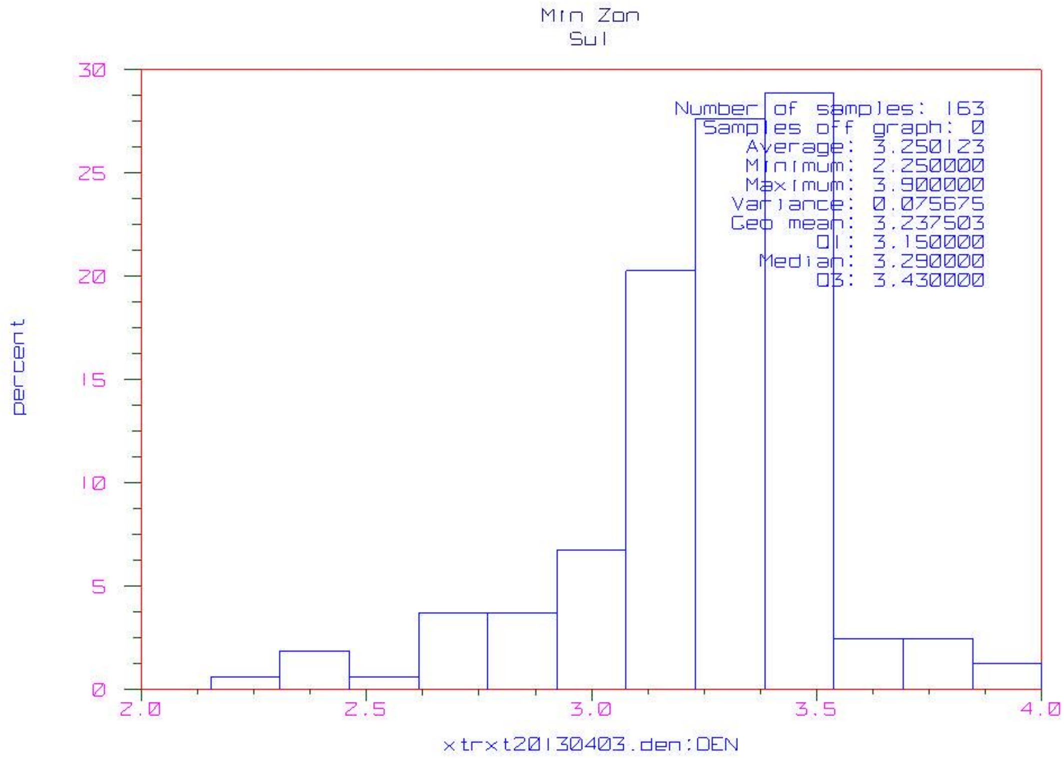
Current density measurements include 92 in-house determinations and 155 ALS determinations (Table 14-9). Xtierra defined the mean density by mineral zone filtering the lower and higher intervals, and the intervals with low numbers of determinations. Xtierra concluded that the core zone density mean is 2.84, 3.04 and 3.46 g/cc in oxide, transitional and sulphide zones, respectively. RPM would consider only dismissing some determinations based on geological or sampling reasons. According of the whole density database, the mean of the sulphide zone has a difference of -6% of the above value (3.46 v 3.25 g/cc.)

Figure 14-9 shows a histogram for densities inside the sulphide zone.

RPM decided to estimate densities using inverse distance with power 1 based on the mineral zones, this way preserving the local density mean.

RPM modeled the density by applying three inverse distance estimation passes. The first pass used a searching ellipsoid of 100 by 100 by 25 m to find at least 4 composites with measured densities from 2 drill-holes and 3 octants. The second pass relaxed the searching distance to 200 by 200 by 25 m and third pass additionally relaxed the drill-holes and octants limits. Non-estimated core blocks were filling in with the Xtierra mean of 2.84, 3.04 and 3.46 g/cc. (Table 14-9). Remaining non-estimated blocks of limestone were filled with 2.57 and granite density was assigned in 2.52 g/cc.

Figure 14-9 Density Determinations in Sulphide Zone



14.2.6 Estimation

Block Ag/Pb/Zn/Cu grades were estimated by OK in Vulcan. Given a poor continuity Ag/Pb/Zn model a standard sampling configuration was defined to estimate the blocks. A second pass was carried out to fill the non-estimated remaining blocks. After that, the mean domain value was assigned to non-estimated blocks in each domain.

The first pass sampling configuration consisted of four minimum and twelve maximum 2 m length composites, four composites maximum by octant and at least two drill holes. One hundred percent of the variogram range was used as searching distances. The first pass estimated 92% of Ag core domain, 72% of Pb core domain and 89% of Zn core domain. The second pass consisted in the mean of the eight composites from the four nearest drill holes. The second pass increased to 97% the estimated blocks in Ag core domain, to 86% in Pb core domain and 92% in Zn core domain. Most of the non-estimated blocks are on the edge of the core body where there is only one drill hole.

Capping and outlier distance restrictions were applied for each domain according to Table 14-10.

Section and level maps were reviewed to confirm that the estimated block grades are in close conformity to the composite grades inside each geological zone. No artifacts were detected in the domains and therefore, RPM confirmed that the grade interpolations are reasonable.

Nearest neighbor estimation (NN) was implemented using the same searching ellipsoid as used for OK. It used one 2.0 m length composite to estimate 1 and 2 m height blocks. In turn, inverse distance power 1 (ID) was

Table 14-10 Ordinary Kriging Plans

Variable	Domain	Pass	Orientation	Major	Semi	Minor	Min - Max Samples		Max/Oct	Max/Dh	Capping	Excl. Dist.	
Ag	0	1	285/-30/15	60	30	10	4	12	4	3	450		
		2	285/-30/15	90	45	15	4	12	4	3	450		
	Core	1	0/0/0	50	50	30	4	12	4	3	450		
		2	0/0/0	100	100	30	8	8		2	450		
	Gra	1	0/0/0	50	50	50	4	12	4	3		400	5-5-2
		2	0/0/0	100	100	50	8	8		2		400	5-5-2
Pb	0	1	0/0/0	100	100	60	4	12	4	3	8		
	Core	1	0/0/0	50	50	10	4	12	4	3	12		
		2	0/0/0	100	100	10	8	8		2	12		
	Gra	1	0/0/0	50	50	40	4	12	4	3	2.5		
		2	0/0/0	100	100	40	8	8		2	2.5		
Zn/Cu	0	1	0/0/0	100	100	40	4	12	4	3	8		
	Core	1	0/0/0	70	70	10	4	12	4	3	15		
		2	0/0/0	120	120	10	8	8	4	2	15		
	Gra	1	0/0/0	70	70	70	4	12	4	3		3	5-5-2
		2	0/0/0	120	120	70	8	8		2		3	5-5-2

executed with the same sample configuration as was used by OK. These two methods were used to validate OK estimation for local and global bias through swath plots. Figures Figure 14-10 through Figure 14-15 compare these estimations in the core, 0 and granite units in the first pass. NN is more variable than ID and OK as was expected. ID and OK are quite close due to the poor continuity of grades and then, OK weights are smoother than the distance weights. Beyond the variogram ranges (second pass) OK estimates the local mean while the ID continues weighting the nearest composites, but the low maximum composites used (12 composites) avoids over-smooth by Kriging.

We can conclude that OK is acceptable for local bias and smoothing in the three domains.

Figure 14-10 Swath Plot Ag Core

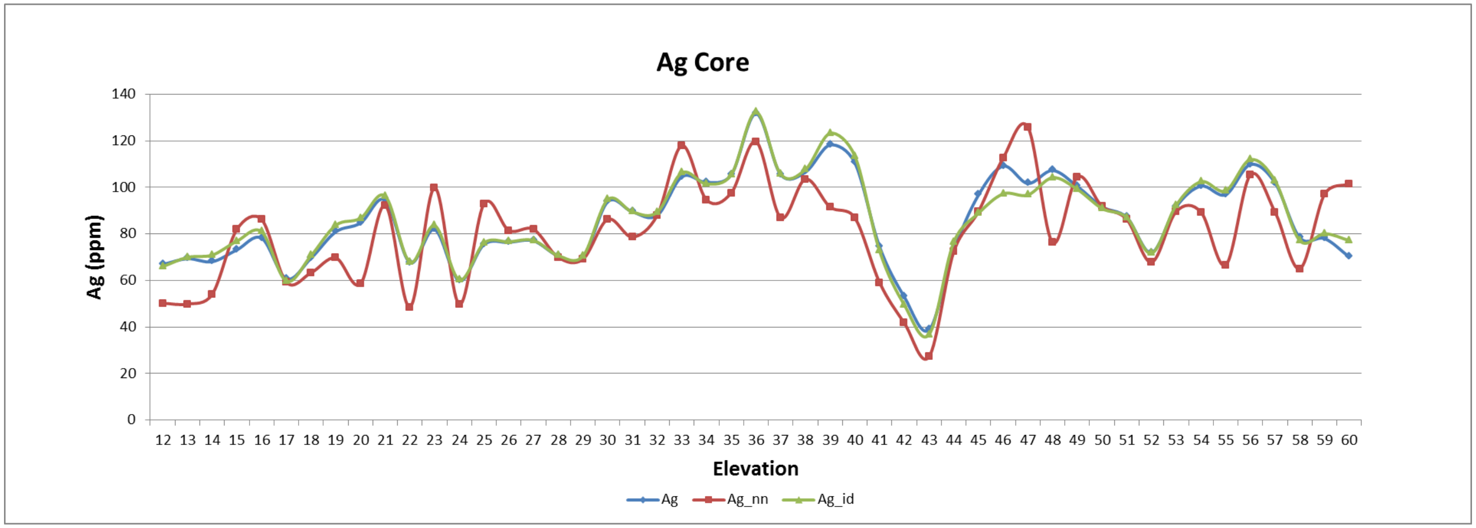


Figure 14-11 Swath Plot Ag Granite

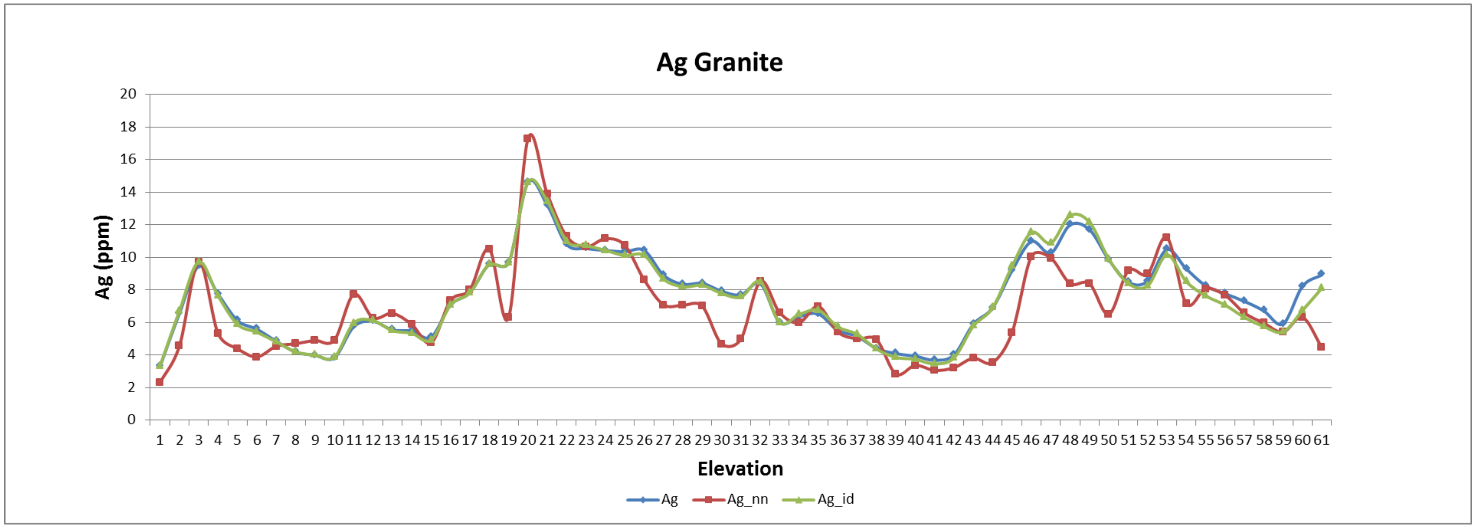


Figure 14-12 Swath Plot Pb Core

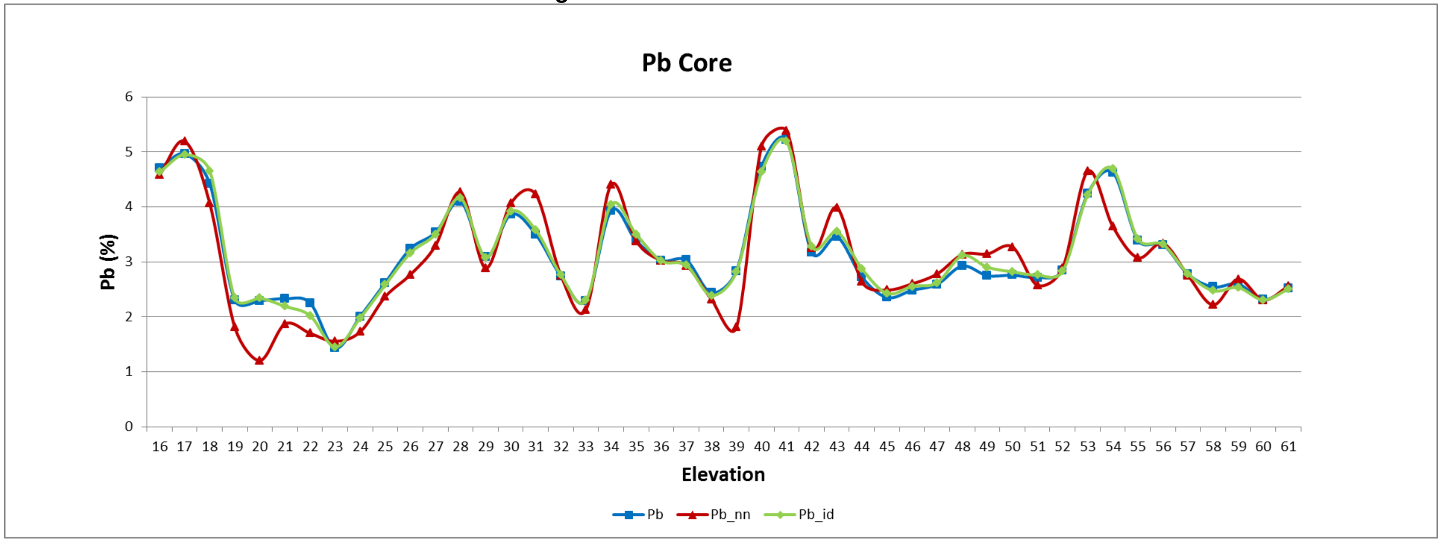


Figure 14-13 Swath Plot Pb Granite

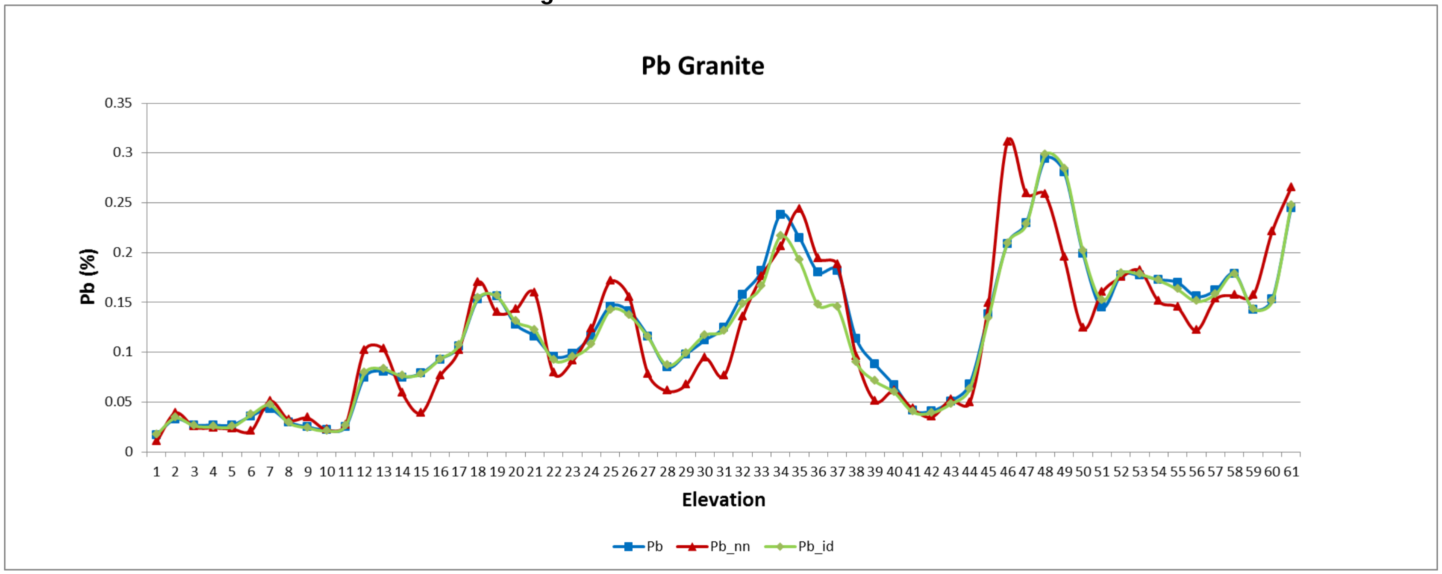


Figure 14-14 Swath Plot Zn Core

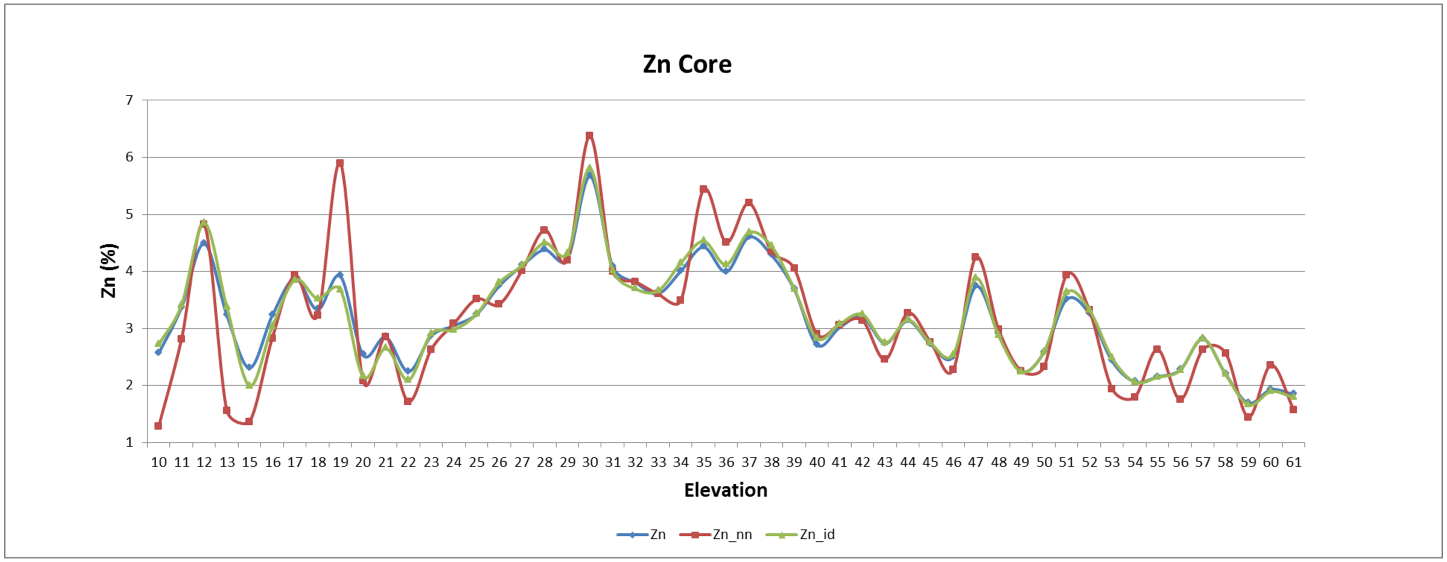
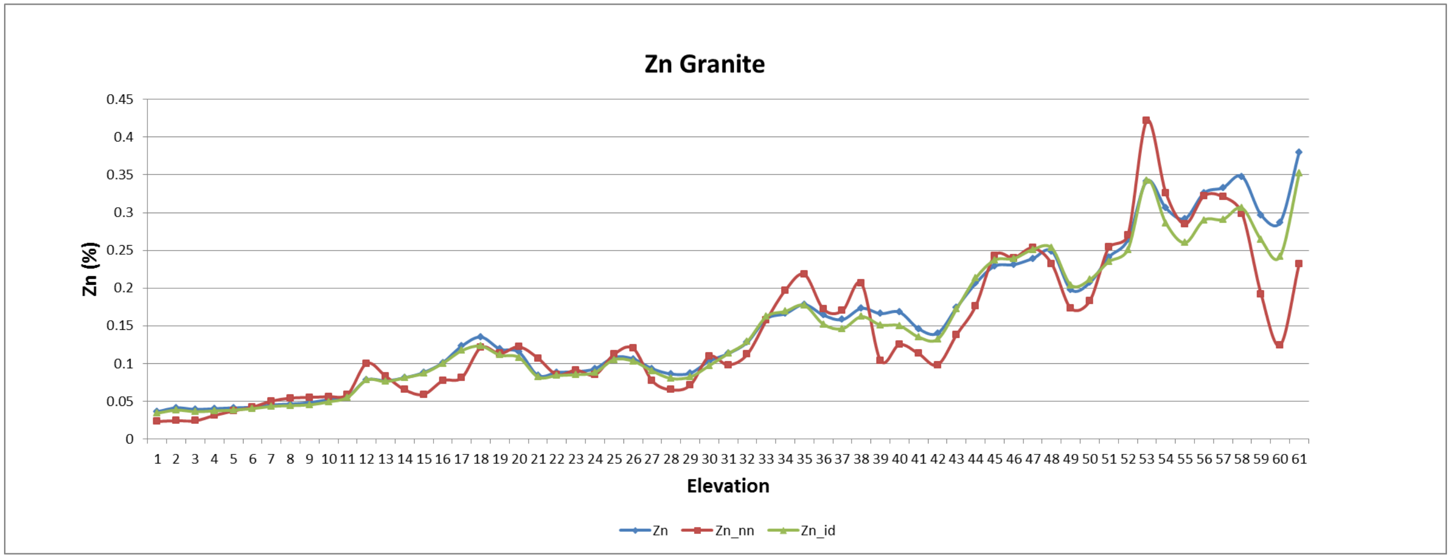


Figure 14-15 Swath Plot Zn Granite



14.3 Classification

Categorization follows the CIM standards whereby mineral resources are defined as the mineral inventory that has a reasonable prospect for economic extraction, and for which geological and grade continuity might reasonably be assumed. RPM considers that on this basis only blocks estimated with at least two qualify as resources. Additionally, RPM considers that the data used in resource estimation conforms to NI 43-101 and industry best practice guidelines in respect of data collection procedures, security, assaying methodology and QAQC programs.

RPM defined grid spacing based on geological and grade continuity to show a reasonable level of confidence in defining measured, indicated and inferred resources. RPM would recommend incorporating, in a feasibility study, the estimation errors associated with annual and quarter production panels to define indicated and measured resources, respectively.

RPM analyzed sections and levels along with indicator and grade correlograms to define a reasonable geological and grade continuity. RPM concluded the Bilbao deposit is a high variability deposit and therefore, elected to define indicated resources as those blocks having a drill hole spacing (of at least three drill holes) of 35 m by 35 m and inferred resources as those blocks having a drill hole spacing (of at least three drill holes) of 50 m by 50 m. No measured resources were classified in this model.

The drill hole grid spacing of 35 m to define indicated resources is tighter than the previous model of 50 m by 50 m. The previous 2010 model used an estimation pass of 40m by 40m by 20 m to assign the indicated category to the blocks. The 40m by 40m by 20m pass distance is equivalent to a grid spacing of 80m x 80m.

RPM stored into the blocks the average distance of four and three nearest drillholes and afterward, tagged the category associated with the drill spacing defined above. RPM then applied a smoothing procedure to reduce "spotted dog effect" by IK of the raw categories, the maximum probability was used to assign the final block category, but some indicated spots still remained within inferred zones and vice versa. RPM detected that non-sampling intervals were producing the spots in spite of the fact the drill hole grid is regular in most of the deposit. Figure 14-16 shows the indicated blocks and the drill holes where white intervals are the non-sampling intervals. Finally, RPM manually fixed the spots by creating polygons to constrain the indicated zone, removing internals of inferred blocks and external indicated blocks.

14.4 Results

For the purpose of determining resources at various cutoff grades, Zn equivalent values were defined, based on the average price of the last 3 years. The utilized prices were US$0.935 lb/Zn, US$1.008 lb/Pb, and US$30.235 oz/Ag. Metallurgical recoveries were applied in the equivalent equation as 76.7%, 90.6% and 73.4% for Zn, Pb, and Ag, respectively. The Zn equivalent equation is as follows:

$$Zneq = Zn + 0.969*Pb + 0.09947*Ag$$

The historical records suggest that approximately 1 Mt has been extracted from the open pit workings and from underground stopes between the upper levels 40-76 according to Gallon, 2010. RPM strongly recommends updating the underground survey in order to provide a better estimate of historical mining. This material was discounted from the indicated oxide resources.

The total sulphide resources are listed by Zneq cutoff in Table 14-11. The total indicated sulphide resources are listed by Zneq cutoff in Table 14-12.

Figure 14-16 EW Section Showing Indicated Blocks and Sampling/Non-Sampling Intervals

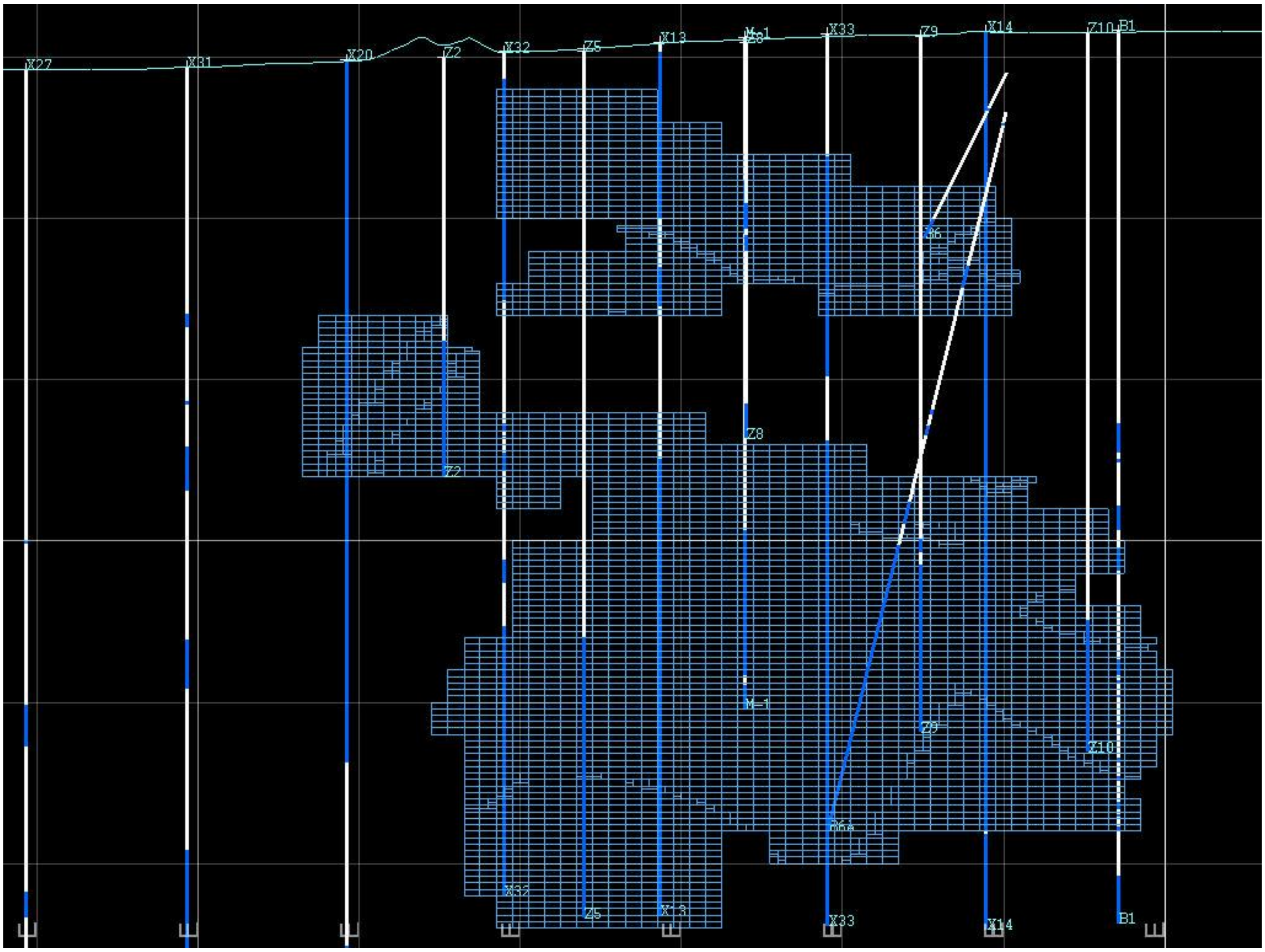


Table 14-11 Total Sulphide Resources

Cutoff	Zn equiv. (%)	Indicated Tonnes	Inferred Tonnes	Total Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
0.0	0.458	17,831,825	209,299,540	227,131,365	0.105	0.078	6	0.015
0.5	1.994	12,978,084	28,432,051	41,410,135	0.449	0.378	24	0.06
1.0	2.906	9,730,543	14,496,555	24,227,098	0.656	0.566	35	0.083
1.5	3.781	7,491,992	8,413,865	15,905,857	0.893	0.742	44	0.101
2.0	4.934	6,014,809	4,177,905	10,192,714	1.273	0.993	54	0.128
2.5	5.979	5,124,220	2,234,647	7,358,867	1.658	1.214	63	0.15
3.0	6.883	4,555,809	1,201,032	5,756,841	2.025	1.403	69	0.167
3.5	7.569	4,138,652	708,864	4,847,516	2.319	1.545	74	0.177
4.0	8.081	3,801,363	474,136	4,275,499	2.546	1.651	77	0.184
4.5	8.551	3,481,995	328,528	3,810,523	2.757	1.751	80	0.189
5.0	8.966	3,183,043	252,741	3,435,784	2.945	1.83	83	0.194
5.5	9.412	2,878,188	189,440	3,067,628	3.13	1.925	86	0.2
6.0	9.844	2,601,525	142,867	2,744,392	3.303	2.019	89	0.206

Table 14-12 Total Indicated Sulphide Resources

Cutoff	Zn equiv. (%)	Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
0.0	2.633	17,831,825	0.756	0.554	27	0.077
0.5	3.521	12,978,084	1.015	0.748	35	0.102
1.0	4.451	9,730,543	1.294	0.952	44	0.126
1.5	5.412	7,491,992	1.599	1.154	53	0.146
2.0	6.316	6,014,809	1.909	1.332	61	0.163
2.5	7.026	5,124,220	2.171	1.467	68	0.175
3.0	7.562	4,555,809	2.382	1.571	72	0.183
3.5	7.997	4,138,652	2.561	1.657	75	0.188
4.0	8.374	3,801,363	2.72	1.733	77	0.192
4.5	8.753	3,481,995	2.88	1.814	80	0.196
5.0	9.129	3,183,043	3.041	1.886	83	0.201
5.5	9.54	2,878,188	3.203	1.969	86	0.206
6.0	9.943	2,601,525	3.361	2.055	89	0.212

Total resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore can be seen in Table 14-13.

Table 14-13 Total Resources

Ore Type	Zn equiv. (%)	Indicated Tonnes	Inferred Tonnes	Total Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.50	791,082	3,069,582	3,860,664	1.70	2.33	42	0.17
Mixed	7.10	778,336	238,923	1,017,259	2.06	2.17	52	0.18
Sulphide	6.88	4,555,809	1,201,032	5,756,841	2.03	1.40	69	0.17
Total	6.76	6,125,227	4,509,537	10,634,764	1.91	1.81	58	0.17

Indicated resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore can be seen in Table 14-14.

Table 14-14 Indicated Resources

Ore Type	Zn equiv. (%)	Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.69	791,082	1.73	2.53	39	0.18
Mixed	7.93	778,336	2.52	2.48	51	0.21
Sulphide	7.56	4,555,809	2.38	1.57	72	0.18
Total	7.5	6,125,227	2.31	1.81	65	0.19

Inferred resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore can be seen in Table 14-15.

Table 14-15 Inferred Resources

Ore Type	Zn equiv. (%)	Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.38	3,069,582	1.69	2.23	42	0.16
Mixed	4.43	238,923	0.59	1.13	55	0.11
Sulphide	4.31	1,201,032	0.67	0.77	60	0.11
Total	5.73	4,509,537	1.36	1.78	47	0.15

Whilst the adoption of a 3.0% Zneq cut-off is deemed in line with the norms of underground sulphide mining operations elsewhere in the world, one must keep in mind that zinc equivalents are price sensitive and in this case are dependent upon the price of the contained mix of metals namely zinc, lead, and silver.

A substantial price change in any of these contained metals will affect that portion of the mineralized body which can be mined economically. Recognizing this, and in order to counteract it, the price of each of the contained metals used in the Zneq calculation in this report is the average of that prevailing over the last three years. The adoption of a three year metal price average offers a degree of assurance that the resource estimate will have validity in the medium term but one must bear in mind that any substantial changes in metal price will affect the tonnage of mineralized material available for exploitation. As a consequence of this, no responsibility is accepted for future changes in metal prices which could affect the resource estimate given in this report.

15. Mineral Reserve Estimates

Due to the preliminary nature of the project, no reserves were estimated.

16. Mining Methods

16.1 Summary

Given the results of metallurgical testwork on the oxide and transition mineral zones, the mine plan incorporated in this study targets the extraction of the sulphide zone only. Underground mining methods will be used to access the sulphide zone located approximately 50 meters below surface, and accessed via a portal and ramp system.

The main proposed mining method is Longhole Open Stoping using downholes, while near the top of the deposit Longhole Open Stoping using upholes will be employed. Stopes will have maximum nominal dimensions of 24 metres wide by 12 metres long and a 24-metre vertical height. Longhole stopes will be backfilled with a cemented rock fill. Also, near the edges of the deposit, remnant Longhole Open Stoping with downholes will be used with no backfill placed in mined out stopes.

The main access to the underground mine will be via a main ramp from surface to the 1860 Level. The main ramp will connect from surface to all production levels in the mine, and provide a passage way for transportation of ore and waste material, personnel, materials and equipment. From the main ramp level accesses in waste at approximately 24 metre vertical intervals will be developed from the 2000 to 1860 Levels.

Underground mining will utilize mobile rubber tired diesel powered equipment to haul 2,000 tonnes per day of potentially economic mineralization or the equivalent of 720,000 tonnes per year.

16.2 Mine Design and Production Planning

16.2.1 Main Access Ramp

The main access to the underground mine will be via a main ramp from surface to the 1860 Level. The main ramp will connect from surface to all production levels in the mine. Trucks will haul ore to surface from the levels developed at 24 metre vertical intervals. Waste (as required), backfill, personnel, equipment and materials will be transported via the main ramp as well. Figure 16-1 shows a three dimensional view of the proposed underground mine design.

The ramp will generally be developed as a series of straight sections with connecting curves. The ramp will be 4.5 metres wide by 4.5 metres high (to accommodate travel of 40 tonne trucks and facilitate ventilation requirements), at a grade of 15 percent. Passing bays will be provided at a maximum nominal distance of 400 to 500 metres along the length of the ramp. Safety bays will be provided every 100 metres in the ramp. The passing bays will be used as remucks.

Services installed in the ramp will include a 203 mm airline, 152 mm waterline, 152 mm drainline, 15kVand 1,000 volt power cable, fibre optic cable and central blasting line.

Ramp development as well as all pre-production development will be undertaken by a contractor using 2-boom electric/hydraulic development jumbos, 4.6 cubic metre load-haul-dump (LHD) units, scissor lifts and 40 tonne haul trucks. Waste rock from the development headings will be transported by the trucks to surface or after mining begins, placed in mined out stopes as backfill.

16.2.2 Level Access Development

From the main ramp level accesses in waste at approximately 24 metre vertical intervals will be developed from the 2000 to 1860 Levels.

Levels accessing mining panels will be developed from the ramp as shown on a representative plan view of the 1940 Level development in Figure 16-2. Levels will be established at 24 metre vertical levels with the top level at the 2000 metre elevation (with surface at 2050 m approximately) and the lowest level at the 1860 metre elevation.

Figure 16-1 Mine Development – Looking North West

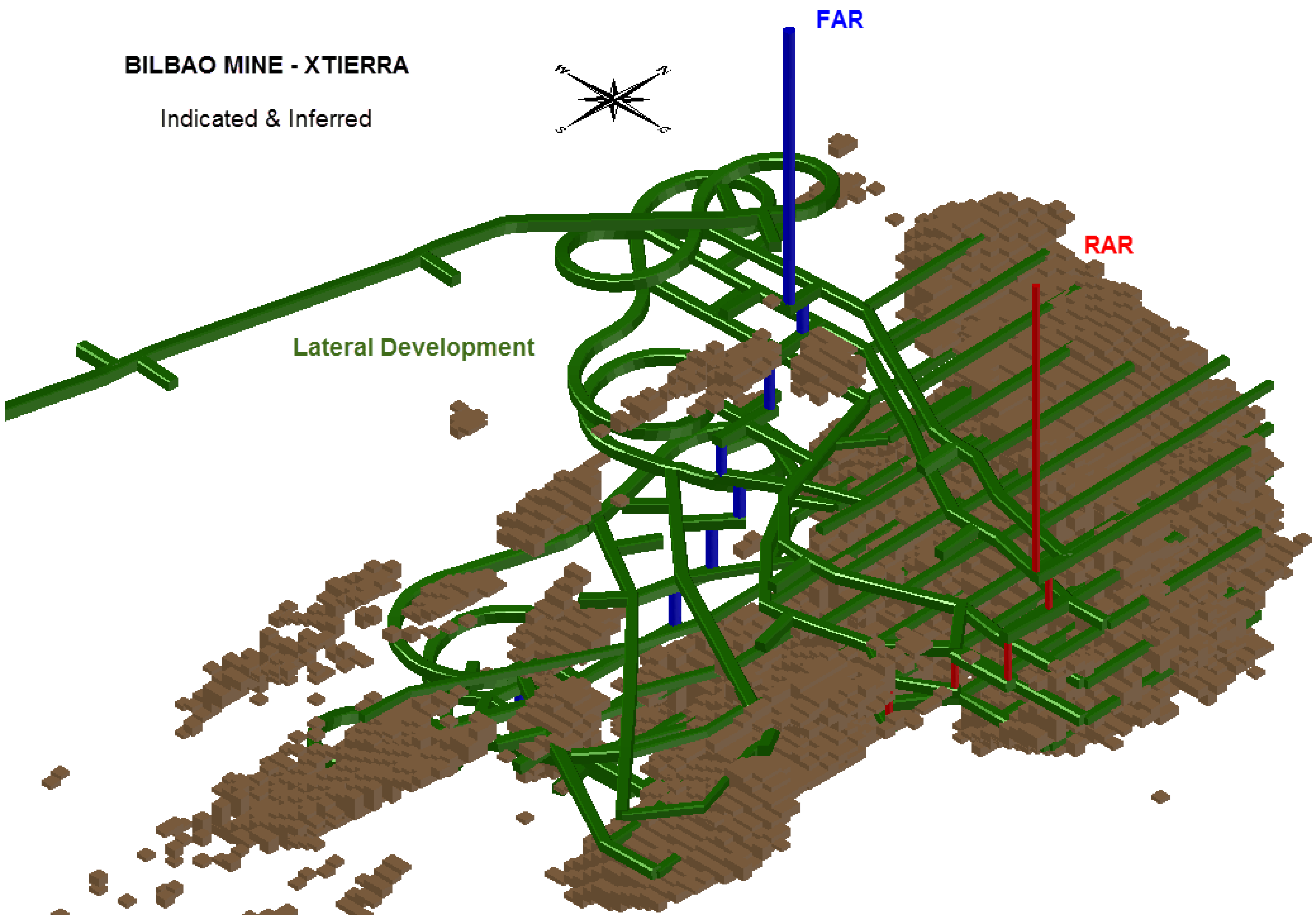
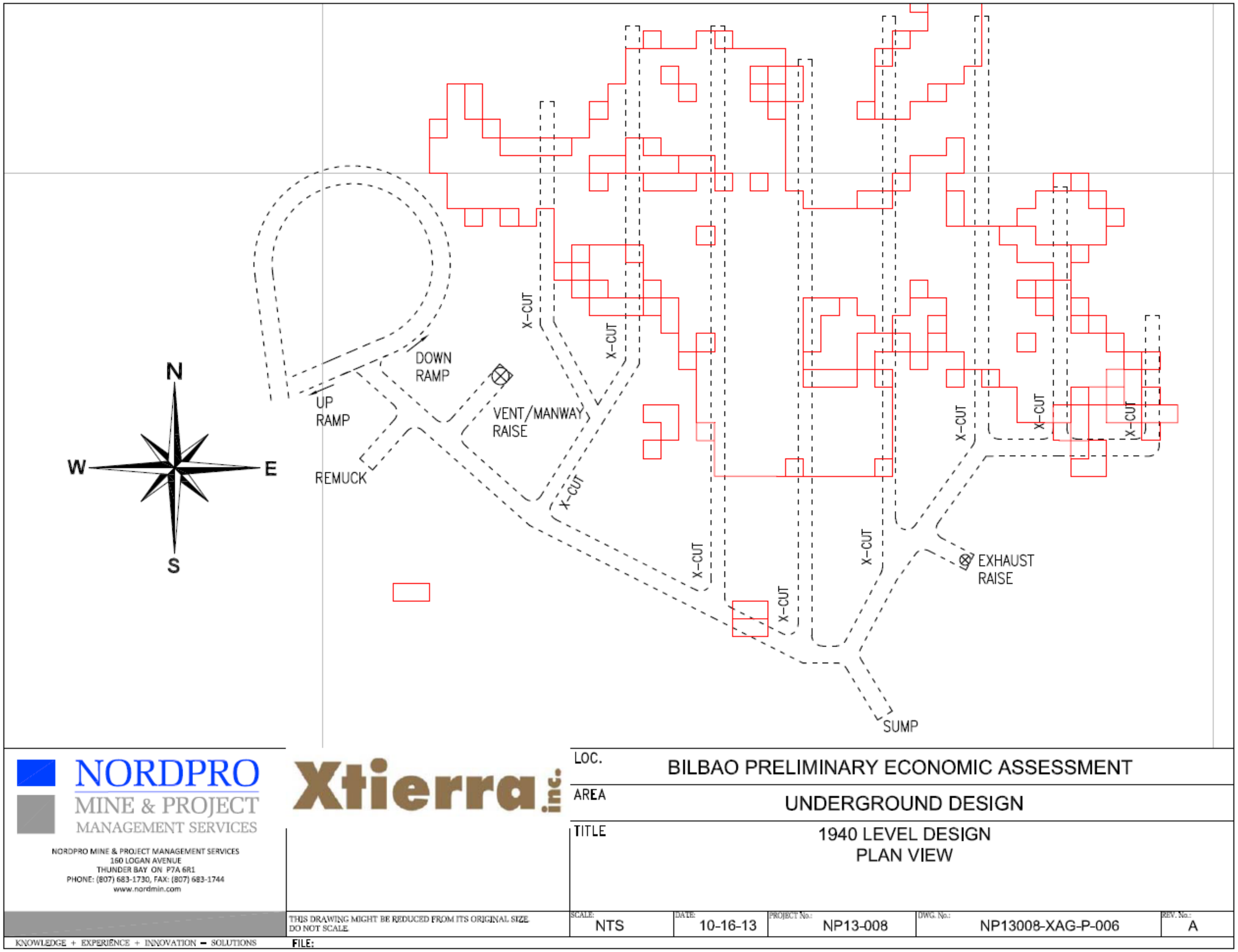


Figure 16-2 Typical Level Configuration



At each level access point, the ramp will be developed horizontally for 20 metres to facilitate equipment movement to and from each sublevel. Remuck stations will be developed approximately every 150 metres along level drifts (corresponding to future locations for stope accesses or other permanent development). Level development will consist of a main level drift in waste, with drawpoint accesses developed, from the main level drift, through the ore. Truck loading stations will be developed on each level where mucking from stopes is being performed, with the stations located centrally.

All lateral development will be 4.5 metres wide by 4.5 metres high using diesel power rubber tired equipment similar to ramp development equipment.

Services installed in the headings will include 152 mm airline, 102 mm waterline, 102 mm pumpline, 660 volt cable and fibre optic cable.

16.2.3 Stoping Methods and Production Planning

The main proposed mining method is Longhole Open Stoping using downholes drilled in a fan pattern. Areas near the top of the ore body with heights of less than 20 metres will be mined using the Longhole Open Stoping with upholes mining method. Stopes will have maximum nominal dimensions of 24 metres wide by 12 metres long and a 24-metre vertical height. Longhole stopes will be backfilled with a cemented rock fill. Also, near the edges of the deposit, remnant Longhole Open Stoping with downholes will be used with no backfill placed in mined out stopes.

16.2.3.1 Stope Development

Stope development will consist of a panel (series of stopes across the width of deposit) access crosscut developed from the main level drift through the ore. This access crosscut will be used as a drawpoint for mucking each stope in a primary or secondary stoping panel.

The deposit geometry with a length of up to 150 to 200 metres and width in excess of 100 metres in areas requires that stopes be combined into 24 metre wide mining panels comprising a number of stopes accessed along a single panel access crosscut. Panels will be mined in a primary/secondary sequence and filled with cemented rockfill after mining.

16.2.3.2 Longhole Open Stoping

A pilot overcut crosscut from the footwall drift will be developed at the top of a stope from the footwall to hangingwall of the potentially economic mineralized zone. A full width and length undercut sill will be developed in initial stopes or have been left from backfilling stopes below the block to be mined. The stope access pilot crosscuts will be developed 4.5 metres wide by 4 metres high, with the undercut height being 4.5 metres. Development will utilize 2 boom electric hydraulic jumbos and 3.7 to 4.6 cu. m. bucket LHD's mucking to 40 tonne underground haul trucks. All stope sills will be resin rebarred and screened using a scissor/ bolter unit where the operator stands on the scissor lift platform to undertake the work.

A three metre diameter slot raise will be developed from the undercut to overcut.

Stopes will be drilled off with 102 mm diameter drill holes, with a fan pattern of down holes drilled from the overcut crosscut. The fan drill holes will have a toe spacing of approximately three metres and ring spacing will be approximately 2.4 metres.

Stope downholes will be loaded with ANFO. The downhole rings will be blasted in two or three blasts.

A stope will be mucked out in approximately 1 to 1.5 months using 4.6 cu. m. bucket LHD's loading the 40 tonne underground haul trucks. Average LHD haul distances will be approximately 150 to 200 metres. Three LHD's will be required to meet daily production from stopes.

Stope panels will be mined from the centre of the deposit to the outsides of the potentially economic mineralization, where most stopes will be backfilled with cemented rock fill to facilitate mining of the adjacent stope(s).

16.2.3.3 Rock Handling Facilities

The 40 tonne haul trucks will be loaded by LHD's at truck loading stations on each level. The truck loading stations will consist of a 20 metre length area with the back height increased to 10 metres to facilitate LHD bucket height for dumping.

16.2.3.4 Backfill

Backfill in stopes will consist of cemented rock fill in primary stopes and uncemented rock fill in secondary and isolated stopes. Remnant stopes will not be backfilled.

Waste rock will be quarried approximately 2 kilometres from the mine, crushed to minus 0.3 metres and trucked to the mine.

Waste rock will be sent underground in a backfill raise equipped with a covered truck dump on surface. The bottom of the backfill raise on each level will be equipped with a finger raise feeding a truck loading chute to load crushed waste rock backfill into 40 tonne underground haul trucks. The haul trucks will haul the backfill rock on each level to the top of the stope being backfilled. For backfilling of primary stopes the haul trucks will dump the rock onto a baffle slide where cement slurry will be sprayed onto the rock as it enters the mined out stope. Cement content in the backfill will be approximately three percent. The cement slurry will be supplied from surface via boreholes and pipelines on the levels, with a holding tank on each level where backfilling is taking place to buffer cement slurry capacity between the surface plant and backfilling operations underground.

The cement slurry plant on surface will comprise of a cement silo equipped with a screw feeder measuring cement into a mixing tank where water is added to create cement slurry. The slurry will be pumped a short distance to the collar of the cement slurry delivery borehole from surface.

Secondary stopes will have uncemented rock backfill trucked to the stopes from the backfill raise and dumped directly into the mined out stopes.

Backfilling will be placed at a nominal rate of 1,200 to 1,300 tonnes per day on one shift using a 40 tonne underground haul truck.

16.2.3.5 Dilution and Losses

Based on the selected mining method a dilution factor of 10% is applied which allows for dilution from hanging and footwall wall exposures and cemented backfill dilution which results from blasting against backfilled stopes.

Mining recovery of 95% is assumed for this deposit.

16.2.3.6 Personnel and Materials Handling

Manpower and materials will enter and leave the mine via the main access ramp from surface. Personnel will travel in vehicles and/or personnel carriers to workplaces or equipment parking areas. Materials will be moved on a services truck, equipped with a boom crane, operating in the ramp. Materials will be transported to and placed in designated storage areas close to mining.

16.2.4 Geotechnical and Ground Control

Golder Associates Ltd.., Mississauga, Canada performed the rock mechanics studies. Design parameters were provided and incorporated into the mine design and cost estimates. An update memo report was prepared in

2013 by Golder for modified stoping configurations and is included in the report entitled "Empirical and Numerical Analyses for Stope Sizing."

Stope dimensions are based on rock quality determinations and expected achievable open spans in stopes. The stope dimensions and extraction ratios anticipated will not be achieved without the use of a good quality, cemented rock fill and to a lesser extent un-cemented backfill.

All permanent openings will have arched backs. Ground support will generally consist of pattern bolting using grouted rebar and welded wire mesh.

16.3 Mine Services

16.3.1 Ventilation

Ventilation will be provided to the mine by a Fresh Air Raise (FAR) and the main ramp from surface. A network of lateral development on each level will connect the mining areas to a Return Air Raise (RAR).

The mining operation to support the mining equipment fleet will require ventilation air volumes of approximately 210 to 230 cu. metres per second (450,000 to 500,000 cfm). The ventilation system will consist of a push-pull system utilizing the ventilation raises and the main access ramp.

Two 3-metre diameter ventilation raises will be developed from surface to the bottom of the mine in legs and be located at either end of the levels. One raise will be an intake raise and the other an exhaust raise. High pressure fans will be located on surface on top of the exhaust raise and low pressure fans on top of the intake raise.

Air will flow from the intake ventilation raise along a level, be picked up by auxiliary ventilation fans and pushed into stope overcut accesses and drawpoints and flow back out to the level drift. Air will travel in the level drift to the exhaust raise and up to surface through the raise.

Approximately one half of the fresh air sent underground will be split off and enter the ramp from the levels and flow up the ramp to surface. If required, low pressure fans will be connected to the ramp near the portal to assist air exhaust to surface.

16.3.2 Electrical Distribution

Primary electrical power for the mine will be provided from the main surface substation connected to the outside powerline.

The powerline will be connected to a surface substation located near to the mine portal. Power from the main substation will feed the main underground power line, a 500 mcm cable, installed in the main access ramp from surface. This power line will feed portable substations located on levels central to working areas. Portable power centres will supply loads on the nearby levels and transform power down to 4160V and 600V as required.

On surface, the substation will also provide 4160V feeds to drive ventilation fans and other power requirements for the underground mine surface facilities. The system will utilize a switch room/MCC panel near the ramp portal.

16.3.3 Compressed Air

Compressed air will be supplied by 2 compressors in enclosures located in a small covered structure, near the ramp portal. They will provide approximately 150,000 litres per minute at a minimum pressure 8.3 bar (120 psi) to the underground mine. Each compressor will operate at half capacity to ensure one compressor could provide mine requirements when the second compressor is being repaired or maintained.

The compressors will supply the main compressed air pipeline located in the main access ramp from surface.

16.3.4 Service Water

The underground mine will require approximately 80 million litres of service water per year for use in drilling, dust suppression, etc.

Water will be sent underground in a pipeline located in the trackless access ramp from surface. This will feed the main distribution lines on the levels, which will send water to the stope access crosscuts. Water pressures and volumes will be controlled by installing water stations, at appropriate vertical intervals within the mine, which will house a transfer station and holding tanks.

16.3.5 Mine Communications and Control Systems

The mine will also have a communications network to provide voice communications and some PLC monitoring within the mine.

16.4 Mine Support Facilities

The majority of underground infrastructure will be associated with facilities located on the 1860 Level and include a breakdown maintenance shop, main dewatering sumps, fuel and lube stations, explosives magazine, refuge station and storage areas.

16.4.1 Mine Dewatering

Water collection sumps will be located on each level. The sumps will be located near the point where the ramp and level access crosscuts intersect and will be designed to prevent water entering the ramp from the levels. Overflow drill holes from the sumps will send water to the main water collection sumps, for settling, recirculation and/or discharge from the mine.

Main collection sumps will be located on the 1860 Level. Each main sump will be comprised of two dirty water sumps and one clear water sump. Dirty water sumps will be sub-divided by removable timber baffle walls into three compartments to aid in settling of solids. The dirty water sumps will be used one set at a time, and slimes removed from the non-operational sump with LHD’s. Water will overflow from the dirty water sumps into a clear water sump.

Each clear water sump, similar in size to the dirty water sumps, will be utilized to treat and store clear water prior to recirculation within the mine or discharge. Water will be pumped to a surface holding pond for underground process water or discharged to the water treatment facility on surface.

16.4.2 Breakdown Maintenance Shop

A mobile equipment breakdown maintenance shop will be used to perform all breakdown maintenance on mobile mining equipment. The shop will be constructed near the 1860 Level, off the ramp. The shop will consist of a main shop area for one large piece of equipment or a couple of smaller units. The facility configuration will consist of an access drift leading to the main shop area, a welding area, wash bay area, parts storage warehouse, electrical room, lunchroom and supervisor’s office.

The main shop area will be equipped with an overhead bridge crane. The electrical room, meeting room and office will be isolated by steel hinged doors. The lunchroom will be equipped with wooden benches and tables and the office will be equipped with computer workstations connected to the mine information management system.

16.4.3 Fuel Stations

Portable self-contained fuelling stations will be located on levels where mining equipment will be parked. The units have built in isolation doors and fire suppression. A lube bay will be included in the maintenance shop complex and be equipped with HDPE lube tanks on a steel beams and grating platform with a surrounding concrete wall, acting as a catchment basin for any leaks from the tanks.

16.4.4 Refuge Station

Main refuge stations will be located on the 1940 Level.

Refuge stations will be fitted with a double door entry system in concrete walls at one end. The facility will include wooden benches and tables, hand washing station and other equipment and supplies, as well as a supervisor's desk and other associated furniture. The refuge stations will also be equipped with safety and rescue equipment. Compressed air and water lines will be connected from the mines supply system to lines inside the refuge station. The facility will be fitted with an electric heater unit and be vented through intake and exhaust ventilation ducts to the outside.

16.4.5 Explosives Storage

All blasting will utilize ANFO explosives. ANFO will be delivered in bulk bags, to explosives magazines.

Explosives magazines will be located on the 1860 and 1940 Levels. The magazine entrance will include a concrete wall with doors to allow access for mobile equipment and people traffic.

Both sides of the magazine will be fitted with wooden shelving on which bulk explosives bags can be placed. This magazine will require a fire suppression system.

Other stick explosives will be stored in this magazine as well.

16.4.6 Detonator Magazine

Detonator magazines will be located near the explosives magazines. The magazines will be equipped with suitable wooden shelving to allow stacking of detonator boxes on each side. The entrance will be blocked with timber posts and screen, with a man door in the wall.

16.4.7 Materials Storage Areas

Storage areas, specially constructed for the purpose, for storing mining consumables including pipe and fittings, ground support materials, ventilation supplies, etc. will be developed on the 1860 & 1940 Levels. The storage areas will include shelving and low wooden racking to safely store articles. Materials and parts will be palletized or placed in specially designed containers (for bulk materials and parts) for sending underground via the ramp. Service vehicles will transport the bulk materials to the storage areas. Materials will be distributed from the storage areas to work place storage areas by service vehicles.

16.4.8 Washrooms

Portable toilet units equipped with a mine toilet and small sink will be located on appropriate working levels and near to refuge stations.

16.4.9 Surface Support Facilities

Surface support facilities will include a main maintenance shop; backfill plant; explosives magazines; laydown yard; mine rescue station; water collection ponds; mine supervision, geology, engineering and administration offices; and power substation.

A small maintenance shop facility will be provided to perform major equipment repairs and rebuilds. A description of the shop facility is contained in Section 18 of this report. The warehouse for mine items only will be a combination of pallet (large or bulk items) and shelved (smaller items) storage.

The explosives storage area for the mine will be located 500 metres from mining and other facilities. The magazines will be housed in metal shipping containers and located so they can be observed by security located at the services site. The magazines will not be in direct line of sight of the mine or other facilities to protect mine personnel, equipment and facilities.

A laydown yard will be constructed near to the ramp portal to store materials and equipment required for the underground mine. This laydown yard will have raised timber stands on which to place large materials such as screen, pipe etc. as well as gravel graded areas for storing equipment and materials. A storage building will store equipment requiring protection from the elements.

A fully equipped mine rescue station is required on property. The mine rescue station will be equipped with all necessary equipment, including self-contained breathing apparatus, flame lamps, gas testing equipment, rescue equipment, etc. and supplies and chemicals required to operate the station. There will be enough equipment to, in an emergency, have three 5-person mine rescue teams operating or on standby at any one time.

All underground mine water will be sent to a water collection pond and reused or discharged.

16.5 Net Smelter Return Cutoff Value

The Net Smelter Return (NSR) cutoff value of $45.21 per tonne of ore used for the Bilbao Project stope tonnes and grade determination was derived as follows:

Table 16-1 NSR Cutoff Value

Component	**Cost ($/t)**
Mining	27.00
Processing	13.21
G&A	5.00
Total	45.21

16.6 Potentially Mineable Resource

The potentially mineable underground resource is estimated to be 5.2M tonnes at grades of 2.10 % Zn, 1.40 % Pb and 63.96 grams Ag per tonne. The tonnes and grade include an average dilution of 10 percent, at zero grade, as well as mining losses of 5%. This Preliminary Economic Assessment relies on Indicated Mineral Resources of the sulphide zone (approximately 75 percent of the total sulphide resource tonnes) as well as Inferred Mineral Resources of the sulphide zone.

It should be noted that the Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that will enable them to be categorized as Mineral Reserves. Also the cost projections range in accuracy from PEA to Feasibility level. Therefore, there is no guarantee that the economic projections contained in this Preliminary Economic Assessment will be realized.

16.7 Development and Production Schedule

The life of mine development schedule is shown in Table 16-2. All development work in waste will be performed by mining contractors. Xtierra personnel will undertake sill development in potentially economic mineralization. To meet the development schedule will require a contractor to have 2 development crews performing ramp and lateral development work for much of the project pre-production period. Following that one crew will be sufficient to complete development work. Contractor Alimak raise crews will also develop ventilation raises. The contractor could be expected to advance at the following rates:

Single heading – Ramp & Lateral Development	1.5 rounds or 5 m/day
Multiple Heading – Ramp & Lateral Development	2.0 rounds or 6.7 m/day
Raising	2.0 rounds or 4.8 m/day

Table 16-2 Mine Development Schedule

Heading	Year									Total
	-1	1	2	3	4	5	6	7	8	Metres
Ramp - Surface to 2020	1,007									1,007
Ramp - 2020 to 2000	130									130
Ramp - 2000 to 1980	164									164
Ramp - 1980 to 1960	143									143
Ramp - 1960 to 1940	132									132
Ramp - 1940 to 1920		178								178
Ramp - 1920 to 1900					162					162
Ramp - 1900 to 1880					126					126
Ramp - 1880 to 1860					147					147
2020 Level Development	78			485						563
2000 Level Development	99			675						774
1980 Level Development	87	621	429							1,137
1960 Level Development	81	629								710
1940 Level Development	112	550								662
1920 Level Development							696			696
1900 Level Development						1,047				1,047
1880 Level Development						537				537
1860 Level Development					420					420
Vent Raise #1 - 2020 to Surface	124									124
Vent Raise #1 - 2000 to 2020	12									-
Vent Raise #1 - 1980 to 2000	21									-
Vent Raise #1 - 1960 to 1980	13									-
Vent Raise #1 - 1940 to 1960		14								14
Vent Raise #1 - 1920 to 1940					18					18
Vent Raise #1 - 1900 to 1920					15					15
Vent Raise #1 - 1880 to 1900					12					12
Vent Raise #1 - 1860 to 1880					16					16
Exhaust Raise #1 - 2020 to Surface		117								117
Exhaust Raise #1 - 2000 to 2020		15								15
Exhaust Raise #1 - 1980 to 2000		18								18
Exhaust Raise #1 - 1960 to 1980		13								13
Exhaust Raise #1 - 1940 to 1960			16							16
Backfill Raise- 2020 to Surface	129									129
Backfill Raise - 2000 to 2020		17								17
Backfill Raise - 1980 to 2000		26								26
Backfill Raise - 1960 to 1980		18								18
Backfill Raise - 1940 to 1960		19								19
Backfill Raise - 1920 to 1940					23					23
Backfill Raise - 1900 to 1920					20					20
Backfill Raise - 1880 to 1900					17					17
Backfill Raise - 1860 to 1880					21					21
Total Lateral Development	**2,033**	**1,978**	**429**	**1,160**	**855**	**1,584**	**696**	**-**	**-**	**8,735**
Total Raising	**299**	**257**	**16**	**-**	**142**	**-**	**-**	**-**	**-**	**668**

The pre-production development period, will require approximately 1 year, after permitting and detailed engineering is completed.

The mine production schedule is shown in Table 16-3. The schedule is based on a production rate of 2,000 tpd of potentially economic mineralization, or 720, 000 tonnes per year. This provides for a mine life of approximately 8 years, mining out the indicated and inferred sulphide resources available.

Table 16-3 Life of Mine Production Schedule

Stoping Area	Year									Total
	-1	1	2	3	4	5	6	7	8	Tonnes
1825 Inferred							16,168			**16,168**
1860 Total							74,714			**74,714**
1860 Remnant Total							13,398			**13,398**
1860 Inferred							29,554			**29,554**
1880 Total							165,157			**165,157**
1880 Remnant Total							28,231			**28,231**
1880 Inferred							169,440			**169,440**
1900 Total							157,685	321,541		**479,226**
1900 Remnant Total								45,577		**45,577**
1900 Inferred								211,963		**211,963**
1920 Total								140,918	171,122	**312,040**
1920 Remnant Total									39,480	**39,480**
1920 Inferred									356,080	**356,080**
1940 Total		298,992	466,897							**765,889**
1940Remnant Total			2,123							**2,123**
1940 Inferred			118,204							**118,204**
1960 Total			132,777	570,868						**703,645**
1960Remnant Total				55,415						**55,415**
1960 Inferred					128,303					**128,303**
1980 Total				93,717	452,191					**545,908**
1980Remnant Total						20,650				**20,650**
1980 Inferred					137,612					**137,612**
2000 Total					1,894	510,249				**512,143**
2000 Remnant Total						61,999				**61,999**
2000 Remnant Upholes Total						32,995				**32,995**
2000 Inferred						92,831				**92,831**
2000 Inferred Upholes						1,276	65,653			**66,929**
Total Tonnes	**0**	**298,992**	**720,000**	**720,000**	**720,000**	**720,000**	**720,000**	**720,000**	**566,682**	**5,185,674**
Grades										
Zn		2.37	2.27	2.91	2.34	2.99	1.94	0.96	0.97	
Pb		1.70	1.63	1.88	1.55	1.66	1.07	0.85	0.95	
Cu		0.16	0.16	0.18	0.19	0.19	0.16	0.16	0.12	
Ag		60.08	62.21	68.28	63.90	61.34	68.86	72.65	48.84	

16.8 Equipment Fleet Requirements

The mining equipment required to develop the mine and produce 2,000 tonnes per day of potentially economic mineralization is presented in Table 16-4.

Table 16-4 Mine Equipment Fleet

Trackless Equipment	# Units
4.6 cu. m. LHD	3
2 boom E/H Dev. Jumbo	1
102mm L/H Drill	2
40 t Haul Truck	3
Block Holer	1
Scissor Lift (3m wide Platform)	4
Service Truck	1
Grader	1
Maintenance Vehicle with Boom and Basket	1
Personnel Carrier	1
Toyota Hilux (or equiv.)	5
Small Equipment	**# Units**
Stopers	10
Jacklegs	10

Mining personnel will be transported via the main access ramp from surface into the mine using vehicles or personnel carriers (carrying 6 to 8 people). During shift, personnel will primarily travel around the mine in personnel vehicles such as Toyota Landcruiser or Hilux vehicles equipped with bench seats in the rear for personnel transport. These vehicles will also be used by geology, engineering and mine staff to travel throughout the mine.

Materials and explosives will be transported using flat-bed service vehicles equipped with a boom crane. Fuel will be transported underground in a rubber tired fuel carrier.

A grader will ensure roadways are kept in good condition and that large rocks spilled from haul trucks and LHD's are removed from travelways.

16.9 Mining Personnel

All mine manpower, except for the technical staff, will be contractor employees.

Manpower estimates for the mine total approximately 186 people. These numbers include mine and surface employees, mine site management, engineers and geology personnel.

The direct mining manpower complement totals approximately 81 persons. Table 16-5 shows the direct mining personnel complement.

Table 16-5 Direct Mining Personnel Complement

Position	Shifts	Complement D/S	Complement A/S	Complement N/S	Absent Replacement	Complement Per Day	Total Complement
Development Miners	3	12	12	12	4	40	**40**
Longhole Driller	3	2	2	2	1	7	**7**
Longhole Driller Helper	3	2	2	2	1	7	**7**
Blaster	3	1	1	1		3	**3**
Blaster Helper	3	1	1	1	1	4	**4**
Stope LHD Operator	3	3	3	3		9	**9**
40 t Haul Truck operator	3	3	3	3	2	11	**11**
Total Direct Mine		**24**	**24**	**24**	**9**	**81**	**81**

The complement for mine services is estimated to be approximately 20 persons and the maintenance department 33 persons. Table 16-6 shows the mine services complement and Table 16-7 the mine maintenance department complement.

Table 16-6 Mine Services and Support Personnel Complement

Position	Shifts	Complement D/S	Complement A/S	Complement N/S	Absent Replacement	Complement Per Day	Total Complement
Serviceman	3	1	1	1	1	4	**4**
Grader Operator	D	1	1	1		3	**3**
Construction/Backfill Leader	3	1	1	1		3	**3**
Lamproom/Dryman	D	1	1	1		3	**3**
General Labourer	3	2	2	2	1	7	**7**
Total Mine Support Services		**6**	**6**	**6**	**2**	**20**	**20**

Table 16-7 Mine Maintenance Department Complement

Position	Shifts	Complement D/S	Complement A/S	Complement N/S	Absent Replacement	Complement Per Day	Total Complement
Leadhand Mechanic	3	1	1	1		3	**3**
Leadhand Electrician	3	1	1	1		3	**3**
Mobile Mechanic	3	1	1	1		3	**3**
Mechanic	3	1	1	1		3	**3**
Electrician	3	4	1	1		6	**6**
Electrician Helper	D	4				4	**4**
Instrumentation Technician	2	1	1			2	**2**
Instrumentation Helper	2	1	1			2	**2**
Parts Warehouseman	D	1				1	**1**
Welder	2	1	1			2	**2**
Welder Helper	2	2	2			4	**4**
Total Mine Maintenance Manpower		**18**	**10**	**5**	**0**	**33**	**33**

Contractor staff will include a Superintendent, 4 supervisors, a safety coordinator and a clerk.

All mine personnel will work three 8 hour shifts, 6 days per week.

Technical support for the mine will be provided by the geology and engineering departments. The geology department will continue to be responsible for mapping and interpretation, sampling of production drill holes, grade control and ore reserve estimations. There will be a separate exploration group to undertake exploration work on the property and to prove up new Mineral Resources for potential mining. The engineering department will continue to be responsible for mine planning and design, production scheduling, surveying, geotechnical design, and performance statistics for the mine and any other technical requirements that support the operation. The mine owner staff complement of 52 is presented in Table 16-8.

Table 16-8 Mine Staff Complement

Position	Total Complement
Mine Superintendent	1
Maintenance Superintendent	1
Mine Supervisor 1	3
Mine Supervisor 2	16
Maintenance Supervisor	1
Electrical Supervisor	1
Mine Services Supervisor 1	1
Mine Services Supervisor 2	4
Mine Trainer/H&S Coordinator	1
Maintenance Planner	2
Chief Engineer	1
Mine Planning Engineer	2
Ventilation	1
Blasting Engineer	1
Surveyor	2
Surveyor Helper	4
Chief Geologist	1
Mine Geologist	2
Geology Modeller	1
Geological Technicians	2
Muestrarios	4
Total Mine Staff	**52**

17. Recovery Methods

17.1 Process Description

The mineral processing plant described in the following sections is for the treatment of a silver-lead-zinc sulphide ore at a design throughput rate of 2000 tpd. The mineral processing plant will produce lead-silver and zinc concentrates, which will be transported off-site. Principal process parameters can be seen in Table 17-1. A general site map showing the location of the plant and sulphide tailings area is shown in Figure 17-1.

17.1.1 Primary Crushing

Run-of-mine (ROM) ore will be delivered by haul trucks from the mine. The ore will be dumped directly into the dump pocket of the primary crusher.

The dump hopper has a capacity of 125 tonnes which corresponds to three truckloads. A heavy-duty, hydraulic rock-breaker is provided to break up oversize boulders. The rock-breaker is controlled remotely by the crusher operator by means of a portable joy-stick controller. ROM ore discharges from the dump hopper onto a grizzly feeder and is fed into a jaw crusher.

The primary jaw crusher is capable of crushing large ROM ore up to 600 mm in diameter.

The crusher will operate at a nominal closed side setting of 70 mm to produce a product with a P80 of approximately 95 mm, at an average throughput rate of105 t/h. The jaw crusher discharges the crushed ore onto the crushed ore conveyor.

The crushed ore is transported to the crushed ore stockpile with a live capacity of 24 hours. Four vibrating pan feeders withdraw crushed ore from the stockpile and feed it to the semi-autogenous grinding (SAG) mill circuit.

17.1.2 SAG Milling

Crushed ore is fed into the SAG mill to produce a product with a P80 of approximately 3.1 mm, at an average throughput rate of 91 t/h. Process water is added to obtain a SAG mill discharge of 70% solids. SAG mill product is sent to the ball milling circuit. A trommel screen captures the oversize from the SAG mill and the oversize material is conveyed to undergo secondary (pebble) crushing. The trommel has an aperture size of 9.5 mm.

17.1.3 Secondary Crushing

Oversize from the SAG mill trommel screen is conveyed to the secondary crusher surge bin from where a pan feeder is used to feed the material to a cone crusher. The conveyor is equipped with a metal detector. The surge bin has a capacity of approximately of 25 tonnes.

The crusher will operate at a nominal closed side setting of 9 mm to produce a product with a P80 of approximately 10 mm, at an average throughput rate of approximately 24 t/h. The cone crusher discharges the secondary crushed ore onto the secondary crusher discharge conveyor. This conveyor feeds the material back to the SAG mill.

17.1.4 Ball Milling

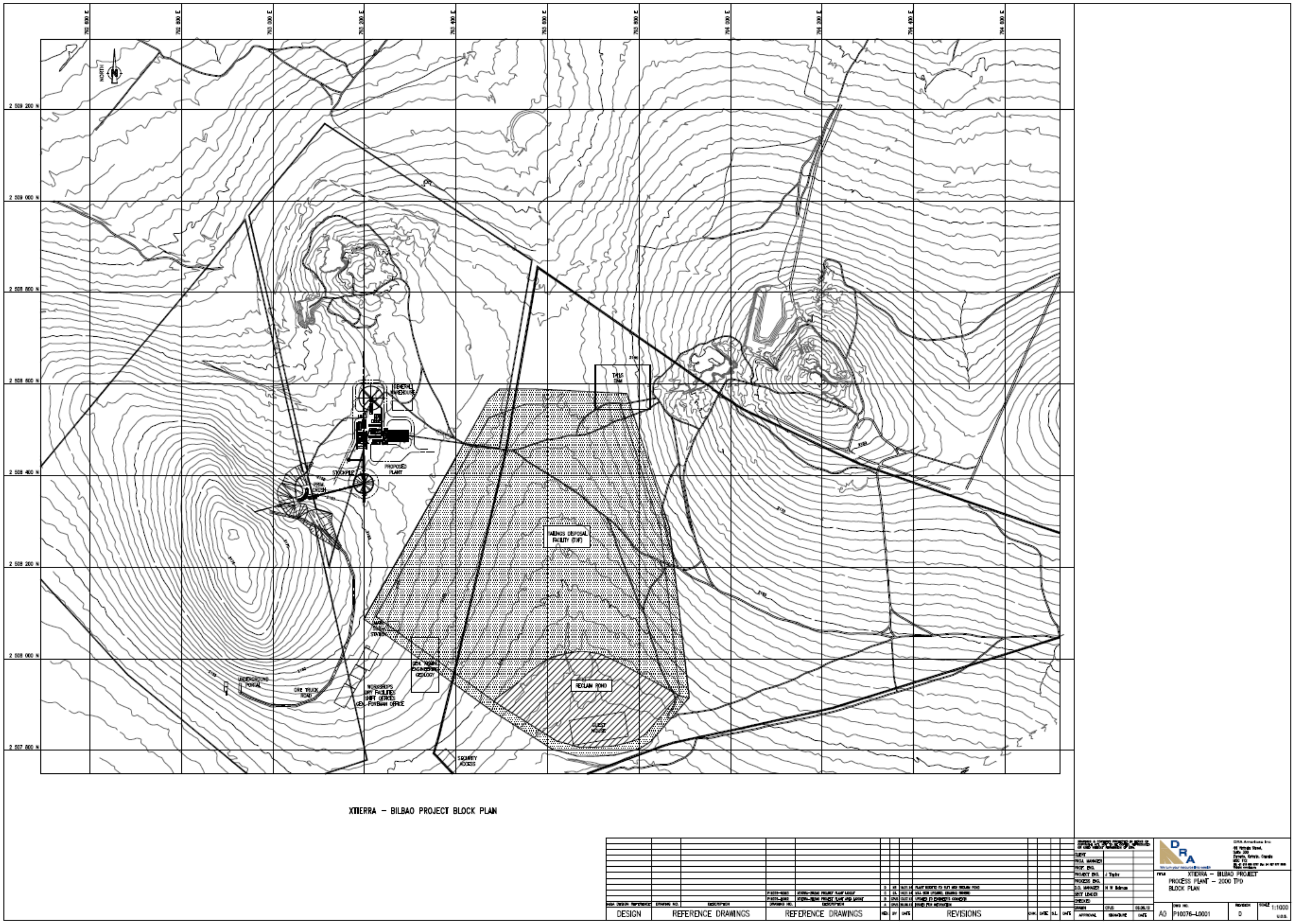
A ball mill is operated in closed circuit with a cluster of cyclones from which the overflows become lead rougher flotation feed and the underflow returns by gravity to the ball mill. The discharge from the mill passes through a trommel to remove tramp metal. The discharge underflow product from the ball mill trommel is directed by gravity via a launder into a pumpbox. Ball mill discharge, SAG mill discharge, lime, NaCN, $ZnSO_4$, and process water are collected in the same pumpbox. From this pumpbox, the slurry is pumped to the cyclone cluster.

Table 17-1 Principal Process Parameters

Parameter	Unit of Measure	Rate
Plant Capacity		
Annual	thousand tonnes/year	720,000
Daily	tonnes/day	2,000
Ore Grade		
Silver	grams/tonne ore	63.96
Lead	percent	1.40
Zinc	percent	2.10
Operating Parameters		
Crush Size	microns (80% passing)	95,000
Primary Grind Size	microns (80% passing)	100
Pb Regrind Size	microns (80% passing)	43
Zn Regrind Size	microns (80% passing)	37
Reagent Consumptions		
Lime	kilogram/tonne ore	3.06
Sodium Cyanide	kilogram/tonne ore	0.055
Zinc Sulphate	kilogram/tonne ore	0.385
Copper Sulphate	kilogram/tonne ore	0.525
A211	kilogram/tonne ore	0.043
Aerophine 3418A	kilogram/tonne ore	0.020
MIBC	kilogram/tonne ore	0.053
Unifroth 250	kilogram/tonne ore	0.038
Aero 3894	kilogram/tonne ore	0.0025
Flocculant	kilogram/tonne ore	0.051
Flotation Lab Time		
Pb Circuit		
Pb Rougher	minutes	6
1st Pb Cleaner	minutes	3
2nd Pb Cleaner	minutes	6
3rd Pb Cleaner	minutes	6
1st Pb Cleaner-Sca	minutes	3
Zn Circuit		
Zn Rougher	minutes	6
1st Zn Cleaner	minutes	3
2nd Zn Cleaner	minutes	6
3rd Zn Cleaner	minutes	3
1st Zn Cleaner-Scav	minutes	3

Parameter	Unit of Measure	Rate
Recovery		
Lead Concentrate		
Silver	percent	73.4
Lead	percent	90.6
Zinc	percent	5.7
Zinc Concentrate		
Silver	percent	6.7
Lead	percent	0.8
Zinc	percent	76.7
Concentrate Grade		
Lead Concentrate		
Silver	grams/tonne concentrate	1,335.00
Lead	percent	54.00
Zinc Concentrate		
Silver	grams/tonne concentrate	91.40
Zinc	percent	43.00
Production		
Lead Concentrate	dry tonnes/year	16,913
Contained Silver	thousand troy ounces/year	726
Contained Lead	pounds/year	20,135,243
Zinc Concentrate	dry tonnes/year	26,966
Contained Silver	thousand troy ounces/year	79
Contained Zinc	pounds/year	25,563,714

Figure 17-1 General Site Layout



Ball mill discharge is targeted for a P80 of approximately 100 microns. Lime is added to control the pH of the flotation feed to 7.7. Process water is added to obtain a ball mill discharge of 70% solids. The ball mill trommel has an opening size of 9.5mm.

The cyclone cluster is fed by a slurry pump equipped with a variable speed drive. The underflow is collected in a rubber-lined launder and flows by gravity to the ball mill feed spout. The cyclone overflow product (P80 of 100 microns, 35% solids) flows by gravity to the lead rougher conditioner tank.

17.1.5 Lead Rougher Flotation

Lead rougher flotation feed conditioning is carried out in the lead rougher feed conditioner where cyclone overflow, lime, MIBC and 3418A are agitated for 2.5 minutes at a pH of 8.0.

Slurry from the conditioning tank then overflows to the first cell of the bank of lead roughers. The flotation cells are stepped for gravity flow. Lead rougher concentrate froths are collected from one side of the flotation cells and flow by gravity into pipe launders leading to the lead rougher concentrate pumpbox. The lead rougher concentrate is then pumped to the lead regrind circuit. Lead rougher tails is collected in a pumpbox and is pumped to the zinc recovery circuit.

MIBC and A211 are added to the lead rougher flotation cell bank. Process water is piped along the flotation cell bank for use as spray water for froth control and dilution.

17.1.6 Lead Concentrate Regrind

The lead concentrate regrind circuit consists of a cyclone cluster ahead of a regrind ball mill operating in closed circuit. The overflow of the cyclone cluster becomes the lead cleaner flotation feed and the underflow returns by gravity to the regrind mill. The discharge from the mill passes through a trommel and the discharge underflow product from the regrind mill trommel is directed by gravity via a launder into a pumpbox. Lead rougher concentrate, and process water, along with the recycle streams of the lead cleaner scavenger concentrate and second lead cleaner tails are collected in the same pumpbox. From this pumpbox, the slurry is pumped to the cyclone cluster. Lime, NaCN, ZnSO4, A3894 are added directly into the regrind mill.

Regrind mill cyclone overflow is targeted for a P80 of approximately 43 microns. Lime is added to control the pH of the slurry to 8.4. Process water is added to obtain a regrind mill discharge of 53% solids. The regrind mill trommel has an opening size of 9.5mm.

The cyclone cluster is fed by a slurry pump equipped with a variable speed drive. The underflow is collected in a rubber-lined launder and flows by gravity to the regrind mill feed spout. The cyclone overflow product (P80 of 43 microns, 32% solids) flows by gravity to the Lead 1st cleaner.

17.1.7 Lead Cleaner and Cleaner-Scavenger Flotation

Lead regrind cyclone overflow is pumped to the first lead cleaner. The first cleaner flotation consists of two stages. Lime, 3418A are added only to the first stage and MIBC is added to both stages. The pH is controlled between 8.7 and 9.0. The first lead cleaner concentrate is collected in a pumpbox and is pumped to the second lead cleaner. First lead cleaner tails gravity flows to the lead cleaner-scavenger.

The second lead cleaner flotation is performed in three stages. Lime is only added to the first stage and MIBC are added to the first and last stages. The pH is controlled between 9.5 and 10.5. The second lead cleaner concentrate is collected in a pumpbox and is pumped to the third lead cleaner. Second lead cleaner tails flows by gravity to a pumpbox and is recycled back to the lead regrind circuit.

The third lead cleaner flotation is done in three stages. Lime is only added to the first stage and MIBC are added to the second and third stages. The pH is controlled between 9.0 and 10.0. The third lead cleaner concentrate, which is the final lead concentrate product, is collected in a pumpbox and is pumped to the dewatering circuit. Third lead cleaner tails flows by gravity to the second cleaner flotation.

The lead cleaner-scavenger flotation is done in a single stage to which MIBC, A211 and 3418A are added. The lead cleaner-scavenger concentrate is collected in the second lead cleaner tails pumpbox and is pumped back to the lead regrind circuit. Lead cleaner-scavenger tails flows by gravity to a pumpbox and is pumped to the zinc recovery circuit.

17.1.8 Zinc Rougher Flotation

Zinc rougher flotation feed conditioning is carried out in two stages. Lime is added to the first tank to target a pH of 10.5 and $CuSO_4$ is added to the second tank.

Slurry from the second conditioning tank overflows to the first cell of the bank of zinc roughers. The flotation cells are stepped for gravity flow. The levels of the rougher cells will be automatically controlled by dart valves.

Zinc rougher concentrate froths are collected from one side of the flotation cells and flow by gravity into pipe launders and eventually to the zinc rougher concentrate pumpbox. The zinc rougher concentrate is then pumped to the zinc regrind circuit. Zinc rougher tails is collected in a pumpbox and is pumped to the tailings thickener.

Lime, A211 and U250 are added to the zinc rougher flotation cell bank. Process water is piped along the flotation cell bank for use as spray water for froth control and dilution.

17.1.9 Zinc Concentrate Regrind

The zinc concentrate regrind circuit consists of a cyclone cluster ahead of a regrind ball mill operating in closed circuit. The overflow of the cyclone cluster becomes the zinc cleaner flotation feed and the underflow returns by gravity to the regrind mill. The discharge from the mill passes through a trommel and the discharge underflow product from the regrind mill trommel is directed by gravity via a launder into a pumpbox. Zinc rougher concentrate, and process water, along with the recycle streams of the zinc cleaner scavenger concentrate and second zinc cleaner tails are collected in the same pumpbox. From this pumpbox, the slurry is pumped to the cyclone cluster. Lime is added directly into the regrind mill to target a pH of 10.1.

Regrind mill cyclone overflow is targeted for a P_{80} of approximately 37 microns. Process water is added to obtain a regrind mill discharge of 53% solids. The regrind mill trommel has an opening size of 9.5mm.

The cyclone cluster is fed by a slurry pump equipped with a variable speed drive. The underflow is collected in a rubber-lined launder and flows by gravity to the regrind mill feed spout. The cyclone overflow product (P_{80} of 37 microns, 35% solids) flows by gravity to an agitated tank where $CuSO_4$ is added before it continues by gravity flow to the first zinc cleaner.

17.1.10 Zinc Cleaner and Cleaner-Scavenger Flotation

Zinc regrind cyclone overflow is gravity fed to the first zinc cleaner. The first cleaner flotation will consist of two stages. Lime and A211 are added only to the first stage and U250 is added to both stages. The pH is controlled to 10.5. The first zinc cleaner concentrate is collected in a pumpbox and is pumped to the second zinc cleaner. First zinc cleaner tails gravity flows to the zinc cleaner-scavenger.

The second zinc cleaner flotation comprises of three stages. Lime is only added to the first stage and A211 and U250 are only added to the last stage. The pH is controlled to 11.3. The second zinc cleaner concentrate is collected in a pumpbox and is pumped to the third zinc cleaner. Second zinc cleaner tails flows by gravity to a pumpbox and is recycled back to the zinc regrind circuit.

The third zinc cleaner zinc flotation has a two stage configuration. Lime is only added to the first stage and U250 is added to the second stage. The pH is controlled to 12.0. The third zinc cleaner concentrate, which is the final zinc concentrate product, is collected in a pumpbox and is pumped to the dewatering circuit. Third zinc cleaner tails flows by gravity to the second cleaner flotation.

The zinc cleaner-scavenger flotation is done in one stage to which A211 and U250 are added. The zinc cleaner-scavenger concentrate is collected in the second zinc cleaner tails pumpbox and is pumped back to the zinc regrind circuit. Zinc cleaner-scavenger tails flows by gravity to a pumpbox and is pumped to the tailings.

17.1.11 Lead Dewatering and Filtration

The final lead concentrate from lead cleaner flotation is pumped to the lead concentrate thickener to produce a concentrate of 76% solids. The thickener is of high-rate design with the addition of flocculant. Thickener overflow flows by gravity to the process water tank and thickener underflow is pumped by a peristaltic pump into the lead concentrate thickener underflow tank. This underflow tank provides a surge capacity between the lead flotation circuit and the lead dewatering circuit.

The thickened lead concentrate is pumped to feed the lead concentrate pressure filter to produce a final lead concentrate filter cake with 7% moisture. The filter cake is discharged onto a stockpile and is transported to the lead loadout area by a front-end loader. The filtrate is recycled back to the lead concentrate thickener.

17.1.12 Zinc Dewatering and Filtration

The final zinc concentrate from zinc cleaner flotation is pumped to the zinc concentrate thickener to produce a zinc concentrate of 76% solids. The thickener is of high-rate design with the addition of flocculant. Thickener overflow flows by gravity to the process water tank and thickener underflow is pumped by a peristaltic pump to the zinc concentrate thickener underflow tank. This underflow tank provides a surge capacity between the flotation and dewatering areas of the zinc circuit.

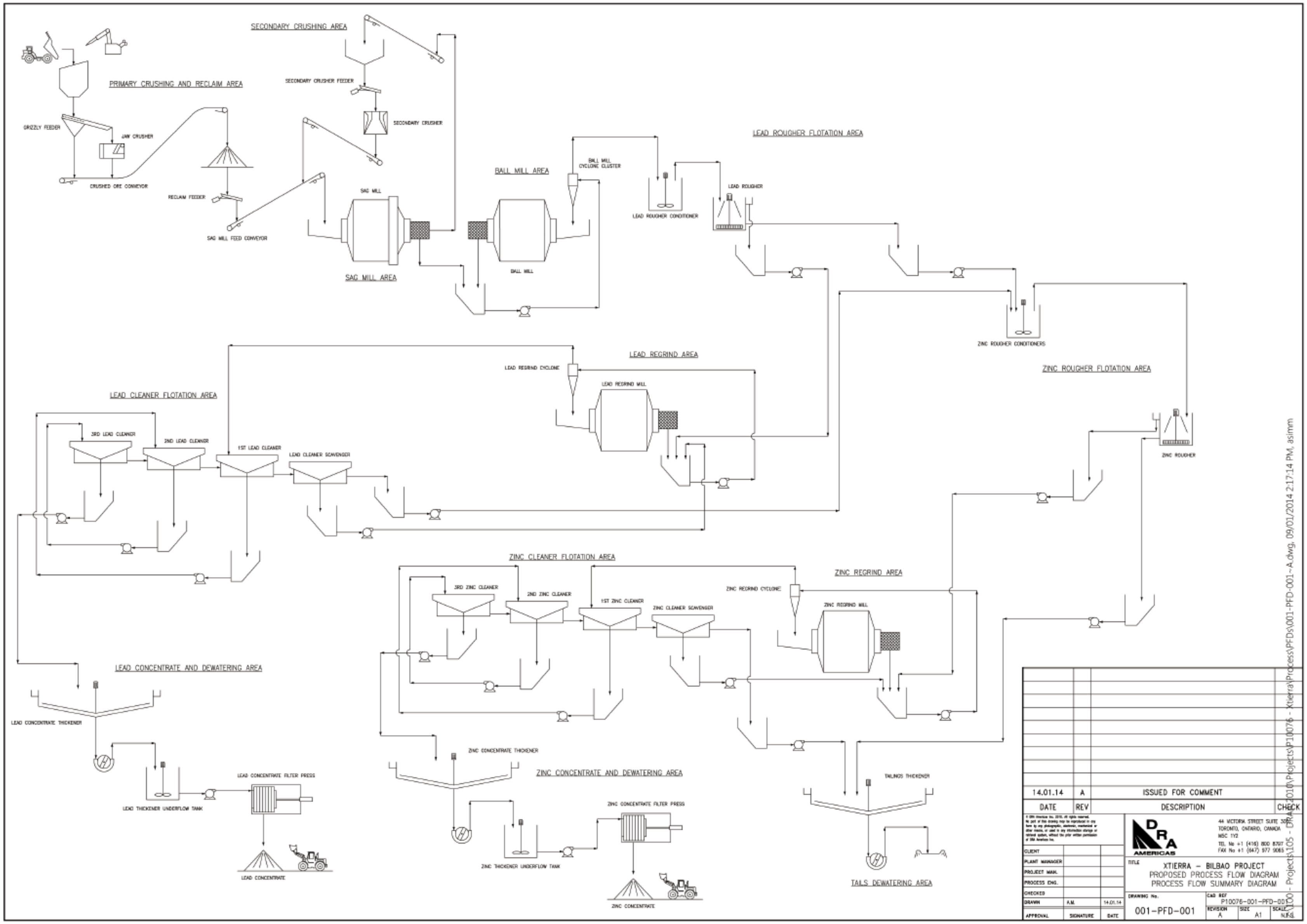
The thickened zinc concentrate is pumped to feed the zinc concentrate pressure filter to produce a final zinc concentrate filter cake with 8% moisture. The filter cake is discharged onto a stockpile and is transported to the zinc loadout area by a front-end loader.

17.1.13 Tailings Dewatering

Tailings from the zinc flotation circuit is pumped to the tailings thickener to produce a thickened tailings with 65% solids. The thickener is of conventional design with the addition of flocculant. Thickener overflow flows by gravity to the process water tank and thickener underflow is pumped to the tailings treatment facility.

A simplified process flow diagram for a 2,000 tpd processing rate can be seen in Figure 17-2.

Figure 17-2 Simplified Process Flow Diagram



18. Infrastructure

18.1 Regional Infrastructure

The state of Zacatecas has a well-developed highway system including several Federal highways and well-maintained primary and secondary roads. A branch of the Mexican National railroad system crosses the central part of the State through the city of Zacatecas connecting Mexico City with Ciudad Juárez. Zacatecas is a large modern city with excellent facilities for business and a pool of experienced mining labour. The Zacatecas International airport is located 28km northwest of the capital with daily connections to Mexico City, Tijuana, Los Angeles, and less frequent services to other destinations in the United States.

The Pánfilo Natera district is located in a developed area of Zacatecas with good infrastructure and services. There are no obvious impediments to mine development in the district. Mining and agriculture have co-existed since early colonial times.

Greater detail on project accessibility and local infrastructure and resources can be seen in Section 4.

18.2 Tailings Disposal Facility

Kam, S. & Welch, D., of Golder Associates Ltd (Golder), undertook a study on the potential sites chosen for tailings disposal at the Bilbao project and presented their findings in a report of 13th July 2010 entitled “Draft-Scoping Study, Tailings Disposal Facility, Rpt # 10-1118-0032”, pp15. In order that the TDF provides secure storage of the tailings over the long term, with minimal adverse environmental impact, the study identified several objectives that should be met namely:

- Ensuring the physical and chemical stability of the TDF and the impounded tailings.
- Minimization of water use overall.
- Protection of the groundwater aquifer.
- Minimizing discharge from the site.
- Ensuring that when the mine closes the procedures meet regulatory requirements in addition to applicable standards and those codes acceptable to international financial institutions are adhered to.

In order to meet the above objectives Golder suggested several measures to obviate any potential problems including the following more important aspects:

- Planning the design of the TDF to accommodate a 500 year rainfall event.
- Minimizing seepage by providing a geo-membrane over the footprint of the TDF.
- Minimizing the use of freshwater.
- Keeping the size of the TDF as small as possible.
- Whilst the area is one of the lowest seismic risk zones in the whole of Mexico, plan safety measures to accommodate any such earthquake event.
- At mine closure, cap the tailings to reduce potential dusting.
- Plant vegetation and/or reseed the area of the TDF and Waste Dumps.

The study suggested that it would be appropriate to undertake drilling over the proposed tailings pond to ensure that no mineralization was sterilized beneath it or that there were structural impediments to the particular area chosen. The Company has drilled two diamond drill holes X87 & X90 in the proposed area for the TDF and found no impediments on either of these counts.

A second study was carried out by Golder in 2013 entitled “Prefeasibility Study Tailings Disposal Facility”. The overall objective of the prefeasibility study is to update the design of the TDF using information that is currently available. The prefeasibility study reflects the changes to the mining method (open pit versus underground mining), current resources estimates and the metallurgical process that has been finalized by DRA Americas (DRA).

A preliminary TDF design for 3.1 Mt of tailings has been proposed. The TDF will be lined to minimize water loss and groundwater impact. The perimeter dams will be raised in two stages after start up during the 8 year mine life. The TDF will be a no discharge facility with all water reclaimed for processing. The water deficit will be met by groundwater wells to be developed off site.

The following studies are recommended during the next stage of study:

- Geotechnical boreholes should be drilled for a better understanding of the geological, geotechnical, and hydrogeological conditions of this area. Additional samples should be taken for testing.
- An investigation into the quantity of material available from the potential borrow sources should be completed.
- A seismic hazard study of the site should be completed to evaluate the peak ground accelerations for the proposed design events.
- Tailings samples should be analysed for the geotechnical properties (consolidation, permeability and shear strength);
- A number of assumptions have been made in the water balance analysis (mine water inflow, water consumption, site footprint etc.). They should be confirmed to reduce operating uncertainty and to improve reliability of cost estimate.
- Additional geochemical testing on waste rock and tailings should be completed to determine water quality impact on the metallurgical process given the water is continually recycled. Treatment of water for discharge to the environment should be considered during extreme wet climatic conditions.
- A detailed TDF closure plan should be developed that will prevent impacted water from being discharged to the environment in the long term.

This full report can be seen in the document entitled “Prefeasibility Study Tailings Disposal Facility” dated February 2014.

18.3 Plant Site/Processing Location

The plant and processing facilities will be sited to the north of the proposed portal and west of the tailings disposal facility, as seen in Figure 17-1. The reason for this is that the basaltic substrate there will furnish a stable foundation for such buildings as are required for the plant. Furthermore it is known from diamond drilling that there is no mineralization which would be sterilized in that area.

18.4 Underground Facilities

The majority of underground infrastructure will be associated with facilities located on the 1860 Level and include a breakdown maintenance shop, main dewatering sumps, fuel and lube stations, explosives magazine, refuge station and storage areas.

18.4.1 Ventilation

Ventilation will be provided to the mine by a Fresh Air Raise (FAR) and the main ramp from surface. A network of lateral development on each level will connect the mining areas to a Return Air Raise (RAR).The mining operation to support the mining equipment fleet will require ventilation air volumes of approximately 210 to 230 cu. metres per second (450,000 to 500,000 cfm). The ventilation system will consist of a push-pull system utilizing the ventilation raises and the main access ramp.

18.4.2 Electrical Distribution

Primary electrical power for the mine will be provided from the main surface substation connected to the outside powerline. The powerline will be connected to a surface substation located near to the mine portal. Power from the main substation will feed the main underground power line, a 500 mcm cable, installed in the main access ramp from surface.

18.4.3 Compressed Air

Compressed air will be supplied by 2 compressors in enclosures located in a small covered structure, near the ramp portal. Service water will be sent underground in a pipeline located in the trackless access ramp from surface. This will feed the main distribution lines on the levels, which will send water to the stope access crosscuts. The mine will also have a communications network to provide voice communications and some PLC monitoring within the mine.

18.4.4 Water Service

The underground mine will require approximately 80 million litres of service water per year for use in drilling, dust suppression, etc.

Water will be sent underground in a pipeline located in the trackless access ramp from surface. This will feed the main distribution lines on the levels, which will send water to the stope access crosscuts. Water pressures and volumes will be controlled by installing water stations, at appropriate vertical intervals within the mine, which will house a transfer station and holding tanks.

19. Market Studies and Contracts

19.1 Market Studies

The following has been prepared by Micon International Limited (Micon) on behalf of RungePincockMinarco (Canada) Ltd. (RPM).

It is the understanding of Micon that Xtierra. (Xtierra) has not commissioned or undertaken a formal market study relating to potential metal production from the Bilbao project. However, Xtierra has obtained indicative terms and conditions from MRI Trading AG (MRI) of Zug, Switzerland, relating to the delivery of zinc and lead concentrates from the Bilbao project to the port of Manzanillo, Mexico. The terms and conditions are dated 18 May, 2012. Micon considers that the document demonstrates that, based on the typical specifications put forward by Xtierra to MRI, the zinc and lead concentrates are likely to be saleable and acceptable to smelters. The report by ConsuMet, dated 19 April, 2013 concluded that the analysis of concentrate samples from the locked cycle test did not indicate the presence of elements of concern in terms of concentrate marketing. However, it is possible that bismuth in the lead concentrate and iron and cadmium in the zinc concentrate will be penalized. Details of the terms and conditions proposed by MRI are confidential to Xtierra.

Zinc and lead concentrates are widely traded internationally. The majority of concentrate sales are priced on the basis of London Metal Exchange (LME) or similar quotes, as reported by organizations such as Kitco (www.kitco.com), Metal Bulletin and Mining Journal. The LME is a terminal market and metals pricing is transparent. Silver is generally contained as a co- or byproduct within base and precious metal concentrates. Prices are determined by the noon fix of The London Silver Market Fixing (www.silverfixing.com), known as the London fix, and which is regarded as an acceptable pricing mechanism for those involved in the market. The London fix is also reported by Kitco, Metal Bulletin and Mining Journal, among others.

19.1.1 Zinc Supply and Demand

The US Geological Survey (USGS) estimated world mine output of zinc in 2012 at 13 million tonnes contained metal. The largest single producer is China where output reached 4.6 million tonnes contained zinc and where production has increased steadily over the past decade. Mexico is a significant producer of mined zinc, ranking ninth among the 10 largest producers in 2012, as shown in Table 19-1.

Table 19-1 World Mine Zinc Production
(Thousand tonnes zinc metal in concentrate and direct shipping ore)

	2008	2009	2010	2011	2012
Australia	1,519	1,290	1,479	1,515	1,490
Bolivia	384	431	411	427	430
Canada	751	699	649	612	640
China	3,340	3,330	3,700	4,310	4,600
India	613	695	700	710	690
Ireland	398	386	342	340	345
Kazakhstan	446	442	459	495	420
Mexico	397	384	518	632	630
Peru	1,603	1,513	1,470	1,256	1,270
United States	778	736	748	769	748
Others	1,571	1,594	1,724	1,734	1,737
Total	**11,800**	**11,500**	**12,200**	**12,800**	**13,000**

USGS, 2013a, 2012a.

Total zinc smelter output in 2011 was 13.1 million tonnes, of which just under 5 million tonnes was primary metal. Significant primary zinc smelting capacity is located in China where production of both primary and secondary metal totalled 5.22 million tonnes in 2011. Other major producers of primary zinc are Republic of Korea (producing 0.829 million tonnes in 2011), Australia (0.507 million tonnes) and Japan (0.444 million tonnes). Mexico's primary zinc production was 0.322 million tonnes in 2011 (USGS, 2012a).

Concentrates are smelted and refined and used in a variety of applications of which galvanizing of steel and alloying in brass and bronze are the two most important. These are followed by zinc alloys for die-casting, chemicals and zinc semimanufactures. Demand for zinc generally follows trends in global economic growth. (USGS, 2012a).

19.1.2 Lead Supply and Demand

In volume terms, world mine production of lead is less than half that of zinc. The USGS estimated world mine output of lead in 2012 at 5.2 million tonnes contained metal. As for zinc, the largest single producer is China where output reached 2.6 million tonnes contained lead and where production has also increased steadily over the past decade. Mexico is a significant producer of mined lead, ranking fourth among the ten largest producers in 2012, as shown in Table 19-2.

Table 19-2 World Mine Lead Production
(Thousand tonnes lead metal in concentrate)

	2008	2009	2010	2011	2012
Australia	645	566	625	621	630
Bolivia	82	85	73	100	110
China	1,500	1,600	1,850	2,350	2,600
India	87	92	97	115	118
Mexico	101	144	192	220	245
Peru	345	302	262	230	235
Poland	88	80	64	60	60
Russia	60	70	97	105	105
Sweden	60	69	68	62	60
United States	410	406	369	342	345
Others	502	456	463	495	692
Total	**3,880**	**3,870**	**4,160**	**4,700**	**5,200**

USGS, 2013b, 2013c.

Total lead refinery output in 2011 was 10.2 million tonnes, of which 4.69 million tonnes was primary metal. Significant primary lead refining capacity is located in China where primary refined lead production was 3.22 million tonnes in 2011. Other major producers of primary refined lead are Republic of Korea (which produced 0.20 million tonnes in 2011), Australia (0.187 million tonnes), United Kingdom (0.15 million tonnes) and Germany (0.135 million tonnes). Mexico's primary lead refinery production was 0.111 million tonnes in 2011 (USGS, 2013c).

Concentrates are smelted and refined in primary facilities. The major end-use for refined lead, accounting for approximately 80% of total demand is in lead-acid batteries for starting-lighting-ignition and industrial applications. Rolled and extruded metal products account for a further 6% of demand, and pigments for 5%. Cable sheathing, lead shot/ammunition and alloys account for the majority of remaining demand (International Lead and Zinc Study Group, ILZG, www.ilzsg.org).

Secondary lead refinery production has exceeded primary output since around 2002. The feedstock is principally scrapped lead-acid batteries. In 2011, the USGS reported secondary refined lead production at 5.27 million tonnes, of which China and the United States accounted for just under 50% (USGS 2013c).

19.1.3 Silver Supply and Demand

Silver is associated with polymetallic deposits and, in descending order of importance, is recovered from lead-zinc, copper and gold mining operations. (USGS, 2013d).

Mexico is the largest single producer, followed by China. See Table 1.3. Until the mid-2000s, Peru held either first or second place among mined silver producers, together with Mexico.

Table 19-3 World Mine Silver Production
(Tonnes silver content)

	2008	2009	2010	2011	2012
Australia	1,926	1,635	1,864	1,725	1,900
Bolivia	1,114	1,326	1,259	1,210	1,300
Canada	728	631	596	572	530
Chile	1,405	1,301	1,287	1,290	1,130
China	2,800	2,900	3,500	3,700	3,800
Mexico	3,236	3,554	4,411	4,150	4,250
Peru	3,686	3,854	3,640	3,410	3,450
Poland	1,161	1,207	1,181	1,167	1,170
Russia	1,132	1,313	1,356	1,350	1,500
United States	1,250	1,250	1,280	1,120	1,050
Others	2,962	3,229	3,426	3,606	3,920
Total	**21,400**	**22,200**	**23,800**	**23,300**	**24,000**

USGS, 2013d, 2013e.

As a co- or by-product, silver is recovered mainly from the smelting and refining of lead-zinc, copper and gold concentrates. Recycled silver scrap accounted for approximately 25% of total silver supply in 2011 (The Silver Institute, www.silverinstitute.com). Producer hedging and net government sales also impact total silver supply.

Silver is used in a wide range of industrial applications as well as in photography, silverware, jewellery and coins. Industrial uses have increased steadily through the 2000s but silver in photography has declined rapidly with the widespread adoption of digital photography. Traditional industrial uses include alloys and solders, catalysts and electrical and electronic applications.

19.1.4 Metal Price Trends

For both zinc and lead prices, 2007 saw peaks significantly higher than had occurred in the previous 15 to 20 years. Although this was followed by sharp corrections, prices for both metals recovered through 2009 and have remained relatively steady since then, in spite of generally uncertain economic conditions.

Zinc prices peaked in 2007, briefly exceeding US$2.00/lb, but fell to around US$0.50/lb early in 2009. From mid-2011, zinc prices have generally fluctuated between US$1.00/lb and US$0.80/lb. Lead prices have followed a generally similar trend. Since mid-2011, the range has also been between US$1.00/lb and US$0.80/lb, but generally US$0.05-0.10/lb higher than for zinc.

Since 2000, silver prices increased from around US$5.00/oz to US$11.57/oz in 2006 and US$14.66 in 2009. In the three-year period from 1 November, 2010, average monthly highs of US$41.99 and US$40.30 were recorded in April and August, 2011, respectively. Through the first 10 months of 2013, prices have trended downwards from approximately US$31/oz in January, to US$21.92/oz in October. Investment demand for silver has maintained prices at relatively high levels, reflecting concerns relating to global economic conditions and offsetting the effects of softer industrial demand.

19.1.5 Prices Used for Economic Analysis

At the request of Xtierra, RPM has based its economic analysis of the Bilbao project on three-year average metal prices.

For the three-year period ending 31 October, 2013, the rolling average prices based on LME cash buyer quotes for zinc and lead, and as reported by Kitco on www.kitco.com for silver are as follows:

Zinc	US$0.92/lb
Lead	US$1.00/lb
Silver	US$30.38/oz

In order to test the sensitivity of the Bilbao project to metal demand and, therefore, decreased and increased metal prices over the projected life of the operation, reductions and increases on the three-year average prices are evaluated in Section 22.

19.2 Contracts

At the time of writing of this report, Micon understands that there are no contracts in place that are material to the issuer relating to property development or marketing of concentrates from the Bilbao project.

20. Environmental Studies, Permitting and Social or Community Impact

The 1998 *Ley General del Equilibrio Ecologico y Proteccion al Ambiente (*General Law of Ecological Balance and Environmental Protection) and subsequent amendments form the guiding basis for environmental policy in Mexico. Environmental regulation of mining projects is administered by the Secretary of Environment and Natural Resources (SEMARNET). All mineral rights are held by the State, while surface property rights are held by individuals or “ejidos” (historical communities living in a given area). Individuals or ejidos are allowed by law to sell these surface rights, allowing mining activity to take place after securitization of mineral right concessions from the State.

The Bilbao project (the “Project”) is located in the Municipality of General Panfilo Natera, in the State of Zacatecas, at an elevation of between 2,145 and 2,160 meters above sea level. The small community of Panfilo Natera is located approximately 3.9 km from the site. The Bilbao mineral deposit is covered by several claims comprising a total of 1,407 hectares. This area is surrounded by flat active and fallow farmlands with a relatively flat morphology and overall arid conditions. The majority of irrigated farmland in the region is owned by the ejidos of Panfilo Nater and Ojo Caliente, with the balance privately owned.

The State of Zacatecas has experienced centuries of mining development, and the overall region has a high density of active and inactive mine workings. Access to the Project is straightforward, with immediate connection from a paved highway. The Bilbao site has undergone historic development, which is evidenced by an abandoned shaft present on site and numerous existing open pits. Oxide “glory hole” materials were historically accessed via crude excavation at these open pits, and from underground workings at two levels accessed via the abandoned shaft and the open pit.

20.1 Environmental Studies

Several environmental or environmentally-related studies have been developed for the Bilbao project. These studies have provided detail on biodiversity baseline conditions, groundwater resources available in the region, and information on the potential for the Project to result in environmental contamination. Existing studies are summarized in the following subsections. A complete analysis of environmental baseline conditions and anticipated Project impacts, as well as the design of appropriate mitigations, will be described in the pending Environmental Impact Statement (in Mexico the “*Manifestacion de Impacto Ambiental*” or “ MIA”). Development and approval of the MIA is a requirement of Mexican regulations, as described in Section 20.3. The MIA will be developed by the environmental consultancy Soluciones de Ingenieria y Calidad Ambiental (SIICA).

20.1.1 Biodiversity Studies

The Mexican environmental consultancy Bufete de Servicios Tecnicos Forestales y de Fauna Silvestre prepared a 2006 study entitled Bilbao Project, Biologic, Climatic and Access Route Aspects. This study presented baseline flora and fauna data for the project site, as well as climate information (average rainfall and temperatures). Six species of cacti were identified which have legal protection status. A subsequent report by the same firm entitled “*Aviso de Apego a la NOM-120-SEMARNET-1997, Para Actividades de Explotacion Minera del Proyecto Bilbao*” was filed with SEMARNET in 2006, detailing efforts that would be undertaken by the Project to avoid any sensitive cacti during exploration drilling, and to reclaim drilling pad locations. These mitigations have been implemented and appropriate rehabilitation is undertaken at the completion of all exploration drilling activity. Additional biodiversity studies will be detailed in the MIA.

20.1.2 Hydrology Studies

Average precipitation at the Project site is approximately 412 mm/year. This value is based on climate records available from the nearby San Pedro Pedra Gorda climate station, which is located approximately 31 km from the Project site. The San Pedro Pedra Gorda climate station has the longest precipitation data set available in the region with records collected since 1943. Average evapotranspiration rate at this same location is 1,486 mm/year, or approximately 3.6 times the rate of precipitation.

The Bilbao Project site is located in the north-eastern quadrant of the Ojo Caliente Aquifer, which in turn borders the La Blanca Aquifer to the east and Chupaderos Aquifer to the north. Site drainage at Bilbao is generally to the south in the direction of a small pond created by construction of an earthen embankment. This pond is normally dry except following high precipitation events.

In 2009 Bilbao Resources contracted with Schlumberger Water Services (“SWS”) to characterize and identify a potential water source for the Project. The results of this initial investigation were published in a report entitled “Phase I Hydrogeologic Study”, which concluded that the Project would need to rely exclusively on ground water for needed make-up water during operations, as very little surface water exists in the region.

A follow-on Phase 2 Hydrologic Assessment was issued by SWS in July, 2011. The report entitled “Phase 2 Hydrologic Assessment” identified a total of 184 production wells in existence within a study area of 10km from the Project site. The majority of these wells (>90%) are dedicated to agricultural use, with the remainder used for municipal water supply, livestock, or other private uses. The exploitable hydrogeologic unit of the identified aquifers consists of basin-fill Quaternary and Tertiary materials. Ranges of depth to groundwater in the Ojo Caliente basin are from 50 – 90 meters below ground surface. From the time period of 1997 – 2007 water levels dropped between 0.4 to 1.8 meters/year in the Ojo Caliente aquifer, reflecting high a high rate of overexploitation. Similar trends are evident in the adjacent La Blanca and Chupaderos Aquifers. Potential yields from wells identified in the Phase 2 Hydrologic Study are estimated to be within a range of 10 to 50 l/s. Groundwater quality is generally good and suitable for human consumption.

Since the early 1960’s there has been a ban on additional groundwater exploitation in the Municipality of General Panfilo Natera. As a result the Project will be required to purchase water rights from existing users. There are two options to obtain these water rights: (1) purchase an existing well that has been previously permitted, then pipe the water to the Project site; or (2) purchase a permitted well, then transfer the groundwater concession rights for this well to a new well located nearer to the mine site. The second option requires identification of a suitable location for pumping of groundwater, then purchase and transfer of an existing concession to allow production to occur at the newly identified location.

The 2011 Phase 2 Hydrologic Assessment provides the following summary conclusions:

- A program of baseline groundwater quality and water levels should be established, to allow environmental monitoring over time once the mine is in operation.
- A hydrogeologic drilling investigation should be completed at candidate well locations near the mine site, as data obtained from this investigation would be necessary to allow any water rights transfers. Water rights in Mexico are administered by the *Comision Nacional de Agua* (“National Water Commission” and “CONAGUA”).
- Six target zones are identified near the Project area for exploratory drilling (at Las Borregas, Bilbao, and La Ardilla).
- Potential groundwater inflow towards the mine should be investigated to incorporate any necessary dewatering costs in feasibility programs.

20.1.3 Geochemistry

Studies modelling anticipated tailings and waste rock geochemistry were completed by Golder in a January 2014 report entitled “Geochemical Results of Waste Rock and Tailings Samples”. Findings from this report will be included in the final MIA, and are summarized in this subsection. Results from the testing program are suitable for making broad decisions regarding mineral waste management at a pre-feasibility level.

Geochemical modelling of waste rock samples has been performed to identify the potential for acid rock drainage. A total of 19 waste rock samples were collected by Xtierra geologists from boreholes drilled to intersect the proposed ramp to underground works. This ramp will be approximately 2,416 in length, and will intersect multiple igneous and sedimentary waste rock types. Sampling locations for waste rock are shown in Figure 20-1.

Acid Base Accounting (ABA) testing was performed on the 19 waste rock samples using criteria identified in the Mining Environment Neutral Drainage (MEND) Program promulgated, by Natural Resources Canada (2009). These samples included eight limestone, six granite, and five sandstone country rock origins. All samples were characterized with low sulphide concentrations (between <0.01 and 0.35 weight percentage as sulphide). As a result acid generation is identified as an insignificant issue, as the Neutralization Potential Ratio (NPR) of all samples was greater than two. It should be noted that one sandstone sample demonstrated a Carbonate Neutralization Potential (CaNPR) of between one and two, indicating an uncertain acid generating potential from this individual location. This sample had less than 0.01 weight % as sulfur, and hence is unlikely to generate acidity.

The results provided in the Geochemical Results of Waste Rock and Tailings Samples Report are preliminary, and additional sampling is required to assess short-term and potential long-term metal leaching characteristics of the waste rock. Further sampling may be required to ensure conformance with best practice sampling guidance. The Geochemical Results of Waste Rock and Tailings Samples Report recommends that elemental analysis and a review of total waste rock tonnages and rock types be performed to verify that the current number of samples is consistent with accepted characterization guidelines. These efforts will be undertaken during the feasibility study stage of Project development.

A total of 283 kg of tailings from pilot testing of the sulfide ore was provided to Golder in May, 2013. Provided rougher and scavenger samples were combined in a 20:1 rougher/scavenger ratio to represent a homogenous tailings blend. The liquid fraction supernatant from the tailings was separated from tailings solids and subject to analytical testing.

Testing for tailings solids included elemental analysis, ABA and net acid generation (NAG) testing, and short-term leaching potential. Major oxides present in the sample included silica, iron and calcium oxides. Sulfide concentration was 5% as weight percentage. There were a number of metals present in concentrations over ten times crustal abundance including silver, arsenic, bismuth, manganese, molybdenum, lead, antimony, tin and zinc

Neutralization potential ratio (NPR) of this sample was between one and two, indicating uncertain acid generation potential. However the sample had a CaNPR value of less than one, which would classify it as potentially acid generating (PAG). However NAG potential pH was over 4.5, suggesting tailings are not likely to generate acidity. In summary the acid generation potential of this individual sample is variable depending on the method of assessment, and the testing program completed to date suggests that tailings should be assumed to generate acidity.

Short-term leach tests for tailings material indicated barium, manganese and zinc may leach at concentrations that are greater than the applicable Mexican Standards for Receiving Body of Water, as promulgated by CONAGUA/SEMARNAT (2009). In addition NAG leach testing results indicate barium and manganese may leach at concentrations that exceed Mexican regulatory standards. Tailings process decant water (supernatant) quality has been modelled with indications that total ammonia, beryllium, manganese, selenium, and zinc concentrations would exceed applicable Mexican regulatory criteria.

With respect to the above regarding tailings and tailings supernatant quality – the Project plans to construct the DF with a HDPE liner to prevent infiltration into groundwater. Discussion of the overall Project water balance is provided in Section 20.2.1. There is the potential for periodic releases of water from the TDF to the environment. In this event a water treatment facility may be required, and this potential will be evaluated during the feasibility stage of Project development. Additional static and possible kinetic testing will also be performed to allow for a more refined understanding of tailings geochemistry.

20.2 Waste and Tailings Disposal, Site Monitoring and Water Management

Initial geochemical testing of waste rock indicates low potential for acid rock drainage or metal leaching. Additional study will be completed at the feasibility study stage, with resultant information to be provided in the pending MIA. A Prefeasibility Study Tailings Disposal Facility report was prepared by Golder in January, 2014. This study included a tailings deposition strategy.

The Project had previously considered returning a portion of generated tailings for use as underground backfill, as described in the Prefeasibility Study Tailings Disposal Facility Report. However due to the high costs of the paste backfill plant the Project has decided to cancel this option and use a cemented rock fill solution for backfill, decreasing upfront capital expenditure. Major impacts to the TDF design are addressed in an updated Prefeasibility Study Tailings Disposal Facility report, issued on February 24, 2014.

The TDF will be constructed with a HDPE liner to prevent seepage to underlying soils and groundwater. Tailings discharged to the TDF will be thickened to approximately 65% solids prior to disposal. As a result significant amounts of water will be recycled through the process circuit. The TDF will have a single tailings cell with enough capacity to contain the estimated 3.4 million m^3 of tailings material to be generated over the life of the Project. The configuration of the TDF is shown in Figure 2 of the report “Prefeasibility Study Tailings Disposal Facility”.

The perimeter dams for TDF cell will be constructed with rockfill. A report entitled Geotechnical Investigation at the Proposed Tailings Disposal Facility Area was prepared by Golder in January 2013. This report assesses the subsurface geology at the proposed TDF location, as well as describing potential borrow locations for TDF construction materials. A total of 18 test pits were excavated and sampled at the TDF location, ranging in depths from 0.20 to 1.84 meter. These tests were carried out until excavator refusal. Test results show a thin veneer of overburden soils (primarily sand/gravel and saprolite), often underlain by calcrete (caliche). Calcrete is a hard material formed by calcite precipitation in soils, and is common in arid and subarid regions.

The Geotechnical Investigation at the Proposed Tailings Disposal Facility Area identified the need for borehole drilling at the TDF location to identify the potential presence of karst (void) features in limestone underlying the site. A total of five boreholes drilled to a depth of 10 m are proposed to assess the potential presence of any voids underlying the TDF, and this work will be executed during the feasibility stage of Project development. Samples of granular materials obtained from two private existing borrow sites indicates that these resources could provide the material required for construction of all Project facilities, including the TDF, although the materials may not be suitable for concrete production. As a result other potential borrow sites will be evaluated during feasibility stage development to confirm borrow locations for concrete sand and aggregate.

The TDF contains a settling pond which will be allowed to form at the toe of tailings beach, and upstream of the separation (South) dam. This will allow settling of finer material prior to discharge to a reclaim pond which will be used to recycle water back to the processing circuit. The rockfill berms (dams) will be constructed in conformance with the Canadian Dam Association’s Dam Safety Guidelines (2007), which will be used to guide design criteria for slope stability, necessary freeboard to accommodate flood events, and earthquake stability. The TDF has been designed to accommodate an Environmental Design Flood (EDF), which is a 1,000 year return, 24 hr event (73.2 mm).

Additional study is recommended in the Prefeasibility TDF Study Update. These studies will be completed during the feasibility stage of Project development and will include the following:

- Geotechnical drilling for a better understanding of geologic, geotechnical and hydrogeology conditions of the TDF area;
- Detailing of quantities of material available from potential borrow sources for TDF construction;
- Seismic hazard analysis to verify peak ground accelerations used in the TDF design;
- Confirmation of assumptions used to develop the water balance; and
- Additional geochemical testing of tailings material to determine the potential need for treatment of water which may be discharged during wet climatic conditions.

20.2.1 Water Balance

All runoff at the plant site (i.e., contact water) will be captured in drainage channels and routed to a mill runoff pond. The mill runoff pond will also receive as input treated domestic effluent, excess water from mine-dewatering activities, and direct precipitation. Water collected in the mill runoff pond will be pumped to the reclaim pond for use in the process circuit.

All runoff at the TDF perimeter, as well as collected seepage, will be routed to runoff collection ditches which lead to runoff collection sumps located to the north and south of the TDF. Runoff collection ditches are designed to accommodate peak flows associated with a 24-hour precipitation event with a 100-year recurrence interval. Water collected at the two runoff collection sumps will be diverted to the reclaim pond for use in the process circuit.

Therefore, flows available in the reclaim pond will consist of the following:

- any excess supernatant in the TDF;
- input from the mill runoff collection pond, including mine water;
- input from the runoff collection sumps; and
- direct input from precipitation.

Losses to the reclaim pond water balance will include evaporation, seepage, and any water used for dust control. The remainder will be available for use as make-up water in the processing circuit.

The supplemental Prefeasibility TDF Study Update provides the anticipated water balance based on return of all tailings to the TDF (i.e., no use of paste backfill). For steady state mining operations and under average climatic conditions the mill will require 238,134 m^3 of water on an annual basis. Resultant annual water balance calculations for the Project are summarized in Table 20-1 for average, 25-year wet, and 25-year dry precipitation conditions.

Table 20-1 Summary of Annual Water Balance for Various Precipitation Conditions

Water Balance	**25- year Wet Conditions** (662.7 mm precipitation; 1,227.9 mm evaporation)	**Average Conditions** (412.3 mm precipitation; 1,486.1 mm evaporation)	**25-year Dry Conditions** (197.4 mm precipitation; 1,756.4 mm evaporation)
Make-up Water Required at Mill (in m^3)	None	107,101	245,459
Accumulation of Excess Water (in m^3)	56,108	None	None

Based on this updated water balance Golder estimates an annual water deficit in the Project process circuit of 107,101 m^3 under average meteorological conditions. Excess water may accumulate under 25-year wet meteorological conditions, and 245,459 m^3 of make-up water would be required under 25-year dry meteorological conditions.

Other conclusions from the water balance analysis include the following:

- During average precipitation years (2-year return period) and for wet years with a return period of 5 years or less make-up water will be required to support the mill.
- During exceptionally dry years with a 100-year return period approximately 270,000 m^3 of make-up water will be required to support the mill.

- During a wet year with a return period of 10 years or more there will be an accumulation of water beyond that needed to support the mill. A contingency plan for water storage would then be required.

Any water deficit will need to be addressed via supply from external water sources. The deficit may be less if actual underground mine inflow rates are greater than the 50 m^3/day currently anticipated. The Tailing Disposal Facility and Water Management Pre-Feasibility Report recommends that water be stored prior to commissioning and operation of the Project. The TDF conceptual design includes construction of the reclaim pond during the start-up phase of the Project.

Exploratory drilling for a suitable water source will be pursued in target zones identified in the 2011 Phase 2 Hydrologic Study. This exploratory drilling will include hydrologic pump tests to verify suitability of the identified resource over time. As mentioned previously water rights will need to be purchased or transferred from existing users in the region, as there is a long-standing ban on further groundwater withdrawal from the limited aquifers of Municipality of General Panfilo Natera.

20.3 Permitting

Prior to construction all mining projects must first prepare an MIA and Environmental Risk Study (“*Estudio de Riesgo Ambiental”* and “ERA”). The MIA and ERA studies detail results of baseline studies, characterize potential environmental and social impacts, and identify appropriate mitigations. In addition management plans and monitoring programs are identified to ensure successful environmental and social performance. These completed studies are jointly submitted to SEMARNET, which then reviews the document and either rejects or accepts the MIA with corresponding conditions of approval in a Resolution Letter (the “Resolucion”).

In addition to the MIA Resolution Letter a project must also obtain a Change in Land Use (“*Cambio de Uso de Suelos”* or “CUS”) permit which is granted after submission and approval of a technical study justifying the change in land use of the project area from its current use to development of a mine. The CUS permit has an associated cost, based on the current land use of the area to be developed.

On March 27, 2013 Bilbao Resources, S.A. de C.V. contracted with the Mexican environmental consultancy SIICA to complete the MIA and ERA. At the time of writing these documents were under development, incorporating information from existing environmental studies as well as studies that are in progress. In addition to the MIA and ERA the environmental consultancy SICCA will assist in development of the required technical study to issue the CUS, and a Program for the Prevention of Accidents (PPA).

Table 20-2 provides a summary of the major permits that will be required to implement the Project.

20.4 Social and Community Impact

Details of potential social and community impact will be addressed in the pending MIA. As discussed previously the State of Zacatecas has experienced centuries of mining development. Anticipated impacts to the Project area of influence are expected to be positive including employment opportunities.

20.5 Closure Planning

Per regulatory requirements the MIA will include information for closure and reclamation plans once mining is completed. Typical design features include the channeling of surface waters into natural drainages, and scarifying and reseeding of waste rock features. Down gradient monitoring of water quality will be performed to ensure no remnant groundwater contamination is present. Conceptual closure information for the TDF is provided in the Tailing Disposal Facility and Water Management Pre-Feasibility Report. A 0.5 m thick compacted sand and gravel cover will be emplaced over the entire tailing surface and runoff sumps will be decommissioned. A small wetland will be allowed to form upstream of the reclaim pond dam to allow for sedimentation and evaporation of accumulated surface runoff.

Table 20-2 Summary of Required Mexican Permits/Approvals/Authorizations

Permit	Involved Agency/Entity	Description / Comments
Environmental Impact Assessment (MIA)	SEMARNAT SEMARNAT State Office	Requires an evaluation of baseline conditions and predicted effects with regards to air, water, soils, wildlife, plants, cultural resources and socioeconomic factors. Requires a discussion and evaluation of mitigation measures such as avoidance strategies, control equipment, monitoring plans, and reclamation plans.
MIA Resolution Memo	SEMARNAT PROFEPA (Secretary for Federal Protection of the Environment) State Office	Required prior to construction. Will contain a series of conditions for construction and implementation of the Project, based on review of the prepared MIA.
Unique Environmental License	SEMARNAT State Office	Required for new operations, planned expansions of existing operations or operations that require regulation. This is a Federal requirement
Land Use Change (CUS)	SEMARNAT Forestry Resources SEMARNAT State Office	This permit from SEMARNAT is required to change the use of land where such a change might have a serious adverse impact on soil or ecology. For example, stripping of vegetation in preparation for mine construction would require such a permit. The permit application process requires submittal of a justification for the change that takes into account not only the predicted effects on soil and ecology, but also the economic benefits that would arise should the change be permitted
Archaeological release letter (Mining)	INAH (State offices)	The *Instituto Nacional de Antropología e Historia* (National Institute of Anthropology and History) reviews project plans and inspects the project area for historic and archaeological resources. Following inspection they will issue a clearance letter or advise on requirements for protection or recovery of identified resources.
Use of Explosives	SEDENA (Secretary of National Defense)	Required prior to use for the purchase, transport, store or use explosives. Permission required, in writing, from the Governor of the State of Zacatecas. Security clearance required. Must inform of location of powder magazines and the closest human activity. Requires monthly reports on usage and inventory
License of Construction	Municipality of General Panfilo Natera	Building permits will be required from the Muncipality prior to construction.
License of Land Use	Municipality of General Panfilo Natera	Approval of Municipality for land use. Equivalent to zoning approval.
Hazardous Wastes	SEMARNAT State Office	Generators of hazardous waste must be licensed. Generators are responsible for ultimate safe disposition of wastes.

No final closure bond is required by Mexican Law, although a bond is required to maintain production facilities during times when mine production may be halted. SEMARNAT approval of the MIA typically includes a condition requiring formal submittal of a reclamation plan prior to the mine's closure (i.e., the development of a detailed Mine Reclamation and Closure Plan is not required before at the initiation of construction and mining). The Technical Memorandum - Prefeasibility TDF Study Update recommends that a detailed TDF closure plan be developed during the feasibility stage of design to prevent the possibility of impacted water from being discharged to the environment at mine closure.

21. Capital & Operating Costs

21.1 Project Capital Cost

Project Capital Costs, as of April 2014, are estimated to be USD 99.5M including an allowance for contingencies of USD 8.7M, equivalent to 8.8% of total capital expenditure. The capital cost summary as presented in Table 21-1 outlines total pre-production capital of USD 91.2M and remaining other capital and sustaining capital costs of USD 8.3M for the 8 year production life, including acquisition to replace mine equipment fleet, plant and infrastructure.

Table 21-1 Capital Cost Summary - USD

Capital Expenditures	Pre-Production Year -1	LOM Production Year 1-8	Total
Exploration	600,000	-	600,000
Mine Facilities & Equipment	11,529,000	-	11,529,000
Mining Equipment - Leased		-	-
U/G Mine Development	3,509,000	3,229,000	6,738,000
Backfill Plant & Distribution System	500,000	600,000	1,100,000
Infrastructure	6,942,462	-	6,942,462
Surface Mobile Equipment	700,000	-	700,000
Processsing Plant	38,321,221	-	38,321,221
Tailings Disposal Facility	6,615,067	4,694,080	11,309,147
EPCM & Contractor O/H	10,318,448	-	10,318,448
Owners Costs	3,980,000	-	3,980,000
Reclamation and Closure		1,181,000	1,181,000
Working Capital	2,017,503	- 2,017,503	- 0
Additional Contingency	6,138,247	660,858	6,799,105
Total Capital Expenditures	**91,170,948**	**8,347,435**	**99,518,383**

21.2 Operating Cost Summary

The operating expenditure is based on all development work in waste being performed by contractors, and stope development by Xtierra personnel and equipment fleets. The strategy was determined as the most cost effective for the operation and ensures sustainability of a skilled labor force.

The average total unit cost for the operational activities is USD 66.90/t of ore. The breakdown of mining, processing, general and administration, freight and insurance, and smelting, refining and penalties is presented in Table 21-2.

The lifetime annual average of all operating costs included from Years 1 to 8 amounts to USD 43.4M.

Mining and Process Plant operating costs are largely variable per tonne of product while the General and Administrative costs are fixed per year. RPM has reviewed the basis of the operating cost estimate and considers the costs to be appropriate for evaluating economic viability of the project.

Table 21-2 Average Unit Operating Cost

Operating Cost	USD/Tonne ROM
Mine	25.73
Process	13.21
Site G&A	5.00
Freight and Insurance	2.08
Smelting, Refining, Penalties	20.88
Total Unit Operating Cost	**66.90**

22. Economic Analysis

This preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The economic analysis was completed for a 720,000 tonne per year processing plant capacity and is based on the mineable resources outlined in Table 22-1.

The financial evaluation incorporates the methodology of capitalization of waste during pre-production and production Year 1 including a portion of the main ramp, level development and raise development, as per industry standards.

The following section is a detailed discussion on the cash flow and economic parameters for the project.

22.1 Project Evaluation

22.1.1 Estimated Product Price and Revenue

At the request of Xtierra, RPM has based its economic analysis of the Bilbao project on three-year average metal prices. For the three-year period ending 31 October, 2013, the rolling average prices based on LME cash buyer quotes for zinc and lead, and as reported by Kitco on www.kitco.com for silver are as follows:

Zinc	US$0.92/lb
Lead	US$1.00/lb
Silver	US$30.38/oz

Sensitivities to these prices are evaluated in this Section.

Total revenue for the project is based on 720 kt/y production to be reached in production period 2 and continuing for the life of the project average USD 73.5 million per year (gross revenue). The current plan estimates 11k tonnes of zinc concentrate and 7k tonnes of lead concentrate in the first production year.

22.1.2 Pre-Tax Cash Flow Analysis

The Project's annual net cash flows were modeled based on the projected revenue, operating costs and capital expenditures summarized in Section 21 of this report.

A Royalty of 1.5% is payable to Minera Portree, S.A de C.V. on NSR.

A pre-tax cash flow was determined excluding corporate tax, profit sharing and mining duty payable to the Mexican government. The pre-tax cash flow can be seen in Table 22-1.

Pre-tax earnings total USD 59.9 million over the 8 year designated mine life. Economic results of the Project cash flow model indicate in Internal Rate of Return (IRR) of 13.2% and a Net Present Value (NPV) of USD 11.0M at a 10% discount rate. The ten percent discount rate is considered appropriate for this evaluation as the overall project risks are considered to be relatively low in terms of total capital committed, geological risk and market risk.

Table 22-1 Pre-Tax Project Cash Flow

	Production Period Financial Year Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Production											
Ore Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Total Tonnes Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Processed Grades											
Zinc	%		2.37%	2.27%	2.91%	2.34%	2.99%	1.94%	0.96%	0.97%	2.10%
Lead	%		1.70%	1.63%	1.88%	1.55%	1.66%	1.07%	0.85%	0.95%	1.40%
Silver	g/t		60.08	62.21	68.28	63.90	61.34	68.86	72.65	48.84	63.96
Contained Metal											
Zinc	lb		15,602,022	36,052,764	46,167,102	37,142,844	47,462,359	30,829,074	15,285,653	12,063,275	240,605,091
Lead	lb		11,196,280	25,829,904	29,788,348	24,663,037	26,280,765	17,021,685	13,558,841	11,924,471	160,263,330
Silver	oz		577,491	1,440,065	1,580,649	1,479,163	1,419,872	1,594,069	1,681,793	889,747	10,662,850
Mill Recovery											
Zinc	76.7%		76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%
Lead	90.6%		90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%
Silver	73.4%		73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%
Recovered Metal											
Zinc	lb		11,966,751	27,652,470	35,410,167	28,488,561	36,403,629	23,645,900	11,724,096	9,252,532	184,544,105
Lead	lb		10,143,829	23,401,893	26,988,243	22,344,712	23,810,373	15,421,647	12,284,310	10,803,570	145,198,577
Silver	oz		423,878	1,057,008	1,160,197	1,085,706	1,042,186	1,170,047	1,234,436	653,075	7,826,532
Concentrate Production											
Zinc Concentration Ratio	26.70		26.70	26.70	26.70	26.70	26.70	26.70	26.70	26.70	
Zinc Concentrate Produced (tonnes)			11,198	26,966	26,966	26,966	26,966	26,966	26,966	21,224	194,220
Lead Concentration Ratio	42.57		42.57	42.57	42.57	42.57	42.57	42.57	42.57	42.57	
Lead Concentrate Produced (tonnes)			7,024	16,913	16,913	16,913	16,913	16,913	16,913	13,312	121,815
Payability of Metal - HG Zn											
Zinc	85%		85%	85%	85%	85%	85%	85%	85%	85%	
Lead	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Silver	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Payable Metal											
Zinc	lb		10,171,738	23,504,599	30,098,642	24,215,277	30,943,085	20,099,015	9,965,481	7,864,652	156,862,489
Lead	lb		9,636,638	22,231,799	25,638,831	21,227,476	22,619,855	14,650,564	11,670,094	10,263,392	137,938,648
Silver	oz		402,684	1,004,158	1,102,187	1,031,420	990,077	1,111,544	1,172,714	620,421	7,435,205
METAL PRICES											
Zinc	$0.9229		$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229
Lead	$1.0047		$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047
Silver	$30.3761		$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761
Revenue From Metal Sales											
Zinc	US$		$9,387,497	$21,692,395	$27,778,037	$22,348,279	$28,557,373	$18,549,381	$9,197,143	$7,258,287	$144,768,391
Lead	US$		$9,681,930	$22,336,288	$25,759,333	$21,327,245	$22,726,168	$14,719,422	$11,724,944	$10,311,630	138,586,960
Silver	US$		$12,231,978	$30,502,392	$33,480,138	$31,330,526	$30,074,664	$33,764,381	$35,622,492	$18,845,964	225,852,534
Total Sales Revenue	**US$**		**$31,301,405**	**$74,531,075**	**$87,017,508**	**$75,006,050**	**$81,358,205**	**$67,033,184**	**$56,544,578**	**$36,415,881**	**$509,207,885**

Table 22-1 Pre-Tax Project Cash Flow (cont.)

	Production Period	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
	Financial Year	0	1	2	3	4	5	6	7	8	
	Unit										
Operating Costs											
Exploration - Definition Drilling	US$0.58/t		$173,415	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$328,675	$3,007,691
Mobile Mine Equipment Leasing	US$		$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$2,741,894	$24,665,894
U/G Mining - Development	US$			$653,000	$1,722,000	$1,474,000	$2,352,000	$1,034,000	$0	$0	$7,235,000
U/G Mining - Ore	US$		$7,722,000	$15,534,000	$18,381,000	$11,728,000	$15,717,000	$11,182,000	$12,944,000	$5,287,000	$98,495,000
Processing	$13.21		$3,949,684	$9,511,207	$9,511,198	$9,511,205	$9,511,194	$9,511,203	$9,511,198	$7,485,867	$68,502,757
General and Administration	$5.00		$1,494,960	$3,600,002	$3,599,999	$3,600,002	$3,599,998	$3,600,001	$3,599,999	$2,833,409	$25,928,371
Concentrate Transportation - Zinc	$35.00		$391,937	$943,821	$943,820	$943,821	$943,820	$943,821	$943,820	$742,841	$6,797,700
Concentrate Transportation - Lead	$27.00		$189,636	$456,660	$456,660	$456,660	$456,659	$456,660	$456,660	$359,418	$3,289,011
Insurance - Zinc	$2.72		$30,459	$73,348	$73,348	$73,348	$73,348	$73,348	$73,348	$57,729	$528,278
Insurance - Lead	$1.51		$10,606	$25,539	$25,539	$25,539	$25,539	$25,539	$25,539	$20,101	$183,941
Smelting - Zinc ($/tonne conc.)	$205.00		$2,295,631	$5,528,094	$5,528,089	$5,528,093	$5,528,087	$5,528,092	$5,528,089	$4,350,928	$39,815,102
Smelting - Zinc penalty ($/tonne Conc.)	$8.85		$99,104	$238,652	$238,652	$238,652	$238,652	$238,652	$238,652	$187,833	$1,718,847
Refining - Zinc ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Smelting - Lead ($/ tonne conc.)	$300.00		$2,107,061	$5,073,999	$5,073,995	$5,073,998	$5,073,993	$5,073,997	$5,073,995	$3,993,530	$36,544,568
Refining - Lead ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Refining - Silver ($/ounce)	$4.06		$1,634,898	$4,076,880	$4,474,879	$4,187,566	$4,019,711	$4,512,870	$4,761,221	$2,518,908	$30,186,933
Total Operating Costs	**US$**	**$0**	**$23,231,392**	**$49,264,802**	**$53,578,778**	**$46,390,484**	**$51,089,600**	**$45,729,782**	**$46,706,120**	**$30,908,134**	**$346,899,093**
Unit Operating Costs											
Mine	US$/tonne ore		$36.88	$27.41	$32.85	$23.27	$30.03	$21.90	$22.91	$14.75	$25.73
Process	US$/tonne ore		$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21
Site G&A	US$/tonne ore		$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00
Freight and Insurance	US$/tonne ore		$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08
Smelting, Refining, Penalties	US$/tonne ore		$20.52	$20.72	$21.27	$20.87	$20.64	$21.32	$21.67	$19.50	$20.88
Total Unit Operating Cost	**US$/tonne ore**		**$77.70**	**$68.42**	**$74.41**	**$64.43**	**$70.96**	**$63.51**	**$64.87**	**$54.54**	**$66.90**
Net Smelter Return											
NSR Zinc	US$		$6,600,824	$14,981,828	$21,067,476	$15,637,714	$21,846,815	$11,838,817	$2,486,582	$1,976,685	$96,436,742
NSR Lead	US$		$7,385,233	$16,805,629	$20,228,679	$15,796,587	$17,195,516	$9,188,765	$6,194,289	$5,958,682	$98,753,381
NSR Silver	US$		$10,597,079	$26,425,512	$29,005,259	$27,142,960	$26,054,953	$29,251,512	$30,861,271	$16,327,056	$195,665,602
Total Net Smelter Return	**US$**	**$0**	**$24,583,137**	**$58,212,969**	**$70,301,414**	**$58,577,261**	**$65,097,284**	**$50,279,093**	**$39,542,142**	**$24,262,423**	**$390,855,724**
Royalties											
Royalty (1.5% NSR) - Minera Portree	US$			$873,195	$1,054,521	$878,659	$976,459	$754,186	$593,132	$363,936	$5,494,089
Total Operating Cost Including Royalties	**US$**		**$23,231,392**	**$50,137,997**	**$54,633,299**	**$47,269,143**	**$52,066,059**	**$46,483,969**	**$47,299,253**	**$31,272,071**	**$352,393,182**
Operating Income		**$0**	**$8,070,013**	**$24,393,078**	**$32,384,208**	**$27,736,907**	**$29,292,146**	**$20,549,215**	**$9,245,325**	**$5,143,811**	**$156,814,703**

Table 22-1 Pre-Tax Project Cash Flow (cont.)

	Production Period / Financial Year / Unit	Year -1 / 0	Year 1 / 1	Year 2 / 2	Year 3 / 3	Year 4 / 4	Year 5 / 5	Year 6 / 6	Year 7 / 7	Year 8 / 8	Total
Capital Expenditures											
Initial Capital	US$										
Exploration	US$	$600,000									$600,000
Mine Facilities & Equipment	US$	$11,529,000									$11,529,000
Mining Equipment - Leased	US$										$0
U/G Mine Development	US$	$3,509,000	$3,229,000								$6,738,000
Backfill Plant & Distribution System	US$	$500,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$1,100,000
Infrastructure	US$	$6,942,462									$6,942,462
Surface Mobile Equipment	US$	$700,000									$700,000
Processsing Plant	US$	$38,321,221									$38,321,221
Tailings Disposal Facility	US$	$6,615,067		$1,937,767			$2,756,313				$11,309,147
EPCM & Contractor O/H	US$	$10,318,448									$10,318,448
Owners Costs	US$	$3,980,000									$3,980,000
Reclamation and Closure	US$									$1,181,000	$1,181,000
Working Capital	US$	$2,017,503								-$2,017,503	$0
Additional Contingency	US$	$6,138,247	$165,200	$197,527	$3,750	$3,750	$279,381	$3,750	$3,750	$3,750	$6,799,105
Total Capital Expenditures	**US$**	**$91,170,948**	**$3,469,200**	**$2,210,293**	**$78,750**	**$78,750**	**$3,110,695**	**$78,750**	**$78,750**	**-$757,753**	**$99,518,383**
Cost of Capital per tonne ore mined	*US$*										*$19.19*
Depreciation											
Depreciation	US$		**$5,737,962**	**$13,817,545**	**$13,817,532**	**$13,817,542**	**$13,817,527**	**$13,817,539**	**$13,817,532**	**$10,875,203**	**$99,518,383**
Salvage											
Mobile Equipment	US$									$0	$0
Fixed Equipment	US$									$430,554	$430,554
Building / Infrastructure	US$									$2,137,439	$2,137,439
Total Salvage	**US$**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$2,567,993**	**$2,567,993**
EBITA											
EBITA (Annual)	US$	$0	$2,332,051	$10,575,533	$18,566,676	$13,919,365	$15,474,619	$6,731,676	-$4,572,207	-$3,163,400	$59,864,314
EBITA (Cumulative)	US$	$0	$2,332,051	$12,907,584	$31,474,260	$45,393,625	$60,868,244	$67,599,920	$63,027,713	$59,864,314	
Before Tax Cash Flow	US$	-$91,170,948	$4,600,813	$22,182,784	$32,305,458	$27,658,157	$26,181,451	$20,470,465	$9,166,575	$8,469,557	$59,864,314
Discount Rate	10%										
Discount Factor		1.00	0.91	0.83	0.75	0.68	0.62	0.56	0.51	0.47	
Discounted Cash Flow	US$	-$91,170,948	$4,182,557	$18,332,880	$24,271,569	$18,890,894	$16,256,621	$11,555,044	$4,703,903	$3,951,111	$10,973,630
Cumulative Discounted Cash Flow	US$	-$91,170,948	-$86,988,391	-$68,655,511	-$44,383,942	-$25,493,049	-$9,236,427	$2,318,617	$7,022,519	$10,973,630	
Net Present Value	10,973,630										
Internal Rate of Return	13.24%										

RPM developed a sensitivity analysis for the pre-tax cash flow model based on variations in key project elements of metal price, operating and capital costs. The sensitivity of the Project's IRR and NPV to +/- 15 percent changes to key assumptions is shown in Table 22-2.

Table 22-2 Pre-Tax Sensitivity Analysis

Item	NPV (USD Million)	IRR (%)
Base Case	**11.0**	**13.2%**
Capital Cost +15%	-3.7	9.01%
Capital Cost -15%	25.7	18.53%
Operating Cost +15%	-23.0	2.08%
Operating Cost -15%	45.0	21.99%
Sale Price (Zinc) +15%	25.4	17.22%
Sale Price (Zinc) -15%	-3.5	8.93%
Sale Price (Lead) +15%	24.8	17.04%
Sale Price (Lead) -15%	-2.8	9.14%
Sale Price (Silver) +15%	32.6	18.94%
Sale Price (Silver) -15%	-10.6	6.55%
Mill Recovery (Zinc) +15%	25.4	17.22%
Mill Recovery (Zinc) -15%	-3.5	8.93%
Mill Recovery (Lead) +15%	24.8	17.04%
Mill Recovery (Lead) -15%	-2.8	9.14%
Mill Recovery (Silver) +15%	29.7	18.22%
Mill Recovery (Silver) -15%	-7.7	7.52%

Spider charts are shown in Figure 22-1Figure 1-6 and Figure 22-2 below for the Project's pre-tax sensitivity to metal prices, capital cost, operating cost, and mill recovery, with key assumptions varying plus and minus 15 percent.

Figure 22-1 Pre-Tax Project NPV

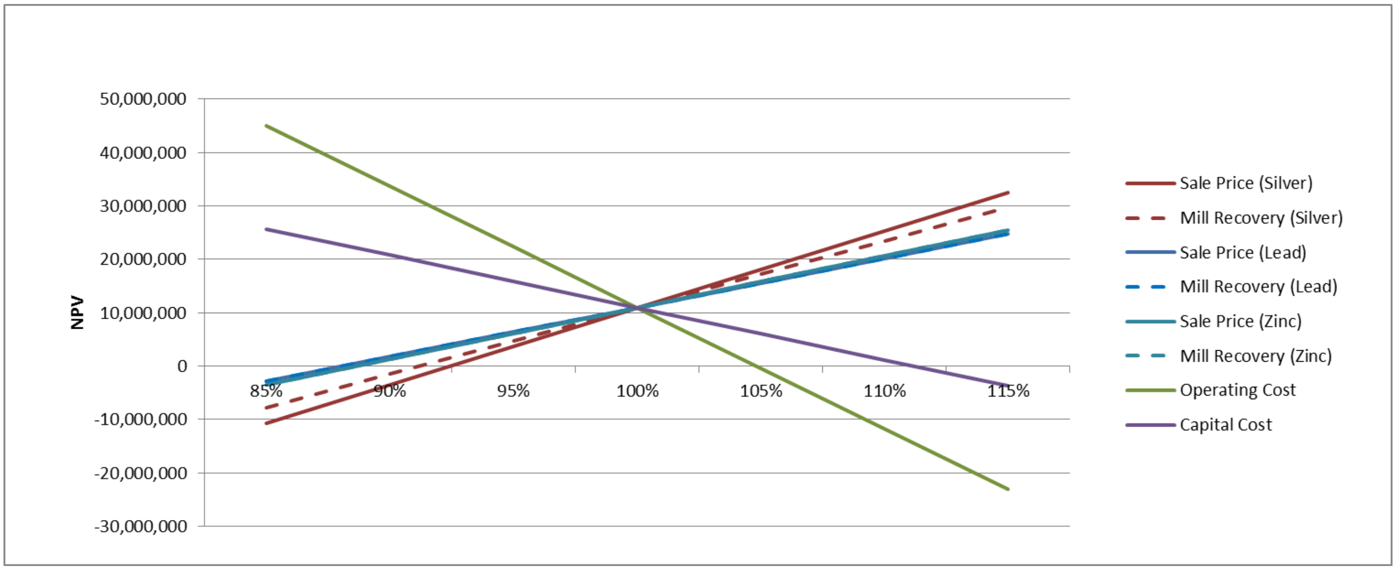
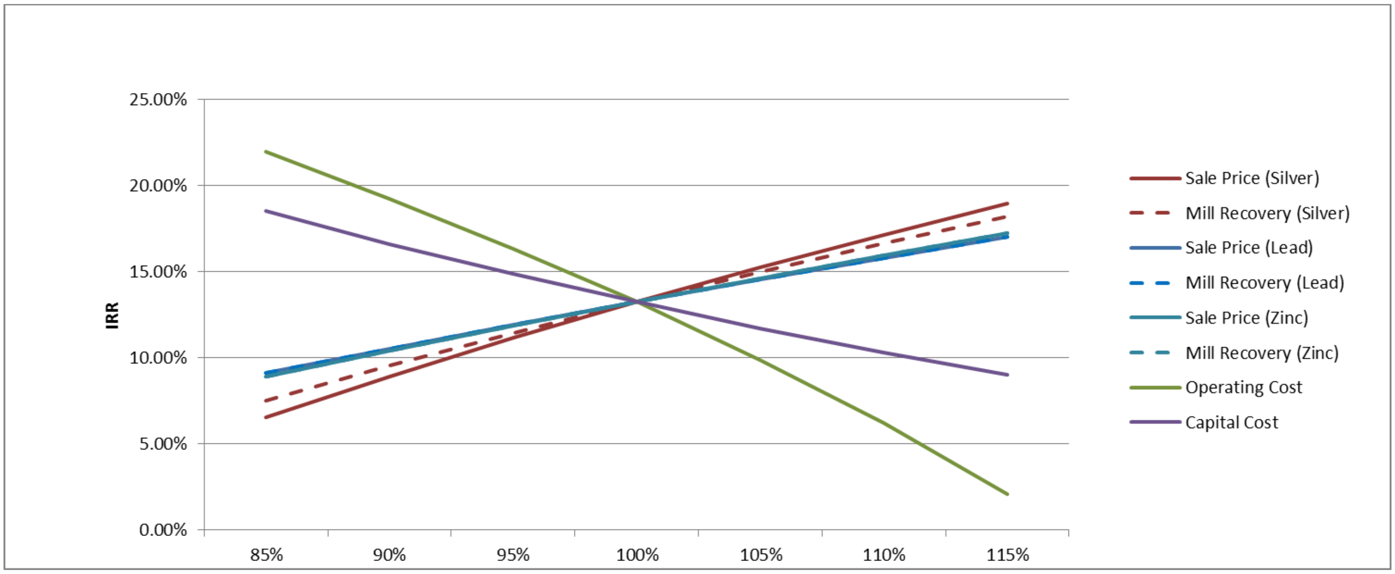


Figure 22-2 Pre-Tax Project IRR



22.1.3 After-Tax Cash Flow Analysis

The Project's annual net cash flows were modeled based on the projected revenue, operating costs and capital expenditures summarized in Section 21 of this report. Table 22-3 details the project cash flow.

A Royalty of 1.5% is payable to Minera Portree, S.A de C.V. on NSR. Applicable taxes considered in the cash flow analysis include the corresponding taxes for México:

- Profit sharing (Repartición de Utilidades) at 10 percent
- Long term corporate tax rate at 30 percent
- Mining duty based on EBITA at 7.5% (8.0% for silver)

After-tax net cash flow totals USD 32.6 million over the 8 year designated mine life. Economic results of the Project cash flow model indicate in Internal Rate of Return (IRR) of 8.1% and a Net Present Value (NPV) of USD -5.8M at a 10% discount rate. The ten percent discount rate is considered appropriate for this evaluation as the overall project risks are considered to be relatively low in terms of total capital committed, geological risk and market risk.

The after-tax cash flow can be seen in Table 22-3.

RPM developed a sensitivity analysis for the after-tax cash flow model based on variations in key project elements of metal price, operating and capital costs. The sensitivity of the Project's IRR and NPV to +/- 15 percent changes to key assumptions is shown in Table 22-4.

Table 22-3 After-Tax Project Cash Flow

	Production Period Financial Year Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Production											
Ore Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Total Tonnes Mined	tonne		298,992	720,000	720,000	720,000	720,000	720,000	720,000	566,682	5,185,674
Processed Grades											
Zinc	%		2.37%	2.27%	2.91%	2.34%	2.99%	1.94%	0.96%	0.97%	2.10%
Lead	%		1.70%	1.63%	1.88%	1.55%	1.66%	1.07%	0.85%	0.95%	1.40%
Silver	g/t		60.08	62.21	68.28	63.90	61.34	68.86	72.65	48.84	63.96
Contained Metal											
Zinc	lb		15,602,022	36,052,764	46,167,102	37,142,844	47,462,359	30,829,074	15,285,653	12,063,275	240,605,091
Lead	lb		11,196,280	25,829,904	29,788,348	24,663,037	26,280,765	17,021,685	13,558,841	11,924,471	160,263,330
Silver	oz		577,491	1,440,065	1,580,649	1,479,163	1,419,872	1,594,069	1,681,793	889,747	10,662,850
Mill Recovery											
Zinc	76.7%		76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%	76.7%
Lead	90.6%		90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%	90.6%
Silver	73.4%		73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%	73.4%
Recovered Metal											
Zinc	lb		11,966,751	27,652,470	35,410,167	28,488,561	36,403,629	23,645,900	11,724,096	9,252,532	184,544,105
Lead	lb		10,143,829	23,401,893	26,988,243	22,344,712	23,810,373	15,421,647	12,284,310	10,803,570	145,198,577
Silver	oz		423,878	1,057,008	1,160,197	1,085,706	1,042,186	1,170,047	1,234,436	653,075	7,826,532
Concentrate Production											
Zinc Concentration Ratio	26.70		26.70	26.70	26.70	26.70	26.70	26.70	26.70	26.70	
Zinc Concentrate Produced (tonnes)			11,198	26,966	26,966	26,966	26,966	26,966	26,966	21,224	194,220
Lead Concentration Ratio	42.57		42.57	42.57	42.57	42.57	42.57	42.57	42.57	42.57	
Lead Concentrate Produced (tonnes)			7,024	16,913	16,913	16,913	16,913	16,913	16,913	13,312	121,815
Payability of Metal - HG Zn											
Zinc	85%		85%	85%	85%	85%	85%	85%	85%	85%	
Lead	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Silver	95%		95%	95%	95%	95%	95%	95%	95%	95%	
Payable Metal											
Zinc	lb		10,171,738	23,504,599	30,098,642	24,215,277	30,943,085	20,099,015	9,965,481	7,864,652	156,862,489
Lead	lb		9,636,638	22,231,799	25,638,831	21,227,476	22,619,855	14,650,564	11,670,094	10,263,392	137,938,648
Silver	oz		402,684	1,004,158	1,102,187	1,031,420	990,077	1,111,544	1,172,714	620,421	7,435,205
METAL PRICES											
Zinc	$0.9229		$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229	$0.9229
Lead	$1.0047		$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047	$1.0047
Silver	$30.3761		$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761	$30.3761
Revenue From Metal Sales											
Zinc	US$		$9,387,497	$21,692,395	$27,778,037	$22,348,279	$28,557,373	$18,549,381	$9,197,143	$7,258,287	$144,768,391
Lead	US$		$9,681,930	$22,336,288	$25,759,333	$21,327,245	$22,726,168	$14,719,422	$11,724,944	$10,311,630	138,586,960
Silver	US$		$12,231,978	$30,502,392	$33,480,138	$31,330,526	$30,074,664	$33,764,381	$35,622,492	$18,845,964	225,852,534
Total Sales Revenue	**US$**		**$31,301,405**	**$74,531,075**	**$87,017,508**	**$75,006,050**	**$81,358,205**	**$67,033,184**	**$56,544,578**	**$36,415,881**	**$509,207,885**

Table 22-3 After-Tax Project Cash Flow (cont.)

	Production Period / Financial Year / Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Operating Costs											
Exploration - Definition Drilling	US$0.58/t		$173,415	$417,600	$417,600	$417,600	$417,600	$417,600	$417,600	$328,675	$3,007,691
Mobile Mine Equipment Leasing	US$		$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$3,132,000	$2,741,894	$24,665,894
U/G Mining - Development	US$			$653,000	$1,722,000	$1,474,000	$2,352,000	$1,034,000	$0	$0	$7,235,000
U/G Mining - Ore	US$		$7,722,000	$15,534,000	$18,381,000	$11,728,000	$15,717,000	$11,182,000	$12,944,000	$5,287,000	$98,495,000
Processing	$13.21		$3,949,684	$9,511,207	$9,511,198	$9,511,205	$9,511,194	$9,511,203	$9,511,198	$7,485,867	$68,502,757
General and Administration	$5.00		$1,494,960	$3,600,002	$3,599,999	$3,600,002	$3,599,998	$3,600,001	$3,599,999	$2,833,409	$25,928,371
Concentrate Transportation - Zinc	$35.00		$391,937	$943,821	$943,820	$943,821	$943,820	$943,821	$943,820	$742,841	$6,797,700
Concentrate Transportation - Lead	$27.00		$189,636	$456,660	$456,660	$456,660	$456,659	$456,660	$456,660	$359,418	$3,289,011
Insurance - Zinc	$2.72		$30,459	$73,348	$73,348	$73,348	$73,348	$73,348	$73,348	$57,729	$528,278
Insurance - Lead	$1.51		$10,606	$25,539	$25,539	$25,539	$25,539	$25,539	$25,539	$20,101	$183,941
Smelting - Zinc ($/tonne conc.)	$205.00		$2,295,631	$5,528,094	$5,528,089	$5,528,093	$5,528,087	$5,528,092	$5,528,089	$4,350,928	$39,815,102
Smelting - Zinc penalty ($/tonne Conc.)	$8.85		$99,104	$238,652	$238,652	$238,652	$238,652	$238,652	$238,652	$187,833	$1,718,847
Refining - Zinc ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Smelting - Lead ($/ tonne conc.)	$300.00		$2,107,061	$5,073,999	$5,073,995	$5,073,998	$5,073,993	$5,073,997	$5,073,995	$3,993,530	$36,544,568
Refining - Lead ($/ pound)	$0.00		$0	$0	$0	$0	$0	$0	$0	$0	$0
Refining - Silver ($/ounce)	$4.06		$1,634,898	$4,076,880	$4,474,879	$4,187,566	$4,019,711	$4,512,870	$4,761,221	$2,518,908	$30,186,933
Total Operating Costs	**US$**	**$0**	**$23,231,392**	**$49,264,802**	**$53,578,778**	**$46,390,484**	**$51,089,600**	**$45,729,782**	**$46,706,120**	**$30,908,134**	**$346,899,093**
Unit Operating Costs											
Mine	US$/tonne ore		$36.88	$27.41	$32.85	$23.27	$30.03	$21.90	$22.91	$14.75	$25.73
Process	US$/tonne ore		$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21	$13.21
Site G&A	US$/tonne ore		$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00	$5.00
Freight and Insurance	US$/tonne ore		$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08	$2.08
Smelting, Refining, Penalties	US$/tonne ore		$20.52	$20.72	$21.27	$20.87	$20.64	$21.32	$21.67	$19.50	$20.88
Total Unit Operating Cost	**US$/tonne ore**		**$77.70**	**$68.42**	**$74.41**	**$64.43**	**$70.96**	**$63.51**	**$64.87**	**$54.54**	**$66.90**
Net Smelter Return											
NSR Zinc	US$		$6,600,824	$14,981,828	$21,067,476	$15,637,714	$21,846,815	$11,838,817	$2,486,582	$1,976,685	$96,436,742
NSR Lead	US$		$7,385,233	$16,805,629	$20,228,679	$15,796,587	$17,195,516	$9,188,765	$6,194,289	$5,958,682	$98,753,381
NSR Silver	US$		$10,597,079	$26,425,512	$29,005,259	$27,142,960	$26,054,953	$29,251,512	$30,861,271	$16,327,056	$195,665,602
Total Net Smelter Return	**US$**	**$0**	**$24,583,137**	**$58,212,969**	**$70,301,414**	**$58,577,261**	**$65,097,284**	**$50,279,093**	**$39,542,142**	**$24,262,423**	**$390,855,724**
Royalties											
Royalty (1.5% NSR) - Minera Portree	US$			$873,195	$1,054,521	$878,659	$976,459	$754,186	$593,132	$363,936	$5,494,089
Total Operating Cost Including Royalties	**US$**		**$23,231,392**	**$50,137,997**	**$54,633,299**	**$47,269,143**	**$52,066,059**	**$46,483,969**	**$47,299,253**	**$31,272,071**	**$352,393,182**
Operating Income		**$0**	**$8,070,013**	**$24,393,078**	**$32,384,208**	**$27,736,907**	**$29,292,146**	**$20,549,215**	**$9,245,325**	**$5,143,811**	**$156,814,703**

Table 22-3 After-Tax Project Cash Flow (cont.)

	Production Period	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total
	Financial Year	0	1	2	3	4	5	6	7	8	
	Unit										
Capital Expenditures											
Initial Capital	US$										
Exploration	US$	$600,000									$600,000
Mine Facilities & Equipment	US$	$11,529,000									$11,529,000
Mining Equipment - Leased	US$										$0
U/G Mine Development	US$	$3,509,000	$3,229,000								$6,738,000
Backfill Plant & Distribution System	US$	$500,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$75,000	$1,100,000
Infrastructure	US$	$6,942,462									$6,942,462
Surface Mobile Equipment	US$	$700,000									$700,000
Processsing Plant	US$	$38,321,221									$38,321,221
Tailings Disposal Facility	US$	$6,615,067		$1,937,767			$2,756,313				$11,309,147
EPCM & Contractor O/H	US$	$10,318,448									$10,318,448
Owners Costs	US$	$3,980,000									$3,980,000
Reclamation and Closure	US$									$1,181,000	$1,181,000
Working Capital	US$	$2,017,503								-$2,017,503	$0
Additional Contingency	US$	$6,138,247	$165,200	$197,527	$3,750	$3,750	$279,381	$3,750	$3,750	$3,750	$6,799,105
Total Capital Expenditures	**US$**	**$91,170,948**	**$3,469,200**	**$2,210,293**	**$78,750**	**$78,750**	**$3,110,695**	**$78,750**	**$78,750**	**-$757,753**	**$99,518,383**
Cost of Capital per tonne ore mined	*US$*										*$19.19*
Depreciation											
Depreciation	US$		**$5,737,962**	**$13,817,545**	**$13,817,532**	**$13,817,542**	**$13,817,527**	**$13,817,539**	**$13,817,532**	**$10,875,203**	**$99,518,383**
Salvage											
Mobile Equipment	US$									$0	$0
Fixed Equipment	US$									$430,554	$430,554
Building / Infrastructure	US$									$2,137,439	$2,137,439
Total Salvage	**US$**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$0**	**$2,567,993**	**$2,567,993**
EBITA											
EBITA (Annual)	US$	$0	$2,332,051	$10,575,533	$18,566,676	$13,919,365	$15,474,619	$6,731,676	-$4,572,207	-$3,163,400	$59,864,314
EBITA (Cumulative)	US$	$0	$2,332,051	$12,907,584	$31,474,260	$45,393,625	$60,868,244	$67,599,920	$63,027,713	$59,864,314	

Table 22-3 After-Tax Project Cash Flow (cont.)

	Production Period Financial Year Unit	Year -1 0	Year 1 1	Year 2 2	Year 3 3	Year 4 4	Year 5 5	Year 6 6	Year 7 7	Year 8 8	Total
Write-offs											
Total Write-Offs	**US$**									**$25,679,930**	**$25,679,930**
Total Taxes and Duties Payable											
ISR	US$	$0	$0	$0	$0	$2,660,484	$5,987,186	$4,514,939	$1,386,556	$0	$14,549,165
PTU	US$	$0	$0	$0	$0	$44,476	$44,476	$44,476	$44,476	$0	$177,903
Mining Duty EBITA (Lead and Zinc)	US$	$0	$344,345	$1,034,757	$1,473,183	$1,151,697	$1,361,528	$668,209	$161,991	$135,104	$6,330,814
Mining Duty EBITA (Silver)	US$	$0	$278,300	$917,561	$1,103,704	$1,060,768	$969,192	$991,516	$614,286	$296,509	$6,231,835
Total Taxes	**US$**	**$0**	**$622,645**	**$1,952,318**	**$2,576,886**	**$4,917,426**	**$8,362,382**	**$6,219,139**	**$2,207,309**	**$431,613**	**$27,289,717**
Total Taxes and Duties Paid											
ISR	US$		$0	$0	$0	$0	$44,476	$44,476	$44,476	$44,476	$177,903
PTU	US$		$0	$0	$0	$2,660,484	$5,987,186	$4,514,939	$1,386,556	$0	$14,549,165
Mining Duty EBITA (Lead and Zinc)	US$		$0	$344,345	$1,034,757	$1,473,183	$1,151,697	$1,361,528	$668,209	$297,095	$6,330,814
Mining Duty EBITA (Silver)	US$		$0	$278,300	$917,561	$1,103,704	$1,060,768	$969,192	$991,516	$910,795	$6,231,835
Total Taxes	**US$**	**$0**	**$0**	**$622,645**	**$1,952,318**	**$5,237,370**	**$8,244,127**	**$6,890,134**	**$3,090,757**	**$1,252,366**	**$27,289,717**
Net Earnings											
After Tax Earnings (Annual)	US$	$0	$1,709,406	$8,623,215	$15,989,790	$9,001,939	$7,112,237	$512,537	-$6,779,516	-$3,595,013	**$32,574,596**
After Tax Earnings (Cumulative)	US$	$0	$1,709,406	$10,332,621	$26,322,411	$35,324,351	$42,436,588	$42,949,125	$36,169,609	$32,574,596	
Net Cash Flow (After Tax)		-$91,170,948	$4,600,813	$21,560,140	$30,353,141	$22,420,787	$17,937,324	$13,580,331	$6,075,819	$7,217,191	$32,574,596
Discount Rate	10%										
Discount Factor		1.00	0.91	0.83	0.75	0.68	0.62	0.56	0.51	0.47	
Discounted Cash Flow	US$	-$91,170,948	$4,182,557	$17,818,297	$22,804,764	$15,313,699	$11,137,667	$7,665,743	$3,117,856	$3,366,873	**-$5,763,493**
Cumulative Discounted Cash Flow	US$	-$91,170,948	-$86,988,391	-$69,170,094	-$46,365,330	-$31,051,631	-$19,913,964	-$12,248,221	-$9,130,366	-$5,763,493	
Net Present Value	- 5,763,493										
Internal Rate of Return	8.11%										
Periods to Discounted Payback											

Table 22-4 After-Tax Sensitivity Analysis

Item	NPV (USD Million)	IRR (%)
Base Case	**-5.8**	**8.1%**
Capital Cost +15%	-18.5	4.57%
Capital Cost -15%	5.9	12.19%
Operating Cost +15%	-34.6	-3.35%
Operating Cost -15%	19.4	15.83%
Sale Price (Zinc) +15%	4.1	11.31%
Sale Price (Zinc) -15%	-16.0	4.59%
Sale Price (Lead) +15%	3.7	11.19%
Sale Price (Lead) -15%	-15.6	4.74%
Sale Price (Silver) +15%	9.1	12.80%
Sale Price (Silver) -15%	-21.7	2.32%
Mill Recovery (Zinc) +15%	4.1	11.31%
Mill Recovery (Zinc) -15%	-16.0	4.59%
Mill Recovery (Lead) +15%	3.7	11.19%
Mill Recovery (Lead) -15%	-15.6	4.74%
Mill Recovery (Silver) +15%	7.1	12.21%
Mill Recovery (Silver) -15%	-19.3	3.26%

The following table summarizes the sensitivity of the discount rate used on the before and after-tax NPV and IRR.

Table 22-5 Discount Rate Sensitivity Analysis

	Pre-Tax		After-Tax	
Discount Rate	NPV	IRR	NPV	IRR
0%	59,864,314	13.24%	32,574,596	8.11%
8%	18,724,880	13.24%	358,817	8.11%
9%	14,747,296	13.24%	- 2,780,353	8.11%
10%	10,973,630	13.24%	- 5,763,493	8.11%
11%	7,390,818	13.24%	- 8,600,429	8.11%
12%	3,986,785	13.24%	- 11,300,255	8.11%

Spider charts are shown in Figure 22-3 and Figure 22-4 below for the Project's after-tax sensitivity to metal prices, capital cost, operating cost, and mill recovery, with key assumptions varying plus and minus 15 percent.

Figure 22-3 After-Tax Project NPV

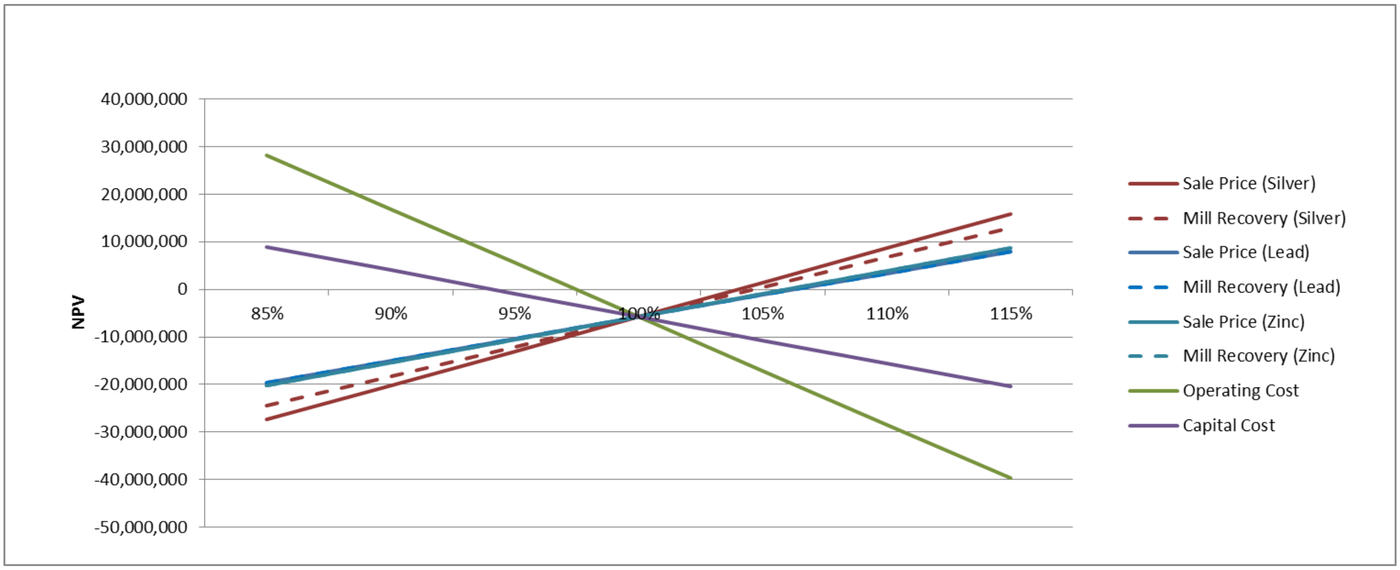
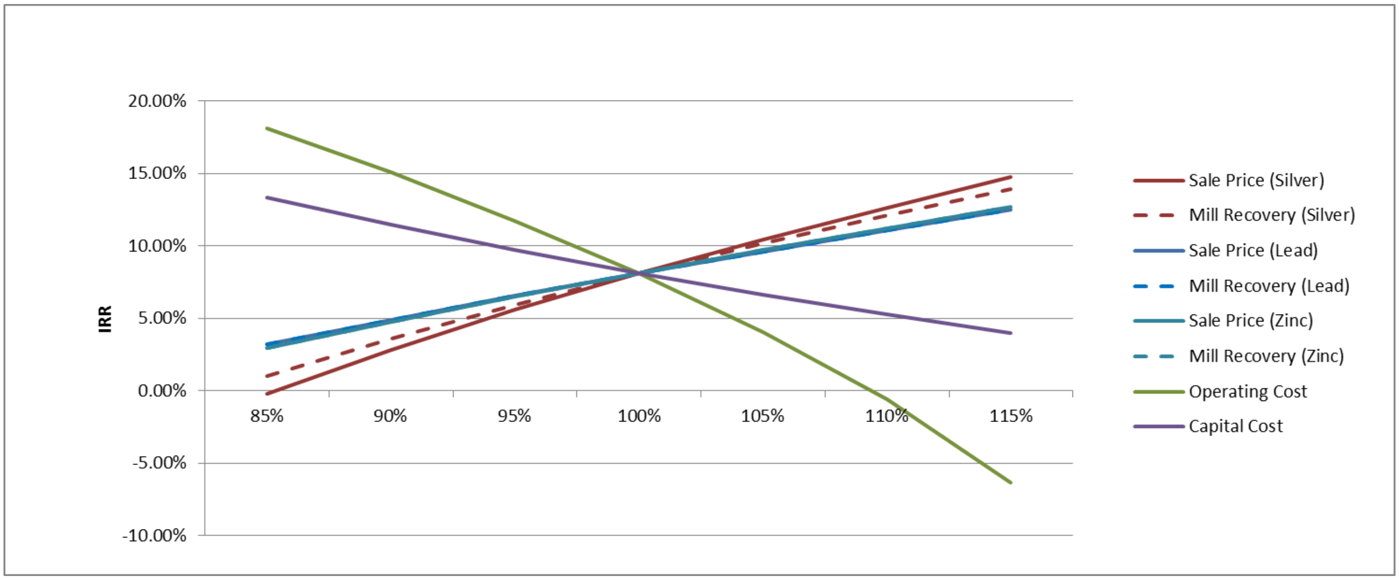


Figure 22-4 After-Tax Project IRR



23. Adjacent Properties

23.1 Information on Adjacent and Nearby Claims

The Company holds several claims abutting and enclosing the Bilbao mining claim including Bilbao-E-14,771, Bilbao II-E-15, La Güera, El Porvenir, Mina Los Compadres, El Milagro, El Trinque and Leonor. The location of these claims is shown in Figure 4-2 of this report and details are given in Table 4-1 in Section 4.2. In essence the Bilbao mineralization is effectively protected on all sides by these surrounding claim holdings.

Most of the above claims have been explored by the Company at least by extensive soil sampling and in some cases by exploratory drilling of discovered targets. These claims have also been investigated by ground-based magnetometry.

Thus far no economically significant mineralization has been encountered on these adjacent claims although several promising zones have been identified for drilling at a future date. In respect of the latter one could mention the Pb-Ag soil anomaly, Ardillas within the northern part of the Bilbao II as a valid drill target to be investigated in the next phase of work. Furthermore there are other metal anomalies and ancient pits in the northern sector of Bilbao & Bilbao II which require further follow-up. Mention has already been made elsewhere in this report of the wollastonite deposits on El Porvenir claim which were worked until recently.

Immediately to the south of the Bilbao mineralization on the enclosed La Africana claim there occurs the old mine of El Cabezón this was a NW-SE trending Pb-Ag vein property. There is also a Ag-Pb-Zn vein structure 3km due south of the Bilbao property at La Aurora as well as several perlite prospects such as La Paloma & Potrero la Habana some five kilometers to the south-west in Tertiary rocks.

23.2 Source of Info on Adjacent Claims

Since most of the surrounding claims have been explored by the Company, the information on these comes from direct experience from ground explorations carried out within them. Other sources of information are in the public domain or in the reports of Servicios Geologico Minero (formerly Consejo de Recursos Minerales).

23.3 Influence of Adjacent Claim Mineralization on Bilbao

There are no direct influences on the Bilbao mineralization from adjacent claims save that of the soil contamination plume from the old processing plant at the old El Cabezón mine which affects the soil geochemistry in the far south of the Bilbao claim. It is possible that the vein system at El Cabezón trends into the western part of the Bilbao claim although this has not been established with any degree of certainty. The vein does strike into the company claim El Trinque to the south-east of La Africana.

Since the Bilbao mineralization is centrally placed within wholly-owned Company ground there is adequate cover of any potential extensions in all directions should further mineralization be discovered in the future.

23.4 Difference Between Adjacent Properties and Bilbao Claims

The claims surrounding the Bilbao property have a similar geological substrate in that they are underlain by limestone sequences with intrusive granite. As such there is a potential for discovery of similar deposits in the immediate vicinity. The Company has explored this limestone/granite contact both in the adjacent properties and in a regional sense. Whilst no economic deposits have been found thus far, promising mineralization has been located on the Company claims Gaby Marina & Cata Negra situated some 6km ESE of the Bilbao property.

23.5 Information on Mineral Resources of Adjacent Claims

Apart from wollastonite workings on the El Porvenir and La Güera claims, which are wholly owned enclaves within the Bilbao claims themselves and the Pb-Ag mineralization of El Cabezón, mentioned in Section 20.3 above, there are no other metallic mineral resources of economic import in adjacent claims. There is a borrow pit in the southern part of Bilbao II which formerly worked Chilitos volcanic rocks for road aggregate.

24. Other Relevant Data and Information

There is no other relevant data and information.

25. Interpretation & Conclusions

25.1 Mineralization

The Bilbao project is located in the Municipality of General Panfilo Natera, in the State of Zacatecas. The deposit is a contact metamorphic deposit, classified as skarn type. It is developed in the marbleized limestone at the contact with the La Blanca granodiorite (granite). The mineralization occurs as sulphide replacement bodies forming along the bedding in the limestone (mantos) and as minor replacement bodies in the intrusive (endoskarn). The highest grades are found in contact with the main intrusive body. Grades sharply decrease from the main intrusive body toward the west.

The principal contained economic metals are silver, zinc, copper and lead together with lesser amounts of gold, cadmium and tin. The deposit is weathered to an average depth of 120 metres so that the upper part of the mineralized body consists of iron oxides containing Ag-Zn-Pb-Cu. Below the oxide capping, the metals occur as sulfides. Based on past metallurgical testwork, it was deemed uneconomic to exploit the oxide material without further advances in processing technology. Xtierra's strategy has since switched to an underground approach to mine and process the sulphide minerals, namely silver, lead and zinc.

25.2 Resource Modeling

Since 2006, Xtierra has drilled 113 diamond drill-holes in the Bilbao deposit, with geological logs provided to RPM. The geological model was generated using 113 holes (all the logged drill holes). The block resource model was estimated using 105 holes which had assays.

For the purpose of determining resources at various cutoff grades, Zn equivalent values were defined, based on the average price of the last 3 years. The utilized prices were US$0.935 lb/Zn, US$1.008 lb/Pb, and US$30.235 oz/Ag. Metallurgical recoveries were applied in the equivalent equation as 76.7%, 90.6% and 73.4% for Zn, Pb, and Ag, respectively.

Total resources by mineral type at 3% Zn equivalent cutoff, not including approximately 1Mt of previously mined out ore, is as follows:

Table 25-1 Total Resources

Ore Type	Zn equiv. (%)	Indicated Tonnes	Inferred Tonnes	Total Tonnes	Zn (%)	Pb (%)	Ag (ppm)	Cu (%)
Oxide	6.50	791,082	3,069,582	3,860,664	1.70	2.33	42	0.17
Mixed	7.10	778,336	238,923	1,017,259	2.06	2.17	52	0.18
Sulphide	6.88	4,555,809	1,201,032	5,756,841	2.03	1.40	69	0.17
Total	6.76	6,125,227	4,509,537	10,634,764	1.91	1.81	58	0.17

25.3 Mining

The sulphide deposit is to be mined out as a long hole stoping operation utilizing a combination of LHD and 40t haul trucks. The mine production schedule resulted in an average annual production of 720,000 t of ROM ore at zinc, lead and silver grades of 2.1%, 1.4% and 63.96g/t respectively.

25.4 Recovery Methods

The mineral processing plant described is for the treatment of a silver-lead-zinc sulfide ore at a design throughput rate of 2,000 t/d. The mineral processing plant will produce lead-silver and zinc concentrates which will be transported off-site.

The process flow sheet selected for the Bilbao process plant comprises of two stages of crushing, two stages of grinding, lead rougher flotation, lead regrind, lead cleaner and lead concentrate and dewatering stages, zinc rougher flotation, zinc regrind, zinc cleaner flotation and zinc concentrate and dewatering stages.

Tailings from the zinc flotation circuit is pumped to the tailings thickener to produce a thickened tailings with 65% solids. The thickener is of conventional design with the addition of flocculant. Thickener overflow flows by gravity to the process water tank and thickener underflow is pumped to the tailings treatment facility.

The plant has an operating regime of 360 d/a, 7 d/w, 24 h/d and a plant utilization of 92%, resulting in an average nominal throughput of 91 t/h. The plant will produce, on average, 16,913 dry t/a of silver-rich lead concentrate, and 26,966 dry t/a of zinc concentrate. Plant recovery is estimated to be 76.7% for zinc, 90.6% for lead and 73.4% for silver over the life of the mine.

25.5 Environmental Studies

Several environmental or environmentally-related studies have been developed for the Bilbao project. These studies have provided detail on biodiversity baseline conditions, groundwater resources available in the region, and information on the potential for the Project to result in environmental contamination. Existing studies include:

25.5.1 Biodiversity Studies

These studies presented baseline flora and fauna data for the project site, as well as climate information, and detailed efforts that would be undertaken by the Project to avoid any sensitive cacti during exploration drilling, and to reclaim drilling pad locations.

25.5.2 Hydrology Studies

In 2009 Bilbao Resources contracted with Schlumberger Water Services (“SWS”) to characterize and identify a potential water source for the Project. The results of this initial investigation concluded that the Project would need to rely exclusively on ground water for needed make-up water during operations, as very little surface water exists in the region. A follow-up Phase 2 Hydrologic Assessment was issued by SWS in July, 2011. The Phase 2 Hydrologic Assessment identified a total of 184 production wells in existence within a study area of 10km from the Project site.

25.5.3 Geochemistry Studies

Geochemical modelling of waste rock samples has been performed to identify the potential for acid rock drainage. As a result acid generation is identified as an insignificant issue. The results provided in the Geochemical Results of Waste Rock and Tailings Samples Report are preliminary, and additional sampling is required to assess short-term and potential long-term metal leaching characteristics of the waste rock. Further sampling may be required to ensure conformance with best practice sampling guidance.

Short-term leach tests for tailings material indicated barium, manganese and zinc may leach at concentrations that are greater than the applicable Mexican Standards for Receiving Body of Water. In addition, NAG leach testing results indicate barium and manganese may leach at concentrations that exceed Mexican regulatory standards. Tailings process decant water (supernatant) quality has been modelled with indications that total ammonia, beryllium, manganese, selenium, and zinc concentrations would exceed applicable Mexican regulatory criteria. Given these findings, the Project plans to construct the tailings disposal facility (TDF) with an HDPE liner to prevent infiltration into groundwater.

25.5.4 Waste and Tailings Disposal, Site Monitoring and Water Management

A study was completed in 2014 by Golder including an update to the previous TDF design, the completion of a geotechnical investigation of the TDF site, and development of a flow model for the TDF.

The TDF will have a single tailings cell with enough capacity to contain the estimated 3.4 million m^3 of tailings material to be generated over the life of the Project. The perimeter dams for TDF cell will be constructed with rockfill. A settling pond will be allowed to form at the toe of tailings beach, and upstream of the separation (South) dam. This will allow settling of finer material prior to discharge to a reclaim pond which will be used to recycle water back to the processing circuit.

The anticipated water balance based on return of all tailings to the TDF (i.e., no use of paste backfill) has een determined. For steady state mining operations and under average climatic conditions the mill will require 238,134 m^3 of water on an annual basis.

Based on this updated water balance Golder estimates an annual water deficit in the Project process circuit of 107,101 m^3 under average meteorological conditions. Excess water may accumulate under 25-year wet meteorological conditions, and 245,459 m^3 of make-up water would be required under 25-year dry meteorological conditions.

Other conclusions from the water balance analysis include the following:

- During average precipitation years (2-year return period) and for wet years with a return period of 5 years or less make-up water will be required to support the mill.
- During exceptionally dry years with a 100-year return period approximately 270,000 m^3 of make-up water will be required to support the mill.
- During a wet year with a return period of 10 years or more there will be an accumulation of water beyond that needed to support the mill. A contingency plan for water storage would then be required.

Any water deficit will need to be addressed via supply from external water sources. The deficit may be less if actual underground mine inflow rates are greater than the 50 m^3/day currently anticipated. The Tailing Disposal Facility and Water Management Pre-Feasibility Report recommends that water be stored prior to commissioning and operation of the Project. The TDF conceptual design includes construction of the reclaim pond during the start-up phase of the Project.

Exploratory drilling for a suitable water source will be pursued in target zones identified in the 2011 Phase 2 Hydrologic Study. This exploratory drilling will include hydrologic pump tests to verify suitability of the identified resource over time. As mentioned previously water rights will need to be purchased or transferred from existing users in the region, as there is a long-standing ban on further groundwater withdrawal from the limited aquifers of Municipality of General Panfilo Natera.

25.5.5 Permitting

On March 27, 2013 Bilbao Resources, S.A. de C.V. contracted with the Mexican environmental consultancy SIICA to complete the MIA and ERA. At the time of writing these documents were under development, incorporating information from existing environmental studies as well as studies that are in progress. In addition to the MIA and ERA the environmental consultancy SIICA will assist in development of the required technical study to issue the CUS, and a Program for the Prevention of Accidents (PPA).

25.5.6 Social and Community Impact

Details of potential social and community impact will be addressed in the pending MIA. The State of Zacatecas has experience centuries of mining development, and anticipated impacts to the Project area of influence are expected to be positive including employment opportunities.

25.5.7 Closure Planning

Per regulatory requirement the MIA will contain information for how closure and reclamation will be accomplished at the end of mining. Typical design features include the channelling of surface waters into natural drainages, and scarifying and reseeding of waste rock features. Down gradient monitoring of water quality will be performed to ensure no remnant groundwater contamination is present. Conceptual closure information for the TDF is provided in the Tailing Disposal Facility and Water Management Pre-Feasibility Report. A 0.5 m thick compacted sand and gravel cover will be emplaced over the entire tailing surface and runoff sumps will be decommissioned. A small wetland will be allowed to form upstream of the reclaim pond dam to allow for sedimentation and evaporation of accumulated surface runoff.

25.6 Capital and Operating Costs

Project Capital Costs, as of April 2014, are estimated to be USD 99.5M including an allowance for contingencies of USD 8.7M, equivalent to 8.8% of total capital expenditure. The capital cost summary as presented in Table 25-2 outlines total pre-production capital of USD 91.2M and remaining other capital and sustaining capital costs of USD 8.3M for the 8 year production life, including acquisition to replace mine equipment fleet, plant and infrastructure.

Table 25-2 Capital Cost Summary - USD

Capital Expenditures	Pre-Production Year -1	LOM Production Year 1-8	Total
Exploration	600,000	-	600,000
Mine Facilities & Equipment	11,529,000	-	11,529,000
Mining Equipment - Leased		-	-
U/G Mine Development	3,509,000	3,229,000	6,738,000
Backfill Plant & Distribution System	500,000	600,000	1,100,000
Infrastructure	6,942,462	-	6,942,462
Surface Mobile Equipment	700,000	-	700,000
Processsing Plant	38,321,221	-	38,321,221
Tailings Disposal Facility	6,615,067	4,694,080	11,309,147
EPCM & Contractor O/H	10,318,448	-	10,318,448
Owners Costs	3,980,000	-	3,980,000
Reclamation and Closure		1,181,000	1,181,000
Working Capital	2,017,503	- 2,017,503	- 0
Additional Contingency	6,138,247	660,858	6,799,105
Total Capital Expenditures	**91,170,948**	**8,347,435**	**99,518,383**

The average unit cost for the operational activities is USD 66.90/t of ore. The breakdown of mining, processing, general and administration, freight and insurance, and smelting, refining and penalties is presented in Table 25-3.

Table 25-3 Average Unit Operating Cost

Operating Cost	USD/Tonne ROM
Mine	25.73
Process	13.21
Site G&A	5.00
Freight and Insurance	2.08
Smelting, Refining, Penalties	20.88
Total Unit Operating Cost	**66.90**

The lifetime annual average of all operating costs included from Years 1 to 8 amounts to USD 43.4M.

25.7 Economic Analysis

This preliminary economic assessment is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

The market prices projected in the cash flow analysis for zinc, lead and silver are based on USD 0.92/lb, 1.00/lb and 30.38/oz respectively.

Total revenue for the project is based on 720 kt/y production to be reached in production period 2 and continuing for the life of the project average USD 73.6 million per year (gross revenue). The current plan estimates 11k tonnes of zinc concentrate and 7k tonnes of lead concentrate in the first production year.

25.7.1 Pre-Tax Cash Flow

Pre-tax earnings total USD 59.9 million over the 8 year designated mine life. Economic results of the Project cash flow model indicate in Internal Rate of Return (IRR) of 13.2% and a Net Present Value (NPV) of USD 11.0M at a 10% discount rate. The ten percent discount rate is considered appropriate for this evaluation as the overall project risks are considered to be relatively low in terms of total capital committed, geological risk and market risk.

RPM developed a sensitivity analysis for the pre-tax cash flow model based on variations in key project elements of metal price, operating and capital costs. Results are seen in Table 25-4.

25.7.2 After-Tax Cash Flow

After-tax net cash flow totals USD 32.6 million over the 8 year designated mine life. Economic results of the Project cash flow model indicate in Internal Rate of Return (IRR) of 8.1% and a Net Present Value (NPV) of USD -5.8M at a 10% discount rate. The ten percent discount rate is considered appropriate for this evaluation as the overall project risks are considered to be relatively low in terms of total capital committed, geological risk and market risk.

RPM developed a sensitivity analysis for the after-tax cash flow model based on variations in key project elements of metal price, operating and capital costs. Results are seen in Table 25-5.

Table 25-4 Pre-Tax Sensitivity Analysis

Item	NPV (USD Million)	IRR (%)
Base Case	**11.0**	**13.2%**
Capital Cost +15%	-3.7	9.01%
Capital Cost -15%	25.7	18.53%
Operating Cost +15%	-23.0	2.08%
Operating Cost -15%	45.0	21.99%
Sale Price (Zinc) +15%	25.4	17.22%
Sale Price (Zinc) -15%	-3.5	8.93%
Sale Price (Lead) +15%	24.8	17.04%
Sale Price (Lead) -15%	-2.8	9.14%
Sale Price (Silver) +15%	32.6	18.94%
Sale Price (Silver) -15%	-10.6	6.55%
Mill Recovery (Zinc) +15%	25.4	17.22%
Mill Recovery (Zinc) -15%	-3.5	8.93%
Mill Recovery (Lead) +15%	24.8	17.04%
Mill Recovery (Lead) -15%	-2.8	9.14%
Mill Recovery (Silver) +15%	29.7	18.22%
Mill Recovery (Silver) -15%	-7.7	7.52%

Table 25-5 After-Tax Sensitivity Analysis

Item	NPV (USD Million)	IRR (%)
Base Case	**-5.8**	**8.1%**
Capital Cost +15%	-18.5	4.57%
Capital Cost -15%	5.9	12.19%
Operating Cost +15%	-34.6	-3.35%
Operating Cost -15%	19.4	15.83%
Sale Price (Zinc) +15%	4.1	11.31%
Sale Price (Zinc) -15%	-16.0	4.59%
Sale Price (Lead) +15%	3.7	11.19%
Sale Price (Lead) -15%	-15.6	4.74%
Sale Price (Silver) +15%	9.1	12.80%
Sale Price (Silver) -15%	-21.7	2.32%
Mill Recovery (Zinc) +15%	4.1	11.31%
Mill Recovery (Zinc) -15%	-16.0	4.59%
Mill Recovery (Lead) +15%	3.7	11.19%
Mill Recovery (Lead) -15%	-15.6	4.74%
Mill Recovery (Silver) +15%	7.1	12.21%
Mill Recovery (Silver) -15%	-19.3	3.26%

The following table summarizes the sensitivity of the discount rate used on the before and after-tax NPV and IRR.

Table 25-6 Discount Rate Sensitivity Analysis

	Pre-Tax		After-Tax	
Discount Rate	NPV	IRR	NPV	IRR
0%	59,864,314	13.24%	32,574,596	8.11%
8%	18,724,880	13.24%	358,817	8.11%
9%	14,747,296	13.24%	- 2,780,353	8.11%
10%	10,973,630	13.24%	- 5,763,493	8.11%
11%	7,390,818	13.24%	- 8,600,429	8.11%
12%	3,986,785	13.24%	- 11,300,255	8.11%

26. Recommendations

The Bilbao deposit contains a reasonable quantity of mineral resources between the oxide, transition, and sulphide mineral zones; however, the lack of metallurgical test data available for the transition zone and identified recovery challenges for the oxide zone currently limit the scope of this PEA to the total mineable sulphide resources to offset the capital costs associated with the project. Recommendations have been made throughout this section identifying various opportunities to increase the mineable resource and reduce operating costs through additional exploration and engineering, improving the overall economics of the project.

26.1.1 Drilling

- Additional definition drilling targeted at the Bilbao transition and sulphide zones could lead to re-classification of inferred resources to indicated resources, potentially contributing to the total mineable resource studied at the pre-feasibility level;
- Exploration drilling at the Bilbao 2 area, approximately 1.5 km south of Bilbao, has potential to offer additional mineral resources to the project due to the fact that current trenching, sampling and resulting soil geochemistry information identifies similarities between the two areas. An additional source of feed to the plant designed in this study could either lengthen the overall life of the mine, increase the daily production rate, or result in a combination of the two, improving the NPV and IRR of the project;
- RPM recommends reporting the detection limit by campaign/laboratory/method and assigning half detection limit values to assays under the detection limit and negative codes to non-sampling and non-recovery intervals. Core loggers considered intervals as “not sampled” in the case they were barren;
- Duplicates show consistently good repeatability in the 2011 scatterplots. RPM recommends indicating the nature of duplicates, coarse or fine, and incorporating the relative error – data percent graphs. The maximum error, currently accepted by industry, is 10% and 20% for 90% data for fine and coarse duplicates, respectively.

26.1.2 Sample Preparation, Analyses and Security

- Results out of the 2 STD lower and upper limits are greater than the industry accepted results of ±5%. Zn has 12%, Pb 7%, Ag 65%, and Cu 30% of the results outside the limits. RPM strongly recommends researching the source of these poor reference sample results. If these out-of-limit results are confined to certain assay batches, RPM recommends re-assaying those batches along with the appropriate QA/QC samples. If the out-of-limit results are random with all batches, RPM recommends sending out at least 10% of the pulps along with the appropriate QA/QC samples to a second lab for a check. If the biases of the assays of the standard samples are representative of the laboratory accuracy and the results from the core samples are similarly biased, the estimation of grade from these samples would be conservative.

26.1.3 Data Verification

- RPM spot checked three lab certificates, ICP certificate 2010 – 4529, 4523 and 4523-2, of the drill hole X-71. RPM detected differences at the third decimal in Zn-Pb; this is irrelevant for resource estimation, however, RPM recommends completely matching lab certificate and database. Zn, Pb, and Ag grade database mistakes were not found by RPM. (Due to the fact that database verification was not part of the original scope, RPM simply spot checked some information.) However, RPM considers it is essential to complete data verification of at least 10% of holes prior to a feasibility study (FS). This data verification should include:
    - iv) Field check of drill hole location;
    - v) Logging review; and
    - vi) Coordinate-logs-assay certificates – database comparison.

26.1.4 Mineral Resource Estimate

- RPM decided on defining grid spacing based on geological and grade continuity which shows a reasonable level of confidence to define measured, indicated and inferred resources. RPM recommends incorporating, in a feasibility study, the estimation errors associated with annual and quarter production panels to define indicated and measured resources, respectively;

26.1.5 Mining

- Level spacing resulting from the proposed stope design (without the use of cable bolt support for backs and walls) is 24 metres. The potential to increase level spacing and correspondingly reduce level development, through use of cable bolts, may lead to lower development costs and should be further assessed;
- The backfilling approach used in this study includes the use of cemented and uncemented rock fill. Further analysis of hydraulic and sand backfilling options, in terms of preparation and distribution, may further reduce overall operating costs;
- There may also be opportunity to reduce operating costs significantly (~$5/t to $6/t) by reducing the number of stopes filled with backfill all together. Further geotechnical study would need to be carried out for this scenario to better understand possible ore losses with pillars left in place, and possible recovery of these pillars through caving activity. Potential also exists for deferral of ramp and associated development;
- Inclusion of transition zone material in the mine plan should be investigated (requiring additional metallurgical testwork) to extend the life of mine and/or potentially increase the mining rate per year;
- Some degree of stope sequencing was achieved in the mine plan to improve mined grades in the opening years of the operation, but further optimization of stope sequencing leading to improved cash flow may be achievable and should be studied.

26.1.6 Metallurgy

Recommendations for future work on samples from the Bilbao deposit include:

- Further scoping level testwork, mineralogy, and flowsheet development on composite and variability samples from the oxide zone to identify the potential for additional economic recovery of metal values. Bilbao contains a substantial in-situ oxide resource of 3.8 million tonnes (3 million inferred and 791,000 indicated) at a Zn equivalent grade of 6.5%, and opportunities may include new technologies for leaching and gravity recovery, or high-grading of the oxide zone to focus specifically on the zinc and/or silver minerals.
- Additional variability testing of samples from the transition zone to better characterize the extent of float recoverable mineralization in this area;
- Mineralogical characterization of transition zone samples from different drill holes to develop correlations between lead and zinc deportment and core log data;
- Mineralogical characterization of sulphide variability composite LS-3 to compare with the results of the transition zone samples and determine if this sample represents another area of altered material.

26.1.7 Environmental Studies

26.1.7.1 Hydrology

The 2011 Phase 2 Hydrologic Assessment provides the following recommendations:

- A program of baseline groundwater quality and water levels should be established, to allow environmental monitoring over time once the mine is in operation.
- A hydrogeologic drilling investigation should be completed at candidate well locations near the mine site, as data obtained from this investigation would be necessary to allow any water rights transfers. Water

rights in Mexico are administered by the *Comision Nacional de Agua* ("National Water Commission" and "CONAGUA").

- Six target zones are identified near the Project area for exploratory drilling (at Las Borregas, Bilbao, and La Ardilla).
- Potential groundwater inflow towards the mine should be investigated to incorporate any necessary dewatering costs in feasibility programs.

26.1.7.2 Geochemistry

- The results provided in the Geochemical Results of Waste Rock and Tailings Samples Report are preliminary, and additional sampling is required is required to assess short-term and potential long-term metal leaching characteristics of the waste rock. Further sampling may be required to ensure conformance with best practice sampling guidance. The Geochemical Results of Waste Rock and Tailings Samples Report recommends that elemental analysis and a review of total waste rock tonnages and rock types be performed to verify that the current number of samples is consistent with accepted characterization guidelines. These efforts will be undertaken during the feasibility study stage of Project development;
- With respect to the leach test results regarding tailings and tailings supernatant quality – the Project plans to construct the TDF with a HDPE liner to prevent infiltration into groundwater. There is the potential for periodic releases of water from the TDF to the environment. In this event a water treatment facility may be required, and this potential should be evaluated during the feasibility stage of Project development. Additional static and possible kinetic testing should also be performed to allow for a more refined understanding of tailings geochemistry.

26.1.7.3 Waste and Tailings Disposal, Site Monitoring and Water Management

Additional studies are recommended in the Prefeasibility TDF Study Update. These studies will be completed during the feasibility stage of Project development and will include the following:

- Geotechnical drilling for a better understanding of geologic, geotechnical and hydrological conditions of the TDF area;
- Detailing of quantities of material available from potential borrow sources for TDF construction;
- Seismic hazard analysis to verify peak ground accelerations used in the TDF design;
- Confirmation of assumptions used to develop the water balance; and
- Additional geochemical testing of tailings material to determine the potential need for treatment of water which may be discharged during wet climatic conditions.

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