UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549

FORM 6-K/A
REPORT OF FOREIGN ISSUER PURSUANT TO
RULE 13a-16 AND 15d-16 UNDER THE
SECURITIES EXCHANGE ACT OF 1934

For the month of: July 2004
Commission File Number: 000-49946

Alamos Gold Inc.
(Translation of registrant's name into English)

Suite 1503, 110 Yong Street
Toronto, Ontario, Canada M5C 1T4
(Address of principal executive offices)

Suite 1400 – 400 Burrard Street
Vancouver, British Columbia, Canada V7X 1A6
(Former Name or Former Address, if Changed Since Last Report)

1. Technical Report, The Estrella Pit Development, Mulatos Sonora Mexico

Indicate by check mark whether the registrant files or will file annual reports under cover Form 20-F or Form 40-F.

Form 20- F...... **Form 40-F..XXX**

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(1): ____

Note: Regulation S-T Rule 101(b)(1) only permits the submission in paper of a Form 6-K if submitted solely to provide an attached annual report to security holders.

Indicate by check mark if the registrant is submitting the Form 6-K in paper as permitted by Regulation S-T Rule 101(b)(7): ____

Note: Regulation S-T Rule 101(b)(7) only permits the submission in paper of a Form 6-K if submitted to furnish a report or other document that the registrant foreign private issuer must furnish and make public under the laws of the jurisdiction in which the registrant is incorporated, domiciled or legally organized (the registrant's "home country"), or under the rules of the home country exchange on which the registrant's securities are traded, as long as the report or other document is not a press release, is not required to be and has not been distributed to the registrant's security holders, and, if discussing a material event, has already been the subject of a Form 6-K submission or other Commission filing on EDGAR.

Indicate by check mark whether by furnishing the information contained in this Form, the registrant is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.

Yes ..... **No ..XXX...**

If "Yes" is marked, indicate below the file number assigned to the registrant in connection with Rule 12g3-2(b): 82- _____

ALAMOS GOLD, INC.

and

MINAS de ORO NACIONAL, S.A. de C.V.

TECHNICAL REPORT
The Estrella Pit Development
Mulatos Sonora Mexico

Prepared by:

M3 Engineering and Technology Corporation
2440 W. Ruthrauff, Suite 170
Tucson, Arizona USA 85705

and

M3 Mexicana
Hermosillo Sonora Mexico
and Consultants

Authors:

Douglas Austin, P.E. (Arizona)
P. Eng (B.C. & ONT)
Qualified Person and Project Manager M3

Michael J. Lechner, R.G. & C.P.G.. Geology
President of RMI

John Marek, P.E. Mining
President of Independent Mining Consultants (IMC) of Tucson, AZ

Dr. Deepak Malhotra, P.H. D. Mineral Economics
President of Resources Development, Inc. (RDI) Denver, CO

Mr. Thomas L. Drielick, P.E. (M3 Arizona)
Process Engineer

Mr. Donald A. Clark, P.E. (M3 Arizona)
Mining Engineer

June 17, 2004
(As Revised July 14, 2004)

Signatures of Authors

Douglas Austin, P.E.   _____*"Douglas Austin"*_____   **July 14, 2004**

Thomas Dreilick, P.E.   _____*"Thomas Dreilick"*_____   **July 14, 2004**

Donald A. Clark, P.E.   _____*"Donald A. Clark"*_____   **July 14, 2004**

John Marek, P.E.   _____*"John Marek"*_____   **July 14, 2004**

Dr. Deepak Malhotra, P.H.D.   _____*"Deepak Malhotra"*_____   **July 14, 2004**

Michael J. Lechner, RPG, CPG   _____*"Michael J. Lechner"*_   **July 14, 2004**

CERTIFICATE OF QUALIFIED PERSON

I, Douglas Austin of 9632 E. Baker St, Tucson, AZ 85748 hereby certify:

1. I am a Graduate of the University of Saskatchewan (1963) and hold a B.Sc. degree in Electrical Engineering.
2. I am presently employed as a Project manager with M3 Engineering and Technology Corp. of 2440 W. Ruthrauff Rd., Suite 170 Tucson, AZ. I am currently a Senior Vice President and Director; I was also President and C.E.O. of M3 from 1991 to 2002.
3. I was a Plant Engineering Superintendent with Noranda. Since 1974 I have been a Project Manager or Project Director for various engineering companies for a number of mining projects worldwide, including Mexico. I have been employed by M3 since 1988.
4. I am a Licensed Professional Engineer in Arizona since 1988. I am also a member of the Association of Professional Engineers and Geoscientists of British Columbia, and the Association of Professional Engineers and Geoscientists of Ontario.
5. I have read the definitions of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for certain sections of this report utilizing data summarized in the References section of this report. A detailed description of the responsible author for each section of this report is found in Appendix V.
7. I have visited the Mulatos Property. I have had no direct involvement with Alamos Gold Inc.
8. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
9. I am independent of Alamos Gold Inc. applying all the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and NI 43-101F1 and the technical report has been prepared in compliance with that instrument and form.
11. I consent to the use of this report for the purpose of complying with the requirements set out in NI 43-101 to support the technical report "The Estrella Pit Development, Mulatos Sonora Mexico" for Alamos Gold Inc. to be submitted to SEDAR for electronic filing.

Douglas Austin P.E.

Dated at Tucson, AZ this 17th Day of June, 2004

CERTIFICATE OF QUALIFIED PERSON

I, Thomas L. Drielick of 831 W. San Martin Drive, Tucson AZ 85704, hereby certify:

1. I am a graduate of Michigan Technological University (1970) and hold a B.Sc. degree in Metallugical Engineering. I also have an MBA from Southern Illinois University.
2. I am presently employed as a Senior Process Engineer with M3 Engineering and Technology Corp., of 2440 W. Ruthrauff Rd., Suite 170 Tucson AZ 85705. I am a Vice President and Director of M3.
3. I have been employed by Newmont, Kennecott and M3 for 32 years. I have been employed with M3 for 15 years.
4. I am a Licensed Professional Engineer in the state of Arizona since 1989.
5. I have read the definitions of "Qualified Person" set out in NI 43-101 and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for certain sections of this report utilizing data summarized in the References section of this report. A detailed description of the responsible author for each section of this report is found in Appendix V.
7. I have not visited the Mulatos Property. I have had no direct involvement with Alamos Gold Inc.
8. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
9. I am independent of Alamos Gold Inc. applying all the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and NI 43-101F1 and the technical report has been prepared in compliance with that instrument and form.
11. I consent to the use of this report for the purpose of complying with the requirements set out in NI 43-101 to support the technical report "Estrella Pit Development, Mulatos Sonora Mexico" for Alamos Gold Inc.

Thomas L. Drielick P.E.

Dated at Tucson, AZ this 17th Day of June, 2004

CERTIFICATE OF QUALIFIED PERSON

I, Donald A. Clark, of 2440 W. Ruthrauff, Suite 170, Tucson, Arizona, herby certify:

1. I am a graduate of the Colorado School of Mines (1973) and hold an Engineer of Mines degree.
2. I am presently employed as a senior estimator with M3 Engineering & Technology Corp. of 2440 W. Ruthrauff, Suite 170, Tucson, Arizona.
3. I have been employed in my profession by various mining, construction and engineering consulting companies since my graduation in 1973.
4. I am a Licensed Professional Engineer especially qualified in Mining Engineering in the state of Idaho since 1979. I am not a member of any professional societies.
5. I have read the definitions of "Qualified Person" set out in NI 43-101 and certify that by reason of my education and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101.
6. I am responsible for certain sections of this report utilizing data summarized in the References section of this report. A detailed description of the responsible author for each section of this report is found in Appendix V.
7. I have visited the Mulatos Property; I have no direct involvement with Alamos Gold Inc.
8. I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the technical report, the omission to disclose which makes the technical report misleading.
9. I am independent of Alamos Gold Inc. applying to all the tests in Section 1.5 of NI 43-101.
10. I have read NI 43-101 and NI 43-101F1 and the technical report has been prepared in compliance with that instrument and form.
11. I consent to the use of this report for the purposes of complying with the requirements set out in NI 43-101 to support the technical report "The Estrella Pit Development, Mulatos Sonora Mexico" for Alamos Gold Inc. to be submitted to SEDAR for electronic filing.

Donald A. Clark P.E.

Dated at Tucson, AZ this 17th Day of June, 2004

CERTIFICATE OF QUALIFIED PERSON

I, Michael J. Lechner, President of Resource Modeling Incorporated, 1960 West Muirhead Loop, Tucson, AZ 85737 USA

do hereby certify that:

1.   I graduated from the University of Montana with a B.A degree in Geology in 1979;

I am a licensed Registered Geologist in the State of Arizona (RPG #37753) and a Certified Professional Geologist (CPG #10690) with the American Institute of Professional Geologists;

I have practiced my profession as a geologist continuously since graduation and have worked as an exploration geologist, mine geologist, geologic consultant, resource estimator and have been president of Resource Modeling Incorporated since October 2001;

and therefore meet the requirements of National Instrument 43-101 for designation as a Qualified Person.

2.   I have never visited the Mulatos Project site.

3.   I was a co-author for the M3 report dated June 17, 2004 entitled "The Estrella Pit Development Mulatos Sonora Mexico" for Alamos Gold Incorporated.

4.   I am not aware of any material fact or change that has not been disclosed in the documentation provided by Alamos Gold Incorporated, which is therefore not reflected in our technical report.

5.   I am independent of Alamos Gold Inc. in accordance with the requirements of National Instrument 43-101.

6.   National Instrument 43-101 and Form 43-101F have been read and our report has been prepared in accordance with the requirements specified therein.

Dated at Tucson, AZ this 17th Day of June, 2004

Signed Mike Lechner

Michael J. Lechner, RPG, CPG
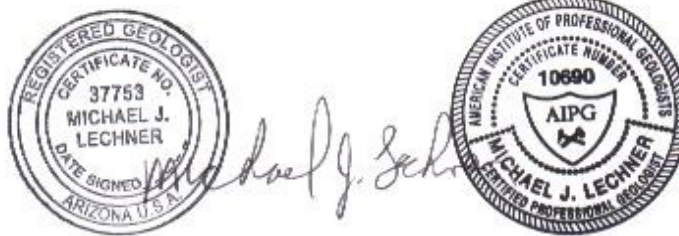
CERTIFICATE OF QUALIFIED PERSON

I, Deepak Malhotra, of 7 McIntyre Circle, Golden, Colorado, hereby certify:

1.   I am a graduate of Indian Institute of Technology, Kanpur. India (1970) and Colorado School of Mines, Colorado (1974, 1978) and hold a BS and MS degrees in Metallurgical engineering and a Ph.D. in mineral economics.

2.   I am presently employed as President of Resource Development Inc., 11475 W. I-70 Frontage Rd. North, Wheat Ridge, Colorado, a testing and consulting company.

3.   I have been employed in my profession by various mining companies since 1990. I worked for AMAX Inc. from 1973 to 1990.

4.   I am a member in good standing of the professional organizations, Society of Mining Engineers and Canadian Institute of Mining and Metallurgy.

5.   I have read the definitions of "Qualified Person" set out in NI43-101 and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purpose of NI43-101.

6.   I am responsible for certain sections (i.e., Metallurgy) of this report.

7.   I have not visited the Mulatos property.

8.   I am not aware of any material fact or material change with respect to the subject matter of the technical report that is not reflected in the technical report, the omission to disclose which makes the technical report misleading.

9.   I am independent of Alamos Gold Inc. and Minas de Oro National, S.A. de C.V.

10. I have read technical report.

11. I consent to the use of this report for the purpose of complying with the requirements set out in NI43-101.

"Deepak Malhotra"

Deepak Malhotra

Dated at Wheat Ridge, Colorado, this 17th day of June, 2004.

CERTIFICATE OF QUALIFIED PERSON

I, John M. Marek P.E. do hereby certify that:

1. I am currently employed as the President and a Senior Mining Engineer by:
    Independent Mining Consultants, Inc.
    2700 E. Executive Drive # 140
    Tucson, Arizona, USA 85706

2. I graduated with the following degrees from the Colorado School of Mines
    Bachelors of Science, Mineral Engineering – Physics   1974
    Masters of Science, Mining Engineering 1976

3. I am a Registered Professional Mining Engineer in the State of Arizona USA
    Registration # 12772
    I am a Registered Professional Engineer in the State of Colorado USA
    Registration # 16191

    I am a member of the American Institute of Mining and Metallurgical Engineers, Society of Mining Engineers

4. I have worked as a Mining Engineer for a total of 28 years since my graduation from university.

5. I have read the definition of "qualified person" set out in National Instrument 43-101 ("NI43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI43-101.

6. I acted in responsible charge of the preparation of the sections related to the open pit reserves of the report titled "TECHNICAL REPORT – The Estrella Pit Development– Mulatos Sonora Mexico" used in support of the technical report relating to Mulatos Project. I have not visited the Mulatos Project.

7. I have not had prior involvement with the property.

8. I am not aware of any material fact of material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.

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

Certificate of Qualified Person (continued) for John M. Marek, P.E.

10. I have read national Instrument 43-101 and Form 43-101F1, and to my knowledge, the Technical Report has been prepared in compliance with that instrument and form.

11. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated 17th Day of June 2004.

John M. Marek P.E.

FORM 43-101F1
TECHNICAL REPORT
TABLE OF CONTENTS

TITLE

CONTENTS OF THE TECHNICAL REPORT

LIST OF FIGURES CONTAINED IN ACCOMPANYING FILE

3 SUMMARY

3.1 LOCATION

The Salamandra Property, which encompasses a total of approximately 19,634 ha, is located in the Sierra Madre Occidental mountain range in the east central portion of the State of Sonora, Mexico. The property is located approximately 220 km by air east of the city of Hermosillo, and 300-km south of the border with the United States of America. (See Figure 1.1)

3.2 OWNERSHIP

The Salamandra Property consists of the Mulatos deposit and eight satellite gold systems known as El Halcon, La Yaqui, Los Bajios, El Jaspe, Cerro Pelon, El Victor/San Carlos, La Dura, and El Carricito. Mineral rights for all concessions comprising the Salamandra Property are controlled by Minas de Oro Nacional, S.A. de C.V., a Mexican company, wholly owned by Alamos Gold, Inc. a British Columbia corporation. A net smelter return royalty is due to the Placer Dome/Kennecott consortium on the first 2,000,000 ounces of product.

3.3 GEOLOGY

The Salamandra mineral deposits are large epithermal, high-sulfidation, disseminated, gold deposits hosted within a mid-Tertiary dacite and rhyodacite dome complex. Gold mineralization is closely associated with silicic alteration. It also is associated with a large hydrothermal alteration zone that covers more than 10km^2. The Mulatos deposit is composed of subdeposits known as Estrella, Mina Vieja, Escondida, El Victor and San Carlos and hosts the only economic mineralization delineated to-date.

3.4 MINERALIZED RESOURCES

The exploration programs completed by Alamos, Placer Dome, Kennecott and Minera Real de Angles have delineated measured and indicated resources of 62.2 million metric tonnes @ 1.51 grams per metric tonne Au and 0.6 grams per metric tonne Ag, which contain 3,020,000 oz of Gold and a small amount of silver. Inferred material is not included as part of the resource tabulation.

These resources are contained in the Estrella, Mina Vieja and Escondida areas only of the Mulatos deposit. Gap, El Victor and San Carlos portions are not included. The area is similar to the Placer/ M3 studies of 1997 and 2000. The resource model is more conservative than Placers.

3.5   EXPLORATION POTENTIAL

In addition to the Mulatos/El Victor zones, the eight satellite systems have known gold mineralization with varying levels of exploration investigation.

1. El Halcon: Drill indicated resources.
2. La Yaqui: Drill indicated resources.
3. Los Bajios: Untested exploration target.
4. El Jaspe: Geochemical anomaly not drill tested.
5. Cerro Pellon: Geochemical anolmaly not drill tested
6. El Victor/San Carlos: Former Producer
7. La Dura: Untested exploration target.
8. El Carricito: Untested exploration target.

3.6   METALLURGY

The Mulatos deposit and surrounding deposits are amenable to cyanidation and heap leaching, as determined by lab scale testing. Mineralized material varies from pure oxide to pure sulfide, with gold recovery typically decreasing from +90% to 55% as material grades from oxide to sulfide. The average recovery is estimated to be between 72 and 74% for the Estrella pit. Applying the modified recovery formulas to the block model has resulted in an estimated average recovery of 72.9%.

3.7   ESTRELLA PIT DEVELOPMENT

M3 has completed a Technical Report Entitled "The Estrella Pit Resource and Reserves" dated May 19, 2004 for Alamos Gold Inc. On June 1, 2004 M3 published a Feasibility Study for Alamos at a production rate of 10,000 M.T.P.D. to a heap leach pad based on 350 operating days per year.

Significant statistics of this feasibility study are:

Ore Reserve

ESTRELLA PIT ONLY – Mina Vieja and Escondida Areas NOT Included

The previously reported sum of the proven and probable open pit reserve was 37.5 Mt @ 1.61 g/t Au using an internal cutoff grade which varies by ore type from 0.34 g/t in the oxide to 0.63 g/t in the silicified ore type.

A mine production schedule has been developed with an elevated cutoff grade strategy in the early years. This approach improves the cash flow of the project. There is no stockpiling of the low grade currently planned. The sum of the mine plan reserve is:

Proven Reserve: 7.3 Mt @ 1.84 g/t Au
Probable Reserve: 29.1 Mt @ 1.59 g/t Au
Total   36.4 Mt @ 1.64 g/t Au

Production Quantities

Mining Method: Open Pit
Waste: Ore Ratio: 1.34:1
Total Material Moved   87.9 Mt

Ore Processing Method   Crushed Heap Leach, 80% minus 9.5mm (3/8")
Processing Rate: 10,000 t/d
Gold Recovery: 72.9 %

Metal Production:
Gold: 1,396,083 oz
Silver, Au Equiv.:   12,830 oz (748,388 oz Ag)

Gold Equiv.:   1,408,901 oz
Production Life: 10.5 Years

Prices (In US Dollars)

Gold Price: $350/oz
Silver Price: $6.00/oz
Exchange Rate: NP$10= $1.00 U.S.

Capital Cost (In US Dollars)

Initial Capital Cost $72,202,000
Sustaining Capital $10,284,000

Project Economics with 50% Alamos Equity (In US Dollars)

Net Present Value (NPV) at 0% $90,758,350
Net Present Value (NPV) at 7% $45,530,616
Net Present Value (NPV) at 10% $32,794,204
Internal Rate of Return   23.3%

Cash Cost $174.50/oz *
Total Cost $239.20/oz **
Operating Cost $6.54/tonne *

* Excluding property tax, Ejido payment, reclamation and closure bond fee and costs,
    and depreciation.
** Includes above items

The Project Economics assume:
- 50% Equity Contributed by Alamos
- All Mexican labor is subcontracted
- After Mexican Taxes
- The Salamandra Property is subject to a Royalty for Technical Expertise Agreement dated March 23, 2001 between Minas de Oro Nacional and Minera San Augusto (the "RTE Agreement"). Pursuant to the RTE Agreement, the Company is obliged to pay to Tenedoramex and Kennecott, as assignees, the following royalty (the "Placer/Kennecott Royalty") from the date of Commencement of Commercial Production (as defined in the RTE Agreement) until such time as the first 2,000,000 ounces of gold have been mined, processed and sold (or deemed sold) from the Salamandra Property:

    (a)   2% of the Net Smelter Returns (as defined in the RTE Agreement) in respect of all Products except gold and silver (as defined in the RTE Agreement) mined and sold (or deemed sold) by Minas de Oro Nacional from the Salamandra Property; and

    (b)   the applicable percentage based upon the Gold Price (as defined in the RTE Agreement) as published in the Wall Street Journal for the calendar quarter in which the royalty is payable of the Net Smelter Returns in respect of all Silver and Gold Products (as defined in the RTE Agreement) mined and sold (or deemed sold) by Minas de Oro Nacional from the Salamandra Property as follows:

Gold Price Range	Net Smelter Return Royalty 100% Basis
US$0.00/oz to US$299.99/oz	1.0%
US$300.00/oz to US$324.99/oz	1.5%
US$325.00/oz to US$349.99/oz	2.0%
US$350.00/oz to US$374.99/oz	3.0%
US$375.00/oz to US$399.99/oz	4.0%
US$400.00/oz or higher	5.0%

The term "Products" (as defined in the RTE Agreement) means ores, minerals, or other commercially valuable products, except any fraction thereof comprising or deemed to comprise Gold and Silver Products, mined from the Salamandra Property.

The term "Gold and Silver Products"(as defined in the RTE Agreement) means ores, minerals, or other commercially valuable products containing gold or silver mined from the Salamandra Property, provided that where such products contain a combination of gold and silver and other commercially viable metals or minerals, Gold and Silver Products shall be deemed to comprise on that fraction of such products as represents the proportionate commercial value of the gold and silver contained in such products, with the remaining fraction of such products deemed to be Products.

3.8 ENVIRONMENTAL

Acid rock drainage (ARD) potential has been identified. Measures to prevent ARD have been incorporated.

Mexican norms, World Bank Guidelines and "Equatorial Principles" have been followed.

3.9 SOCIAL ISSUES

The nearby village of Mulatos should be largely protected from noise, dust, vibration and fly rock by the Mina Vieja outcrop which will not be mined at this stage. Money is provided for the installation of water, sewer and power to the village.

4 INTRODUCTION AND TERMS OF REFERENCE

4.1 TERMS OF REFERENCE

M3 Engineering and Technology (M3) assisted Placer Dome with their 1997 Mulatos Feasibility Study and 2000 update. When Alamos Gold purchased an interest in Mulatos from National Gold in 2002, M3 was asked to assist. In 2003 Alamos Gold took over the project and M3 continues to assist the owner and has prepared a new feasibility study. On May 19, 2004 M3 issued a Technical Report "The Estrella Pit Resource and Reserves Mulatos Sonora, Mexico" for Alamos. On June 1, 2004 M3 issued Alamos' "Mulatos Feasibility Study Phase One – Estrella Pit" which necessitated this technical report.

The nominated sub-consultants include:

- Mintec inc., Tucson Arizona and Resource Modeling, Inc. (RMI), Tucson Arizona did the mineralized model and resource estimate.

- Independent Mining Consultants, Inc. (IMC), Tucson Arizona did the mine model, reserve estimate and pit cost estimates.

- Resource Development, Inc. (RDI), Denver Colorado reviewed past metallurgical testing and directed the recent metallurgical testing at METCON Research, Inc., Polysius Research Center and at RDI. RDI recommended the metallurgical recoveries for this study.

- Water Management Consultants, Inc. (WMC), Denver Colorado did the water resource planning.

- AGRA Earth and Environmental now AMEC E&C Services, Ltd., did the leach pad design which is unchanged from the 1997 and 2000 Placer Dome Study.

- Ken Balleweg of Alamos is directing the geological program.

- Laura Cabellero of Alamos is directing the environmental program.

4.2   PURPOSE FOR WHICH THE TECHNICAL REPORT WAS PREPARED

The main objective of this report is to give Alamos Gold an independent opinion regarding the potential development of the Estrella Pit portion of the Mulattos deposit. It was prepared in accordance with Canadian National Instrument 43-101 requirements.

4.3   THE SOURCE OF DATA

Reference is made to the 1997 Placer Dome Feasibility and 2000 update with which M3 also assisted.

In January 2001, Behre Dolbear prepared a qualifying report for National Gold. The suggestions made by Behre Dolbear have been incorporated into this new geological model.

In 2002 Pincock Allen and Holt (P.A.H.) did a preliminary scoping study, for a smaller pit.

On May 19, 2004 M3 issued a Technical Report "The Estrella Pit Resource and Reserves Mulatos Sonora, Mexico" for Alamos. On June 1, 2004, M3 issued Alamos' "Mulatos Feasibility Study Phase One – Estrella Pit".

4.4   THE EXTENT OF FIELD INVOLVEMENT

Mr. Doug Austin, P.E. visited the site again in October 2003 for 2 days. He had previously been on the site during preparation of the 1997 Placer Dome Feasibility Study.

5   DISCLAIMER

M3 has not verified the land status itself.

6   PROPERTY DESCRIPTION AND LOCATION

6.1   AREA OF THE PROPERTY IN HECTARES

5,,070.8779 hectares held are licensed for exploration and MON has applied for exploration permits for 1, 729.4533 hectares. 12,834.4329 hectares held are licensed for exploitation. The above areas were awarded by the Mexican Department of Economy Direcion General of Mines.

6.2   LOCATION REPORTING

Location reported by Section, township, range mining division or district municipality province state, country and national topographic system designation or universal transverse mercator (UTM) system as applicable, reported by latitude and longitude:

- The claims are in the Sahuarita Sonora Mexico municipal region. Mexico uses the U.T.M. system. The claims lie generally between 700,000 meters and 730,000 meters east and 3,160,000 meters and 3,185,000 meters north as shown on the map in Table 6-1.

6.3   CLAIM

The claim number or equivalent, whether patented or unpatented or the applicable characterization in the jurisdiction in which they are situated, and whether the claims are contiguous. "Mon" refers to Minas de Oro Nacional, S.A. de C.V., a wholly owned subsidiary of Alamos Gold.

The following list provides the name of the lot, the name of the holder, file, title, date of issuance of the title, the area of the concession and the expiry of title that form part of the Mulatos Project.

Lot Name	Holder	File	Title	Date of Title	Expiration Date	Area in Has.
Poryecto Mulatos, Sahuaripa, son.						
Explotaition Concession						
Alejandra	MON	4/1.3/1632	217765	13-Ago-02	12-Ago-52	405.6606
Betty	MON	321.1/4-700	191273	19-Dic-92	18-Dic-41	453.7237
Capulin 2	MON	4/2.4/01996	217556	16-Jul-52	15-Jul-52	12.0000
Carolina	MON	321.1/4-701	191272	19-Dic-91	18-Dic-41	347.0000
Cont. De Virgencita	OCAÑA	321.1/4-632	190634	29-Abr-91	28-Abr-41	100.0000
Cristina	MON	321.1/4-704	191271	19-Dic-91	18-Dec-41	290.0000
El Jaspe	MON	4/1.3/1611	209714	03-Ago-99	02-Ago-49	78.0000
El Marrano	MON	4/1.3/2004	217518	16-Jul-02	15-Jul-52	434.0000
El Victor De Mulatos	MON	82/6061	196110	23-Sep-92	22-Sep-42	18.0000
La Central	MON	82/7157	196111	23-Sep-92	22-Sep-42	96.0000
La Central No. 1	MON	82/2310	196108	23-Sep-92	22-Sep-42	81.2560
Mirtha	MON	4/1.3/1471	206755	12-Mar-98	11-Mar-48	470.3190
Nuevo Mulatos	MON	82/0891	180600	13- Jul -87	12-Jul-37	30.0000
Salamandra Fraccion 1	MON	45/2.4/01966	212185	30-Ago-96	29-Ago-46	8,072.6559
Salamandra Fraccion 2	MON	4/2.4/01966	212186	30-Ago-96	29-Ago-46	1,161.5005
Salamandra Fraccion 3	MON	4/2.4/01966	212187	30-Ago-96	29-Ago-46	604.000
San Carlos	MON	82/2289	196112	23-Sep-92	22-Sep-42	9.0000
San Lorenzo	MON	4/1.3/1633	210493	08-Oct-99	09-Oct-49	60.0000
San Lorenzo	MON	4/1.3/1739	211573	26-Jun-00	15-Jun-50	15.6160
San Miguel 2	MON	321.1/4-703	195438	14-Sep-92	13-Sep-42	20.2516
San Miguel 1	MON	321.1/4-702	191139	29-Abr-91	28-Abr-41	16.7056
Tequila	MON	4/1.3/1470	206724	12-Mar-98	11-Mar-48	18.7440
La Estrella	MON	4/1.3/1919	217206	25-Jul-02	24-Jul-52	40.0000
						12,834.4329
Exploration Concessión						
EL CARRICITO	MON	82/19625	206895	03-Abr-98	02-Abr-04	2,176.8440
EL CARRICITO 2	MON	82/26288	212507	31-Oct-00	30-Oct-06	100.0000
CERRO PELON	MON	82/26815	213670	08-Jun-01	07-Jun-07	500.0000
CERRO PELÓN 2	MON	82/26914	214866	04-Dic-01	03-Dic-07	500.0000
LOS COMPADRES	MON	82/28236	218820	21-Ene-03	20-Ene-09	10.0000
CARBONERAS	MON	82/28557	220715	30-Sep-03	29-Sep-09	801.3822
CARBONERAS 2	MON	82/28680	221518	19-Feb-04	18-Feb-10	132.0000
OSTIMURI 1	MON	82/28803	2221082	7-May-04	6-May-10	482.6517
CARBONERAS 3	MON	82/28841	In process	In Process	In Process	1,729.4533
CERRO PELÓN 3	MON	82/27376	216744	28-May-02	27-May-08	368.0000
						6,800.3312

On May 27, 2004 the Company reached a new surface rights agreement (the "New Surface Agreement") with the Ejido Mulatos (the "Ejido"), a local community of people that own the surface rights to an area of land covering all of the known mineral deposits in the Mulatos area of the Salamandra Property. The New Surface Agreement is required in order for the Company to hold surface rights for the development of certain mineral concessions on the Salamandra

Property and enables the Company to locate the crushing plant, waste dumps, recovery plant, and heap leach pad sites. The New Surface Agreement enables the Company to perform different mining works and activities and set up the infrastructure required for the Company's exploration and exploitation of the mining concessions. The New Surface Agreement, which was approved on May 27, 2004 by the Agrarian Court for the State of Sonora, supersedes the 1995 Surface Agreement. The Company is no longer bound by the 1995 Surface Agreement.

The New Surface Agreement provides, among other things, for the lease of 1,200 hectares of land for an initial term of eight years with an option to extend the term for further ten year term. If the 1,200 hectares of land are divided into parcels and the Ejido assign their rights to individual possessors of the land, the New Surface Agreement grants the right to Minas de Oro Nacional to negotiate for the purchase of this land from or negotiate a lease with the individual possessors.

On May 27, 2004 the Company also entered into a settlement agreement (the "Settlement Agreement") with the Ejido, to settle two outstanding appeals by the Company to a legal action by the Ejido disputing the 1995 Surface Agreement, the annual surface rights lease payments due to them in respect of the Salamandra Property and the ability of the Company to reduce the annual lease payments. The Settlement Agreement is binding on all members of the Ejido and precludes the Ejido, its lawyers, individual Ejidatarios or their legal representatives from commencing any legal action against Minera de Oro Nacional or the Company over land issues covered thereby.

The company continues to negotiate purchase agreements with individual Ejido residents.

6.4 PERMIT LIST

The following is a listing of permit status. Most permits are in place. The owner is working on those not yet in place.

Permit/Approval Name	Approving Authority	Approval Status
Pre-Construction Period		
Land Use Agreement	Ejido Mulatos	Completed
Manifesto Impacto Ambiental	SMARNAP - INE	Approved
Construction Water Well	Comision National del Agua (CNA)	Approved
Surface Use Change	SMARNAP	Pending
Access Road	SMARNAP - INE	Approved
Land for Mulatos Town Relocation	Ejido Mulatos, Municipality of Sahuaripa	In-progress
Mulatos Town Relocation	SMARNAP - INE	Approved
Mulatos Town Access Road	SMARNAP - INE	Approved
Transmission Line Right-of-Way	Local Landowners	In-progress
Power Transmission Line	INI, CFE	In-progress
Sand and Gravel Barrow Pit	Comision National del Agua	Approved
Clay Barrow Pit	SMARNAP - INE	Approved
Access Road Right-of-Way	Local Landowners	In-progress
Access Road Construction Water Supply	Comision National del Agua Town of Yecora	Partial Approval
Equipment Importation Permit	Hacienda	In-progress
Pre-Operations Period		
Operations Water Supply	SIUE	Yes
Garbage Dump	SIUE	No
Camp Sewage Treatment Plant	Comision National del Agua (CNA)	No
Camp Water Supply	Comision National del Agua (CNA)	Approved
Air Quality Permit	SMARNAP	No
Explosives Permit – Mine	SEDENA	Yes
Explosives Permit – Road Construction	SEDENA	Pending
Operations Period		
Closure Plan	SMARNAP	No

6.5 NATURE AND EXTENT

The nature and extent of the issuer's title to the property including surface rights, the obligations that must be met to retain the property, and the expiration date of claims, licenses or other property tenure rights.

The mineral rights claims were issued by the Mexican Department of Economy, Direcion General of Mines (SEMARNAP) as described in 6.1, 6.2 and 6.3 above.

Surface rights in the exploitation area are held privately and by the Mexican Government through the "Ejido Mulatos". Ejidos are Agrarian land grants to a group of people.

MON has entered into a new surface rights agreement dated May 27, 2004 with the Ejido Mulatos (the "New Surface Agreement") which is required in order for MON to perform different mining works and activities and set up the infrastructure required for MON's exploration and exploitation of the mining concessions. The New Surface Agreement was approved on May 27, 2004 by the Agrarian Court for the State of Sonora and supersedes a previous surface rights agreement.

The New Surface Agreement provides, among other things, for the lease of 1,200 hectares of land for an initial term of eight years with an option to extend the term for further ten year term. If the 1,200 hectares of land are divided into parcels and the Ejido assigns their rights to individual possessors of the land, the New Surface Agreement grants the right to MON to negotiate for the purchase of this land from or negotiate a lease with the individual possessors.

The Salamandra Property is subject to a Royalty for Technical Expertise Agreement dated March 23, 2001 between Minas de Oro Nacional and Minera San Augusto (the "RTE Agreement"). Pursuant to the RTE Agreement, the Company is obliged to pay to Tenedoramex and Kennecott, as assignees, the following royalty from the date of Commencement of Commercial Production (as defined in the RTE Agreement) until such time as the first 2,000,000 ounces of gold have been mined, processed and sold (or deemed sold) from the Salamandra Property:

(a)   2% of the Net Smelter Returns (as defined in the RTE Agreement) in respect of all Products except gold and silver (as defined in the RTE Agreement) mined and sold (or deemed sold) by Minas de Oro Nacional from the Salamandra Property; and

(b)   the applicable percentage based upon the Gold Price (as defined in the RTE Agreement) as published in the Wall Street Journal for the calendar quarter in which the royalty is payable of the Net Smelter Returns in

respect of all Silver and Gold Products (as defined in the RTE Agreement) mined and sold (or deemed sold) by Minas de Oro Nacional from the Salamandra Property as follows:

Gold Price Range	Net Smelter Return Royalty 100% Basis
US$0.00/oz to US$299.99/oz	1.0%
US$300.00/oz to US$324.99/oz	1.5%
US$325.00/oz to US$349.99/oz	2.0%
US$350.00/oz to US$374.99/oz	3.0%
US$375.00/oz to US$399.99/oz	4.0%
US$400.00/oz or higher	5.0%

The term "Products" (as defined in the RTE Agreement) means ores, minerals, or other commercially valuable products, except any fraction therof comprising or deemed to comprise Gold and Silver Products, mined from the Salamadra Property.

The term "Gold and Silver Products"(as defined in the RTE Agreement") means ores, minerals, or other commercially valuable products containing gold or silver mined from the Salamandra Property, provided that where such products contain a combination of gold and silver and other commercially viable metals or minerals, Gold and Silver Products shall be deemed to comprise on that fraction of such products as represents the proportionate commercial value of the gold and silver contained in such products, with the remaining fraction of such products deemed to be Products.

The expiration dates of each claim are listed in the table contained in 6.3.

6.6   WHETHER OR NOT THE PROPERTY HAS BEEN LEGALLY SURVEYED

The Mulatos area was flown by Cooper Aerial surveys headquartered in Tucson, Arizona in the 1990's. "Orthoshop: Hermosillo office did the digitizing for the exploitation area. A specially licensed Mexican mining engineer (a "perito") has stamped the claims map.

The location of all known mineralized zones, mineral resources, mineral reserves and mine workings, existing takings ponds, waste deposits and important natural

features and improvements, relative to the outside property boundaries by showing the same on a map.

6.7   TERMS

To the extent known, the terms of royalties, back in rights, payments or other agreements and encumbrances to which the property is subject.

Item 6.4 above describes the Ejido commitments and Placer Dome/ Kennecott N.S.R.

6.8   ENVIRONMENTAL LIABILITIES

To the extent known all environmental liabilities to which the project is subject.

As described in the history section 8, the area was first discovered by European Jesuit priests in 1635. Considerable small and medium scale underground and placer mining occurred up to the Mexican revolution in 1917. Since then a number of companies have done exploration work.

The Mulatos River flows northward 1½ km east of the Estrella pit eastern boundary. The pit eastern boundary is the high point of land and so the pit and mine dump area do not drain directly into the Mulatos River they drain naturally into the Mulatos wash. The Mulatos wash (Arroyo) does not flow continuously. It discharges into the Mulatos River several kilometers north of the mine. There is evidence in the Mulatos wash, which will form the pit northwest boundary of some acid drainage. About 70% of the Estrella pit is sulfide ore. Means have been established to contain acid water. These means include capping the waste dump during and after mining and dams and a 48" storm water pipe through the area which will be disturbed. This pipe will bypass the upstream Mulatos wash storm water through the mining area.

The Ejido village of Mulatos lies on the west side of Mulatos wash ½ km northwest of the Estrella pit. The village was established to serve the Mina Vieja Deposit; two portals of which are clearly in view from the village. The Mina Vieja outcrop was included in the minable resources in the Placer Dome Feasibility study. It has not been included in the Alamos 2004 Feasibility Study and will serve as buffer to the village, being between the village and pit.

The costs in the Alamos Feasibility study include costs for power, water and sewer utilities for the Mulatos village.

7 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

7.1 TOPOGRAPHY, ELEVATION AND VEGETATION

The project is located in rugged mountains in east central Sonora Mexico, just west of the Chihuahua border. The low project elevation is 950 meters at the Mulatos River, 1½ km east of the Estrella Pit. Average project elevation is 1,400 meters with peaks rising to 1,700 meters. The higher elevations host ponderosa pine, the lower elevations shrubs and cactus.

7.2 THE MEANS OF ACCESS TO THE PROPERTY

It is now 8 hours (380 kilometers) by road from Hermosillo Sonora the state capital. Road construction projects incorporated in this project will shorten the driving time to 6 ½ hours.

Highway #16 is narrow but paved from Hermosillo and passes south of the project from the highway to Mulatos. Dirt roads from Highway #16 to site are being improved.

Various small unpaved airstrips exist in the area. The nearest serviceable air strip is 15 km to the east at Matarachi.

7.3 PROXIMITY OF PROPERTY TO POPULATION CENTER AND NATURE OF TRANSPORT.

The village of Mulatos has no power, water or sewer systems. Some deposits in the area have been mined intermittently for 400 years. Alamos plans to install utilities for the village.

The village is located ½ km north east of the Estrella Pit. It was located close to and shielded somewhat from the Estrella Pit and by the Mina Vieja outcrop, which was mined by underground means. The village has approximately 100 structures and 300 population.

Matarchi is a slightly larger town with an air strip located 15 km the northest.

The larger towns in the area are Yecora with a population of 10,000 located to the south of Mulatos and Sahuara with a population of 7,000 located to the north of Mulatos. Both towns are within 100 km of site but take approximately 3 hours to drive to.

Transport in the area is largely by "pickup truck". The mine will have buses and accommodations for 50% of the workforce at a time. The other 50% will be on "offshift" or living in one of the surrounding communities.

7.4 CLIMATE AND LENGTH OF OPERATING SEASON

To the extent relevant to the mineral project, the climate and length of the operating season.

From July to September, the air is humid and hot, typically 35°C. In this period called the "monsoon" over half of the average rainfall of 0.8 meters falls. The winter months are cooler and an occasional frost of -2°C occurs. None of this restricts the mining activities.

7.5 SUFFICIENCY AND SURFACE RIGHTS

To the extent relevant the sufficiency of surface rights for mining operations, the availability and sources of power, water, mining, personnel, potential waste disposal areas, heap leach pad areas and potential processing plant sites.

The problem is to find a flat enough surface to locate the required areas and sites for mining operations and infrastructure, as the terrain is mostly up and down.

On May 27, 2004 the Company reached a new surface rights agreement (the "New Surface Agreement") with the Ejido Mulatos (the "Ejido"), a local community of people that own the surface rights to an area of land covering all of the known mineral deposits in the Mulatos area of the Salamandra Property. The New Surface Agreement is required in order for the Company to hold surface rights for the development of certain mineral concessions on the Salamandra Property and enables the Company to locate the crushing plant, waste dumps, recovery plant, and heap leach pad sites. The New Surface Agreement enables the Company to perform different mining works and activities and set up the infrastructure required for the Company's exploration and exploitation of the mining concessions. The New Surface Agreement, which was approved on May 27, 2004 by the Agrarian Court for the State of Sonora, supersedes the 1995 Surface Agreement. The Company is no longer bound by the 1995 Surface Agreement.

The company continues to negotiate purchase agreements with individual Ejido residents. The New Surface Agreement provides, among other things, for the lease of 1,200 hectares of land for an initial term of eight years with an option to extend the term for further ten year term. If the 1,200 hectares of land are divided into parcels and the Ejido assign their rights to individual possessors of the land, the New Surface Agreement grants the right to Minas de Oro Nacional to negotiate for the purchase of this land from or negotiate a lease with the individual possessors.

Power will be produced on site by 5- 1000 kW diesel generators. Quotations have been obtained from the national government owned utility "C.F.E." for a 115 kV

line to Mulatos or in conjunction with other developing mines in the area but this alternative is too expensive initially.

Alamos has purchased water rights. The water will come from the nearby Mulatos River and small local impoundments.

Skilled miners are available in Sonora. A minimum of expatriate supervisors is allowed for.

Mine waste pile disposal is centered on Mulatos wash. A continuous 2 km long 48 inch pipe under the waste pile and dams bypasses runoff water from upstream undisturbed areas. The mine plans to resurface finished dump areas on a continuous basis, to minimize rainfall infiltration and the potential generation of acid drainage from the dumps.

The heap leach pad is located 2 km south west of the pit in a relatively flat area. The ponds and A.D.R. plant are located at the south end of the leach pad. Later an area on the north side of the phase I pad will be constructed.

8   HISTORY

8.1   THE PRIOR OWNERSHIP OF THE PROPERTY AND OWNERSHIP CHANGES

Mulatos was first discovered in 1635 by Jesuit priests. The area saw considerable activity by various groups throughout the 1800's and 1900's. The owner of the first registered claim was Thomas Suza, in 1806. Succeeding owners include: N.Y. Ancheta and Ramon Bringas in 1821 and Mr. Ortese in 1863. In 1869, the property was bought by the Aguayo brothers. In 1887, they sold it to Hobart and Hayward of San Francisco, California. After a long lawsuit in 1890, the property was given to the Rey del Oro Mining Company and later transferred to Greene Gold Silver Company, which worked the claim until the Mexican Revolution in 1910.

Companies that have been interested in the district since 1960 include: Phillips Petroleum in 1962, Theodore A. Dodge in 1963, Cannon Hicks Associates in 1972, Tormex Developers in 1973, Explomin S.A. de C.V. in 1974 (formerly part of Minera Real de Angles), Homestake Mining Company in 1975, British Petroleum in 1982, Papanton Minas in 1984, and Kennecott in 1990.

Kennecott conducted exploration activities on the ground surrounding the Nuevo Mulatos and Tequila claims for many years. Their efforts focused on the El Victor- San Carlos area as well as the area immediately surrounding the Nuevo Mulatos claim.

Minera Real de Angles (MRA) acquired the Nuevo Mulatos claim in 1986 and carried out extensive exploration activities. MRA culminated their efforts with a

pre-feasibility study in 1990. As part of that study, MRA calculated a lognormally kriged mineral resource of 15.5 Mt grading 1.83 g/t Au at a cut-off grade of 1.0 g/t Au.

Placer Dome, Inc. (PDI) acquired full ownership of the claims from MRA in 1993. Subsequently, PDI and Kennecott entered into a joint venture agreement covering the Mulatos deposit and 34,000 ha of surrounding land. PDI functioned as the developer and potential operator with a 70% interest.

"Canmex", a subsidiary of Placer International Exploration, Inc., undertook exploration work on the property from 1993 to 1999.

In 2001 National Gold Corporation (National), through its Mexican subsidiary Minas de Oro Nacional, S.A. de C.V. (MON) (formerly O.N.C. de Mexico, S.A. de C.V.) acquired a 100% interest in the Salamandra Property from Minera San Augusto, S.A. de C.V. (MSA) a Placer subsidiary, for cash and a sliding scale Net Smelter Royalty in favor of MSA on the first two million ounces of gold. Alamos Minerals (AM) optioned 50% of the assets by being responsible for exploration and other expenditures.

In 2003, Alamos and National Gold merged to form Alamos Gold, Inc. (AGI). AGI, through its wholly owned Mexican subsidiary MON owns 100% interest in the Salamandra Property.

The Salamandra Property consists of the Mulatos deposit and eight satellite gold systems known as El Halcon, La Yaqui, Los Bajios, El Jaspe, La Dura, Cerro Pelon, El Victor/San Carlos and El Carricito.

The Mulatos deposit consists of the Estrella pit which is the subject of this Feasibility study. Immediately north of Estrella are the Viejo, Escondido, Gap and Victor Potential Pits.

8.2 TYPE, AMOUNT, QUALITY AND RESULTS OF EXPLORATION

The type, amount, quality and results of the exploration and/or development work undertaken by the owners or previous owners.

Within the area of the geologic model, 325 reverse circulation drill holes have been drilled to date. These include 121 holes by MRA, 66 holes by Kennecott and 147 holes by Placer Dome (PDI). Figure 2.16 shows the location of reverse circulation drill holes within the deposit area.

112 core holes have been drilled within the area of the geologic model. MRA drilled 11 core holes and PDI drilled 101 core holes. The 101 holes by PDI include 21 holes drilled for metallurgical test work, eight in 1994 and 13 in 1996. 17 of the PDI core holes were logged for geotechnical information. Figures 2.17,

2.18 and 2.19 show the location of all core drill holes within the deposit area, the location of geotechnical drill holes and the location of metallurgical drill holes.

Alamos has drilled 11 core holes from existing adits.

8.3   HISTORICAL MINERAL RESOURCES AND MINERAL RESERVES

Historical mineral resource and mineral reserve estimates, including the reliability of the historical estimates and whether the estimates are in accordance with the categories set out in sections 1.3 and 1.4 of the instrument.

Minera Real de Angeles (MRA) acquired the Nuevo Mulatos concession in 1986 and carried out extensive exploration activities, including the drilling of 121 reverse circulation holes of a total of 20,688 meters, 11 diamond core holes for a total of 1,928 meters, and driving 1,061 meters of exploration drift from which a bulk sample was taken. MRA performed a pre-feasibility study on the property in 1990. As part of that study, MRA calculated a lognormally kriged mineral resource of 15.5 Mt grading 1.83 g/t Au at a cut-off grade of 1.0 g/t Au. This resources estimate was not reviewed by M3 or its consultants.

Placer Dome on behalf of the Placer Dome/Kennecott Consortium and with the help of M3 completed a feasibility study in June 1997.

The Mineral Resource   83 MT @ 1.04 g/t Au @ 0.50 g/t cutoff
The Mineral Reserve   49.7 MT@ 1.23 g/t Au

Placer Dome updated this study in 1999/2000:

The Mineral Reserve   43.5 MT @ 1.587 g/t Au

Behre Dolbear, Vancouver B.C., reviewed the Placer work in January 2001 for National Gold and produced a qualifying report just before NI 43-101 was implemented. Many Behre Dolbear recommendations have been followed in the work for the 2004 Alamos Feasibility Study.

In September 2002, Pincock Allen and Holt of Denver, Colorado did a "preliminary assessment and scoping study for the Estrella (pit) development alternative for the Mulatos deposit". In it the Mina Vieja and Escondida, the Northern parts of the Placer Dome pit, were eliminated. The new smaller pit called "Estrella" was to operate at 17,500 MTPD.

The mineral Resource statement is "62.2 million tonnes measured and indicated at an average grade of 1.51 grams of gold per tonne".

M3 has just (April 2004) produced a Feasibility Study for Alamos Gold. The Estrella Pit chosen is similar to that in the P.A.H. report but a complete new

geological model was produced. The recommendations made by Behre Dolbear in their January 2001 report were followed.

The reserves were calculated to be 36.4 m tonnes averaging 1.637 g/t at varying cutoff grades, which is 0.8 g/t in the early years and declines to internal cutoff in the later years to enhance economics.

8.4 TYPE, AMOUNT, QUALITY AND RESULTS OF EXPLORATION

No accurate records are available for historic gold production from the Estrella pit area.

9 GEOLOGIC SETTING

General descriptions of regional and local geology were previously provided in the Minera Real de Angeles (MRA) pre-feasibility and the Placer Dome Inc. (PDI) feasibility study. However, since that time major revisions have been made to the understanding of local geology and deposit genesis. Some of the first attempts to unravel details of the local geologic sequence was made by consulting geologist J. I. Lyons and is described in his report "Geology of the Mulatos Prospect, Sonora" dated March 8, 1993. More detailed investigations were made by J.M. Staude during the course of a University of Arizona Ph.D. thesis in 1994. Placer Dome Exploration (PDX) geologists made extensive revisions during the geological modeling process for the 1997 feasibility study, followed by a new geologic model resulting from geologic and exploration work completed in late 1997 and 1998. The geologic model resulting from the 1997 and 1998 revisions is believed to be the most accurate and current, and is being used for the 2004 Estimate.

9.1 REGIONAL GEOLOGY

The Sierra Madre Occidental volcanic province is composed of two distinct packages of volcanic rocks, a lower early Oligocene (28 to 36 Ma.) group of predominantly andesitic volcanic rocks, and a younger Miocene (18 to 24 Ma.) group of bimodal rhyolitic to basaltic volcanic rocks. Paleozoic to Cretaceous-age sedimentary rocks and early Tertiary sediments are inferred to underlie the volcanic rocks at depth in the project area, but are not exposed at any location within the district. The sub-volcanic sedimentary package is well exposed along the road between the towns of Arivechi and Tarachi, however. Several large intrusive bodies of presumed mid-Tertiary age are present within the area, one near Matarachi and the other about 10 km north of Mulatos along the Rio Mulatos. The regional geology is shown in Figure 9.1.

9.2 LOCAL GEOLOGY

The Mulatos deposit is a large epithermal, high sulfidation or acid sulfate, disseminated gold deposit hosted within a mid-Tertiary dacitic to rhyodacitic

volcanic dome complex. Gold mineralization is closely associated with silicic and advanced argillic alteration occurring near the upper contact of a rhyodacite porphyry and in overlying dacite flows and volcaniclastic rocks. The deposit is located within a large area of hydrothermal alteration approximately three square kilometers in extent. Significant concealed mineralization was discovered below barren post-mineral rocks, however, suggesting the limits of the mineralized system may be greater than previously assumed.

9.2.1   Lithology

Volcanic rocks in the Mulatos project area consist of dacitic to rhyodacitic porphyry flows, volcaniclastic rocks, lithic to lithic crystal tuffs, and basalt flows. Significant changes have been made in the understanding of the stratigraphy of the volcanic succession hosting the Mulatos deposit since completion of the 1996 Placer Dome Inc. feasibility study, particularly involving the units in the northern portion of the deposit. Volcanic stratigraphy was previously assumed to be a normal stratigraphic sequence consisting of dacitic to rhyodacitic flows deposited in a volcanic dome complex overlain by post-mineral tuffs. A large intrusive hydrothermal breccia was believed to crosscut the dome complex rocks.

The breccia complex is now believed to be volcaniclastic material derived from partial erosion and destruction of the dome complex prior to deposition of the post-mineral volcanic units. The lower volcanic flow units are largely unchanged from the Placer Dome feasibility study interpretation and descriptions, but the upper units were found to be lateral equivalents of the same unit. One of the units previously thought to be post-mineral was also found to be one of the primary host rocks. The stratigraphy of the post-mineral volcanic rocks was also defined in an attempt to determine structural offset along faults, and predict depth to mineralization.

The lowest unit hosting mineralization in the deposit is a dacite porphyry (Tdf4), a composite unit of several lava flows and some volcanic sediment, with one or two minor pyroclastic intervals. It is overlain by a medium to coarse grained rhyodacite porphyry (Trf), one of two main host rocks for gold mineralization. The rhyodacite appears to be comprised of several distinct flows, with texture and mineralogy varying slightly between flows. It is largely intact in the southern portion of the deposit, but is thin to absent in the northern portion due to partial erosion and destruction of the dome complex. The rhyodacite porphyry is overlain by a another dacite porphyry unit (Tdf3) very similar in composition and texture to the lower dacite porphyry and only distinguishable on the basis of stratigraphic position. It is absent from the central deposit area due to erosion during subaerial exposure of the dome complex, but hosts significant mineralization in the southern portion of the deposit.

The dome complex appears to have been subject to a long period of erosion and possibly explosive destruction following deposition of the dacitic and rhyodacitic flow units. Host rocks for the northern portion of the deposit are comprised of fragmental volcaniclastic sedimentary rocks derived from erosion and partial destruction of earlier dome complex units (Tdf4, Trf, Tdf3). The fragmental rocks unconformably overlay the dacitic and rhyodacitic flows, with over 300m of relief on the basal unconformity surface.   Fragmental rocks are comprised of two predominant facies, a coarse-grained clast supported conglomeratic faces (Tpcg), and coarse to fine grained volcaniclastic sandstone (Tpqz). Gold mineralization is generally confined to the coarse grained facies. The fragmental rocks were previously interpreted as a breccia pipe, but textures within the breccia are frequently stratified, and no breccia roots are indicated by deep drill holes.

Table 9.1 is a summary of the main lithologic units present in the Mulatos deposit (youngest at the top of the table and oldest at the bottom).

Table 9.1
Main Lithologic Units

Age	Unit	Name	Description	Mineralization
	Tvu	Undefined volcanic rocks	Comprised of felsic pyroclastic rocks and basalt flows located west of the Mulatos fault and north of the Estrella deposit; overly the Escondida zone.	Unmineralized
	Tplt	Post Mineral Rhyolite tuff	Comprised of a rhyolite crystal tuff (rich in biotite) that unconformably overlies the altered and mineralized dome complex.	Unmineralized
	Tpcg, Tpqz, Ttq	Volcaniclastic Fragmental Unit	Comprised of fine to coarse grained volcaniclastic fragmental rocks derived from erosion and partial destruction of the earlier dome complex rocks (Tdf4, Trf, Tdf3). Maximum thickness of this unit is 300m, in the northern portion of the deposit.	Major host of gold mineralization in the northern portion of the Mulatos deposit.
	Tdf3	Dacite Porphyry Flow	Similar to Trf, distinction is quartz is rare to absent. Up to 90m thick in the southern portion and removed by an erosional event in the central and northern area.	Significant gold mineralization in the Estrella is in the basal portion of Tdf3
	Trf	Rhyodacite porphyry	Comprised of lava flows or dome-flow complexes, between 100 to 150m thick in the southern portion (Estrella) portion of the deposit. The unit includes an abundance of large di-pyramidal Quartz phenocrysts (<= 10mm) and is the only dome complex flow containing appreciable quartz.	Gold mineralization in the Estrella is predominately located along the upper contact of the Trf
	Tdf4	Dacite Porphyry	Lowest dacite flow, medium grained, Composite of several lava flows and some volcanic sediment with one or two minor pyroclastic intervals.	Minor gold mineralization
	Ts	Andesitic Tuff	Sequence of stratified andesitic lithic lapilli tuffs	Locally copper rich

OLDER ↓ (arrow indicating increasing age, leftmost column)

9.2.2 Structural Geology

Tilting and post-mineral normal faulting associated with late Tertiary extensional tectonics have affected both the mineralized flow dome complex and overlying volcanic rocks. Faults have been defined by surface and underground mapping, as well as sectional interpretation. Three dominant structural trends are present in the project area. Primary mineralized structures are northwest trending in the Estrella deposit area, with high-angle southwest dips. Mineralized structures north and south of the Estrella portion of the deposit are northeast-oriented, with high angle to near-vertical dips. Post-mineral faults are dominated by the high-angle, north-south trending Mulatos normal fault and associated parallel structures, which down-drop stratigraphy and mineralization to the west. Other significant post-mineral structures include the northeast trending Escondida fault, which offsets the Mina Vieja mineralization, and the northwest trending San Francisco fault. Post-mineral faults result in the Mulatos deposit being down-dropped to the north in stair-step fashion.

9.2.3 Alteration

All lithologic units of the dome complex are intensely altered. Alteration assemblages are typical of high sulfidation deposits, and show zonation patterns from distal propylitic alteration to illite to kaolinite to dickite/pyrophyllite to pervasive and vuggy silica alteration. Gold is predominantly hosted within silicic alteration. Two periods of alteration and perhaps gold mineralization are suggested, as the fragmental unit contains clasts of varying alteration assemblages, plus is overprinted by strong silicic and/or argillic alteration.

Gold mineralization controls are both structural and stratigraphic. A series of northwest trending, en echelon structural zones is the primary control of silica alteration and higher-grade gold concentrations in the Cerro Estrella portion of the deposit, with important secondary stratigraphic control along flow boundaries and within coarse grained volcaniclastic fragmental rocks.

The altered and mineralized units are locally overlain by a thick section of unaltered volcanic rocks that are believed to be post-mineral in nature. Although the basal unit is locally argillized, clay mineralogy is low temperature, and altered intervals are barren of gold concentrations. The post-mineral units form a relatively thick sequence on to the northeast of the Mulatos deposit, and extend from Puerto del Aire to the El Victor area. Maximum thickness is 200m, but in general range from 0-150 m.

9.2.4 Maps and Sections

The immediate deposit vicinity has been mapped during numerous mapping campaigns at a scale of 1:1000 and 1:2000. Generalized local project geology is depicted in three accompanying figures: lithology on Figure 9.2; alteration on Figure 9.3; structure on Figure 9.4.

Two east-west oriented cross sections (at 4200N and 4500N) showing the deposit geology, alteration, oxidation state and gold mineralization are included as Figures 9.5 and 9.6. A north-south, longitudinal section at 1850E is included as two figures: Figure 9.7 shows the lithology and alteration, and Figure 9.8 shows the oxidization and gold mineralization. Figure 9-9 shows the lithology, alteration, oxidization and gold mineralization distribution on the 1250 plan.

10   DEPOSIT TYPE

The Mulatos deposit is a large epithermal, high sulfidation or acid sulfate, disseminated gold deposit hosted within a mid-Tertiary dacitic to rhyodacitic volcanic dome complex. Gold mineralization is closely associated with silicic and advanced argillic alteration occurring near the upper contact of a rhyodacite porphyry and in overlying dacite flows and volcaniclastic rocks. Gold occurs in oxide, mixed oxide/sulfide, and sulfide ore types, with pyrite as the primary sulfide mineral. The deposit is amenable to cyanidation in all ore types, but gold extraction decreases with decreasing levels of oxidation.

11   MINERALIZATION

Gold mineralization within the Mulatos deposit occurs primarily within areas of pervasive silicic alteration of the volcanic host rocks. Gold also occurs within advanced argillic alteration assemblages proximal to silicic alteration, largely consisting of pyrophyllite or dickite dominant alteration. Quartz veins and quartz stockwork zones are rare to absent. Silicified rocks host approximately 80% of the contained gold within the deposit.

Staude describes three main mineralization assemblages. From oldest to youngest they are: 1) quartz + pyrite + pyrophyllite + gold; 2) quartz + pyrite + kaolinite + gold + enargite; and 3) kaolinite + barite + gold. Macroscopic minerals identified during core and reverse circulation chip logging at the project include: pyrite, enargite, chalcopyrite, molybdenite, gold, chalcocite, covellite, bornite, tetrahedrite/tennantite, marcasite, copper oxides, specularite, hematite, limonite, goethite, jarosite, pyrophyllite, kaolinite, alunite, montmorillonite, barite, chlorite, and epidote.

Free gold is commonly found in hematite-filled fractures. Gold also occurs in pyrite, as gold/silver tellurides, and possibly as a solid solution in some copper sulfide minerals.

Supergene oxidation and perhaps remobilization and secondary enrichment of gold have been ongoing since the post-mineral volcanic cover was removed.

12 EXPLORATION

Jesuit priests are reported to have first discovered Mulatos in 1635. The area saw considerable activity by various groups throughout the 1800's and 1900's, with the majority of historic production attributable to Greene Consolidated Gold and Silver Mining Company in the late 1800's. Gold production largely ceased during the Mexican Revolution in 1910.

Companies that have been interested in the district since 1960 include: Phillips Petroleum in 1962, Theodore A. Dodge in 1963, Cannon-Hicks Associates in 1972, Tormex Developers in 1973, Explomin S.A. de C.V. in 1974 (formerly part of Minera Real de Angeles), Homestake Mining Company in 1975, British Petroleum in 1982, Papanton Minas (subsidiary of Placer Amex) in 1984, and Kennecott Minerals in 1990.

Kennecott conducted extensive exploration activities on ground surrounding the Nuevo Mulatos and Tequila claims from 1991 through 1993. Their efforts focused on the El Victor/San Carlos area as well as the area immediately surrounding the Nuevo Mulatos claim.

Minera Real de Angeles (MRA) acquired the Nuevo Mulatos claim in 1986 and carried out extensive exploration activity thereafter, culminating their efforts with a pre-feasibility study in 1990.

Placer Dome, Inc. (PDI) acquired full ownership of the claims from MRA in 1993. Subsequently, PDI and Kennecott reached a 70/30 joint venture agreement, covering the Mulatos deposit and 35,000 hectares of surrounding land, with PDI as operator. Exploration work was conducted by Placer Dome Exploration (PDX), a subsidiary of PDI, and Empressa Minera Can-Mex, S.A. de C.V. (Can-Mex), a subsidiary of PDX. PDX conducted extensive exploration in the deposit area and reconnaissance exploration with limited drilling on the remainder of the land position from 1993 through 1996, which resulted in a feasibility study and a positive mine construction decision in 1997. Additional exploration work undertaken in late 1997 and 1998 resulted in the discovery of the Escondida deposit to the northeast of Mulatos, and additional mineralization between Escondida and the El Victor areas. Placer Dome suspended all exploration and development activities in the district in the second quarter of 1999.

In 2001 National Gold Corporation (National), through its Mexican subsidiary Minas de Oro Nacional, S.A. de C.V. (MON) (formerly O.N.C. de Mexico, S.A. de C.V.) acquired a 100 % interest in the Salamandra Property from Minera San Augusto, S.A. de C.V. (MSA), a Placer subsidiary, for cash and a 2% Net Smelter Royalty in favor of MSA on the first two million ounces of gold. The Salamandra Property is comprised of the Mulatos deposit, the Salamandra concession, and numerous adjacent concessions.

Alamos Minerals (AM) optioned 50% of the assets by being responsible for exploration and other expenditures.

In 2003 AM and National merged to form Alamos Gold Inc, (AGI). AGI, through its wholly owned Mexican subsidiaries MON and Minera Beinvienidos, S.A. de C.V. (MB) owns 100% interest in the Salamandra Property.

The Salamandra Property consists of the Mulatos deposit and eight satellite gold systems known as El Halcon, La Yaqui, Los Bajios, El Jaspe, Cerro Pelon, El Victor/San Carlos, La Dura, and El Carricito. Numerous smaller areas of hydrothermal alteration similar to those known to host gold mineralization are also present on the property.

AGI drilled 13 underground core holes in the Estrella area in 2003 as part of its continued exploration activities on the property. The collection of geologic information continues in the Mulatos deposit and many of the satellite gold systems.

The resource model area of Mulatos has been explored using surface and underground geologic mapping, core and reverse circulation drilling, channel sampling and assaying of bulk samples taken during underground excavation. Table 12.1 summarizes the drilling information collected through the end of 2003.

Table 12.1
Drilling Summary

Company	Reverse Circulation Holes	Core Holes (Surface and Underground)			Other Drilling & Sampling
		Assay & Logged	Metallurgical Sample	Geotechnical Logged	
Minera Real de Angeles (MRA)	119	11	0	0	0
Kennecott	69	0	0	0	0
Placer Dome, Inc. (PDI)	172	110	21	17	61
Alamos Gold Inc (AGI)	0	13	0	0	0
Total	360	134	21	17	61

13 DRILLING

The Mulatos deposit has been drilled using both core and reverse circulation techniques. Table 12.1 summarized the drilling by type and company. Figures 13.1 and 13.2 show the drill hole traces within the resource model area. As mentioned previously, 360 reverse circulation and 134 core holes are included in the drilling database.

Data collection began with the geologists logging the drill holes on site. Reverse circulation holes were logged from chip trays containing representative samples collected

from each sample interval. Geologists logged onto paper sheets. Logging included the notation of various aspects of lithology, alteration, and mineralization. Core drill holes were also logged onto paper sheets. Core hole logging was more detailed and included core recovery, RQD, lithology, structure, alteration, and mineralization.

Drill hole geologic data from MRA's project is available as both basic graphic and descriptive logs, the majority of which have been translated into the Geology format. Kennecott logs are available as paper copies depicting graphic and descriptive information, and as digital Geolog files. The majority of Kennecott and MRA reverse circulation sample chip trays are still available and are stored at the project site. They have been re-logged to conform to the currently understood stratigraphy and mineralization.

Prior to 1996, information from drill hole logs was compiled and entered into the Paradox database, then transformed into Geolog type files. In 1996, drill hole geology and other information were input directly to Geolog type files. As part of the re-modeling exercise, all holes were re-logged for rock types, alteration, and oxidation in the spring of 1996.

Thirteen underground core holes were drilled by AGI from the Nopal, Cantil, and Nopalito adits during the fall of 2003. All core was logged on site with paper logs and entered as digital Geolog files. The drill core was photographed using a digital camera and then cut and sampled on site. A one-half split for all core is archived on site.

Additional information collected from the drilling included specific gravity samples and geotechnical logging. This work is briefly summarized below.

Density determinations were completed by Placer Dome on approximately 2,800 core samples. These samples were collected from a variety of rock, alteration, and oxidation types. According to Placer documents, the submersion, "quick submersion", and plastic wrap determination methods were used. The submersion methods were used for competent core samples. With this method the initial core samples were weighed in air (natural weight), weighed in water, dried for 24 hours at 100º C, weighed in air again, and then weighed in water. The bulk density was calculated by the following formula:

Bulk Density = weight in air / (weight in air - weight in water)

For clay altered samples and vuggy or fractured samples Placer used the plastic wrap method. The method is very similar to the immersion method only the initial wet sample was first weighed in air, then tightly wrapped in cellophane and weighed in water, then dried for 24 hours, then the dried sample was re-wrapped in cellophane to protect it from decomposing or taking on water in open vugs and weighed in water. The bulk density calculation is the same as the one shown above. Placer also calculated moisture content from the samples using the following formula:

Moisture Content (%) = ((Natural Weight – Dry Weight) / Natural Weight) * 100

The specific gravity values were loaded to the drill hole database so that statistics could be reviewed by various geological types. The number of samples and average specific gravity (SG) were calculated for various combinations of lithology, alteration, and oxidation. By examining these data for various geologic combinations it became apparent that unique SG's were required. Table 13.1 tabulates the SG values that were put into the block model for certain material types.

Table 13.1
Density Values

Material Type	SG (tonnes/m^3)
Overburden	2.24
Oxide	2.30
Post Mineral Volcanics	2.30
Rhyolite Flow - Mix-1/Mix-2	2.44
Rhyolite Flow - Sulfide	2.53
DF4 Sulfide	2.61
Mix-1 & Sulfide	2.50
Mix-2 & Sulfide	2.42
Arg-1 Vuggy Silica Sulfide	2.48
Arg-2 Silicified Sulfide	2.53
Default	2.50

Geotechnical data was collected under the guidance of Golder Associates Inc. during the 1994 and 1996 core drilling program. Geotechnical drill holes were treated the same as all other core holes with respect to geological logging and sampling. Additional geotechnical data as prescribed by Golder Associates was also collected. Data that was described and recorded for these holes included fracture frequency, fracture angles, descriptions of fracture mode of occurrence and alteration, rock resistance to breakage, and point load test data. This data was compiled into the Paradox/Geolog databases and then verified by FSSI/project staff.

In 1996, three core holes were specifically drilled to obtain geotechnical and structural information. These three core holes were oriented in space using the clay imprint method. True dip/azimuth of structures were measured. Golder Associates utilized the data for an independent evaluation of slope stability and selection of pit wall slope angles.

14 SAMPLING METHOD AND APPROACH

The drill holes and other sample collection have been done by four different exploration companies during the recent history of the Mulatos project, utilizing at least four different drilling contractors. Summarized below is the current understanding of the sampling protocol used for each company's drilling and sample collection of information used to generate the resource and reserve estimate contained in this report.

14.1 MINERA REAL DE ANGELES

The following is a brief synopsis of MRA's sample collection techniques as described in the MRA pre-feasibility report. A copy of the *Sampling and Assaying* section of the 1990 pre-feasibility report is presented in Appendix II to the MSA/Placer 1997 Feasibility Study Report.

Reverse circulation drilling was accomplished using a Drill Systems MPD-1000 truck-mounted rig. Samples were collected on 3 m intervals. In most cases holes were drilled dry down to a depth of 120 m. Below 120 m, water was injected to obtain a wet slurry sample. The entire 3 m sample weighting approximately 80 kg was collected in the cyclone on the drill. It was passed directly from the cyclone on the drill into a Jones type splitter. The sample volume was reduced by multiple passes through the splitter to ultimately obtain two samples weighing approximately 10 kg each. One sample was sent for assay analysis, while the second sample was retained and stored as an archive sample.

Core drill holes were sampled on 3 meter intervals. In the early stages of MRA's core drilling program, the entire drill core was bagged and shipped for assay analysis. Later, the core was split; half was sent for assay, and the other half was retained for archive storage.

14.2 KENNECOTT

Other than sample length, specific techniques, procedures, and methodologies used by Kennecott are unknown. The reverse circulation cuttings from holes drilled by Kennecott were sampled on 5 ft (1.52 m) intervals.

14.3 PLACER DOME INC

14.3.1 Drilling Techniques

Two different drilling contractors were used for reverse circulation drilling by PDI during 1993-1994: Dateline Drilling of Landusky, Montana; and Drilling Services Inc., of Hermosillo, Sonora, Mexico. Both companies are U.S. based and used American drillers.

Dateline used a track-mounted type reverse circulation rig. This unit operated with a 900 cfm/350 psi compressor. Drill rods were 10 ft in length, and hole diameter was 4.5 inches. At various times Dateline had difficulties obtaining an adequate sample recovery volume. Also, they were unable to drill many of the strongly silicified zones, and geologists sometimes had to stop the hole short of planned depth. Ultimately, Dateline's contract was terminated in March 1994 due to problems with recovery and an inability to drill strongly silicified zones.

Drilling Services Inc. used a Cyclone Model TH-100A truck-mounted rig utilizing a 750 cfm/250 psi compressor. Drill rods were 20 ft in length and hole diameter was 5.5 inches. Drilling Services was usually able to recover samples of adequate volume. They did have difficulty drilling some of the strongly silicified zones, particularly in the Buena Vista breccia. Several holes were stopped short of planned depths because of an inability to penetrate these zones.

Major Drilling Inc. and Layne de Mexico were contracted for diamond core drilling. Holes were collared with HQ diameter core, and, only if necessary due to hole conditions, were they reduced to NQ diameter.

Three drilling companies were involved in the 1996 drilling program. Reverse circulation holes were completed by Layne of Mexico (formerly Drilling Services Inc.) and Boytec Sondajes de Mexico. Both companies used Cyclone Model TH-100A truck-mounted rigs utilizing a 750 cfm-250 psi compressor. Core drilling was contracted to Major Drilling Inc. Holes were collared with HQ diameter core and were reduced to NQ diameter if necessary due to hole conditions.

Layne de Mexico conducted both core and reverse circulation drilling during the 1998 exploration programs. Core was drilled with HQ diameter size.

14.3.2   Sample Collection – Reverse Circulation Drilling (RC)

Reverse circulation cuttings from holes drilled by PDI were sampled on 5 ft (1.52 m) intervals and handled using the following protocol:

- In almost all instances holes were naturally dry, but water was injected during drilling to obtain a wet slurry.

- The entire 5 ft sample was collected in the cyclone on the drill.

- The entire wet sample was passed directly from the cyclone on the drill through a rotary splitter reducing volume to obtain a sample of approximately 10 to 15 kg. Sample cuttings and water passed directly from the rotary splitter into 5 gallon buckets.

- Afterwards, polymer was added, the sample was set aside, and allowed to settle for approximately 2 days. Clear water was then decanted. The remaining sample cuttings were bagged and shipped to Hermosillo for analysis.

The primary laboratory used for assaying of PDI reverse circulation samples during 1993 and 1994 was SGS/XRAL, in Hermosillo. Check assays during this period were performed by Bondar Clegg in Vancouver, British Columbia, and Rocky Mountain Geochemical in Salt Lake City,

Utah. During 1996, the primary laboratory used for assaying was Barringer Laboratory in Reno, Nevada, with check assays sent to the PDI Research Center in Vancouver, British Columbia.

14.3.3 Sample Collection – Core Drilling

Core drilled by PDI was logged and sampled at site. After completion of geological logging, measurement of core recovery, and collection of RQD information, geologists defined and labeled the intervals to be sampled. Core holes were consistently sampled on 5 ft (1.52 m) intervals with the exception of tops and bottoms of holes and intervals adjacent to missing samples. Skeleton core samples approximately 4 cm long were collected and saved for each 10 ft (3.05 m) interval down the hole. Skeleton core is stored at the project's core storage facility. Most of the core boxes were photographed prior to sampling; pictures are stored in Hermosillo, with copies available at the project site.

Core drilled prior to 1997 was not split. The entire core, minus skeleton core samples, was bagged by sample interval and shipped to the SGS/XRAL Lab in Hermosillo for analysis. All core was cut on site during the 1997 and 1998 exploration programs, with one half split used for sampling, and the other split saved on site. Prior to April 1994, check assays were performed by Bondar Clegg laboratories in Vancouver, British Columbia. Beginning in April 1994, Rocky Mountain Geochemical, in Salt Lake City, Utah also performed check assays.

Core logging and sampling procedures in1996 were similar to those used in 1994, except that sampling intervals were based on geological contacts (rock types, alteration, and/or oxidation states), with 5 foot intervals as a standard sample length in rock types presenting similar characteristics. The entire core was bagged and shipped to the PDI Research Center in Vancouver, British Columbia, for sample preparation, analysis, and metallurgical testing. Check assays and QA/QC procedures were performed internally by the PDI Research Center.

Core logging and sampling procedures in 1997 and 1998 also involved sampling to geologic contacts, with five foot (1.5m) sample intervals being the standard length. A one-half split was sent to Barringer Laboratory in Reno, Nevada after the core was cut with a diamond saw.

14.3.4 Sample Collection – Metallurgical Drill Holes

Metallurgical drill holes were processed at site in a similar manner to other core drill holes. Geologic logging, sample interval definition, measurement of core recovery and collection of RQD information collection was completed by geologists. All samples were bagged and

shipped to Hermosillo. Samples were then loaded into 55-gallon barrels and shipped to the PDI Research Center, in Vancouver, British Columbia.

Core samples from the 1994 campaign were sawed in half. One half was crushed and a split was analyzed for gold, silver, and 26 other elements. The rejects and the other half-core were then utilized for metallurgical test work. In 1996 the samples were first crushed to 1/2 inch, and then spilt using a Jones Riffle splitter in two halves. The first half was further reduced to minus 10 mesh and assayed for gold, silver, and copper. The second half was used for metallurgical test.

14.3.5 Sample Collection – Underground Channel Samples.

Metallurgical samples were collected from three underground audits: El Nopal, El Cantil, and Buena Vista II. Channel samples were cut from the rib of the workings using pneumatic equipment. All sample intervals were 5 ft (1.52 m) in length. The work was contracted to COMYCSA, of Hermosillo, and was supervised by Can-Mex geologists. Sample intervals were described by geologists using a format similar to the drill hole logging techniques. Samples were bagged and shipped to Hermosillo, loaded into 55 gallon drums, and shipped to the PDI Research Center, in Vancouver, British Columbia. The sampling protocol for the 1994 channel sample program is presented in Appendix III to the MSA/Placer 1997 Feasibility Study Report.

In 1996 additional channel samples were collected from the El Nopal, Nopalito, Cantil, Buena Vista I, Buena Vista II, San Francisco, El Salto, Escondida, and Hule underground workings. Channel samples were cut from the rib of the workings using pneumatic equipment. All sample intervals were 1.5 m in length. The work was contracted to Construcciones Tres Hermanos of Sahuaripa, Sonora, and was supervised by Can-Mex geologists. Sample intervals were merged with the Geolog files created from the 1996 underground re-mapping exercise. Samples were bagged and shipped to Barringer in Hermosillo for sample preparation. Each entire sample (20-40 kg) was crushed to minus 10 mesh. A 1-kg split was fine crushed to minus 150 mesh before assay on a 30-g aliquot was performed (Au, Ag and Cu). Assaying was performed by Barringer Laboratory in Reno, Nevada.

14.4 ALAMOS GOLD INC.

14.4.1 Drilling Techniques

Underground core drilling was conducted in the fall of 2003 by Layne de Mexico, located in Hermosillo, Sonora. A Hagby Electric Short Feed Frame underground drill was used, with NQ size core. No hole reductions

were necessary. A combination of Canadian and Mexican national drillers were used.

14.4.2 Sample Collection – Core Drilling

Core was logged and sampled at site. After completion of geological logging, measurement of core recovery, and collection of RQD information, geologists defined and labeled the intervals to be sampled. Core holes were consistently sampled on 5 ft (1.52 m) intervals with the exception of tops and bottoms of holes and intervals adjacent to missing samples. All core was cut on site with a diamond saw, with one half split used for sampling, and the other split saved on site. All core was digitally photographed prior to sampling.

The split core was bagged by sample interval and shipped to the BSI Inspectorate sample prep lab in Durango, Mexico, and with pulps being sent to Reno, Nevada for analysis. AGI QA/QC protocol included the submission of standards and blanks every 20th sample, and utilized the same standards and procedures as used for the Placer Dome Inc. 1996-1998 drilling programs.

15   SAMPLE PREPARATION, ANALYSES AND SECURITY

The methods used to collect the samples on the property are discussed in Section 14. This section presents the assay laboratory protocol.

Laboratory protocols and analytical methods used by SGS/XRAL (Hermosillo) and Barringer (Reno) Laboratories are outlined below.

In March and April 1994, a review of SGS/XRAL laboratory procedures was undertaken by various Placer Dome people as well as a consulting chemist. Based upon recommendations from those people, SGS/XRAL laboratory procedures were changed in May 1994. The following sections describe the procedures prior to and after May 1994, as well as 1996 Barringer Laboratory procedures.

Prior to May 1994, SGS/XRAL prepared samples according to the following protocol:

- Samples were sorted, and then dried at 110°C.

- The entire sample was jaw crushed to minus 1/4 inch.

- The resulting sample was riffle split until a 1-kg sample was retained. The remaining sample was saved as a coarse reject.

- The 1-kg sample was pulverized to minus 200 mesh using a mixer-mill pulverizing/homogenizing bowl and puck system. This sample was assayed as described below under Analytical Methods.

- Every tenth 1-kg sample was riffle split to form a second pulp, which was assayed as a duplicate assay.

Beginning in May 1994, SGS/XRAL prepared samples in the following fashion:

- Samples were sorted, and then dried at 110°C.

- Samples were then jaw crushed to minus 1/4 inch. In the case of core, samples were further disc ground to minus 10 mesh.

- The resulting sample was riffle split and a 1.5-kg sample was retained. The remainder of the sample was saved as a coarse reject.

- The 1.5-kg sample was pulverized to minus 200 mesh.

- This 1.5-kg pulp sample was riffle split. One half of the sample was bagged and used for SGS/XRAL assays. The second half was riffle split four ways and then bagged to form four separate pulp samples. The four extra samples were either stored as spare duplicates at the Can-Mex warehouse facility or used for check assays.

All 1996 reverse circulation and underground channel samples were sent to Barringer Laboratories. Sample preparation of channel samples sent to Barringer is described above and will not be repeated here.

- Samples were sorted, and then thoroughly dried at 110°C.

- Samples were then crushed using combination of jaw and roll mill to 70% passing minus 40 mesh.

- The resulting sample was riffle split and a 0.3-kg sample was retained. The remainder of the sample was saved as a coarse reject.

- The 0.3-kg sample was pulverized to minus 150 mesh with a ring and puck pulverizer. Clean sand was employed between each sample to clean the pulverizer.

- This 0.3-kg pulp sample was sent to Barringer Laboratory in Reno, Nevada for assaying. The rejects were return to Can-Mex and stored as spare samples at the Can-Mex warehouse facility.

As part of the sulfide sulfur modeling program, a total of 6,068 sulfur analyses were performed. Samples consisted of pulp composites from contiguous sample intervals (drill holes or channel samples). Original pulps were sent to Barringer for compositing and assaying. The compositing procedures were as follows:

- Individual original pulps were first homogenized by rolling;

- Approximately 10 g of material was split from each individual pulp sample;

- Four different interval splits forming the composite were mixed together and homogenized; and

- An aliquot was collected from the composite sample for assaying.

SGS/XRAL performed gold fire assays with an atomic absorption finish for all samples. For most samples a 50-g aliquot was used. Prior to May 1994, for all samples with a resulting assay equal to or greater than 10 g/t Au, a second aliquot of pulp was taken to produce a fire assay with a gravimetric finish. Beginning in May 1994, the threshold for a re-assay with gravimetric finish was changed to 5.0 g/t Au.

Samples with gold assays greater than 0.50 ppm were assayed for cyanide soluble gold and copper (CNSAu and CNSCu) using the following methodology:

- Twenty grams of sample pulp was leached with 40 ml of 2.0% NaCN solution;

- The solution/slurry was shaken manually every 20 minutes during a 2 hour leach period;

- pH of the solution was monitored and adjusted to remain within the range of 9.5 to 10.5;Gold concentration in the cyanide solution was determined by atomic absorption spectroscopy with a detection limit of 0.05 ppm; and

- Copper concentration was determined by atomic absorption spectroscopy of the same solution with a detection limit of 5 ppm.

In 1995 an extra set of 1403 samples were sent to Min-En to complete the CNSAu and CNSCu database. The procedures were identical to SGS except for the shaking occurred continuous during the 2 hour leach period.

Total copper and silver analyses were performed by SGS/XRAL using perchloric acid and nitric acid digestion of a 0.2-g sample. The acid solution was diluted with de-ionized water and mixed. The concentration of metal ions was determined by atomic absorption spectroscopy. Copper and silver were determined using an air acetylene flame.

All 1996 samples were assayed by Barringer. Fire assays with an atomic absorption finish was the standard assaying procedure for gold and silver. For all samples a one assay-ton aliquot was used. All samples with a resulting assay equal to or greater than 3 g/t Au were re-assayed using a fire assay with a gravimetric finish. Barringer carried a systematic QA/QC procedure on all batches of samples sent to their Reno, Nevada laboratories. Every tenth sample was repeated and for every 20 samples run, a standard or blank was also analyzed. Total QA/QC samples represented approximately 15% of all samples assayed.

Total copper analyses were performed by Barringer using multi-acid digestion of 1 g of pulp sample. The acid solution was diluted with de-ionized water and mixed. The concentration of metal ions was determined by atomic absorption spectroscopy.

Sulfide sulfur analyses were performed by Barringer using an induction type furnace made by LECO. Two analyses are conducted to get the three results of total, sulfide, and sulfate sulfur analyses. Sulfur is first volatilized at 3000°F with Fe and W compounds used as accelerator. The volatilized sulfur is carried by a stream of O_2 into an IR detector to measure the amount of sulfur by voltage reading. Calibration is done using a standard between every sample string (usually 20 samples). The second analysis starts by roasting the sample at 1400°F to burn off the sulfide sulfur leaving only the sulfate sulfur. The

roasted sample is again put in the LECO furnace. The new result is subtracted from the first to get the sulfide sulfur. Approximately 10% of the samples run though the LECO represented QA/QC samples.

16   DATA VERIFICATION

A study of check assay data was completed by FSS International Consultants Inc. (FSSI). The following is summarized from that study.

Prior to May 1994, 10% of the samples were sent to Bondar Clegg for check assays. Beginning in May 1994, 20% of the samples were sent to Bondar Clegg and to Rocky Mountain Geochemical for check assays. A total of 2,949 pulp samples were sent to Bondar Clegg and a total of 2,147 pulp samples were sent to Rocky Mountain Geochemical.

In July 1994, FSSI performed a preliminary check assay study making recommendations for further work. FSSI's study revealed that the SGS/XRAL assays made prior to May 1994 were 5 to 10% higher than the Bondar Clegg check assays. These assays were called the "Phase 1" assays. The study also showed that samples in the range below 0.5 g/t Au were as much as 20% higher than the Bondar Clegg check assays. FSSI also determined that the SGS/XRAL assays from May 1994 onward agreed favorably with check assays from Bondar Clegg and Rocky Mountain Geochemical laboratories. FSSI also pointed out that there were insufficient gravimetric check assays for higher grade samples to make good statistical comparisons. FSSI recommended that all samples analyzed gravimetrically by SGS/XRAL prior to May 1994 be sent for re-assay.

As a result of FSSI's recommendations, Can-Mex sent 790 sample pulps (all samples from SGS/XRAL with assays greater than or equal to 4.0 g/t) for check assaying by Bondar Clegg and Rocky Mountain Geochemical. In October 1994, FSSI reviewed these higher grade check assay results statistically and determined that there were no significant discrepancies among the three laboratories for samples in this grade range.

In 1996, further studies were completed on the Phase 1 assays and a major re-assay program was completed

16.1   MRA CHECK ASSAYS

Check assays for the MRA assays were done at four assay labs in 1988; Comision de Fomento Minero (CFM) in Hermosillo, Sonora; Skyline Labs in Tucson, Arizona; Cortez Mines in Nevada; and the Placer Dome Research Center in Vancouver, British Columbia. A summary of the 1988 laboratory results for the MRA check assays is presented in Table 16.1.

Table 16.1
Comparison of 1988 MRA Check Assay Results

Laboratory	Number of Assays	Correl. Coef.	Regression Equation
CFM - CFM	84	0.98	Cfm=(0.96*cfm)+0.02
CFM - Skyline	105	0.79	Sky=(0.64*cfm)+0.53
CFM - Cortez1	108	0.82	Ctz1=(0.62*cfm)+0.49
CFM - Cortez2	100	0.81	Ctz2=(0.62*cfm)+0.51
CFM - Placer	104	0.81	Pdi=(0.69*cfm)+0.48

Although the CFM check assays appear to be acceptable, the assays from the other labs show a systematic bias of 15% to 20% lower than the original CFM assay. Check assay plots for the 1988 check assays are presented in Appendix IV of the MSA/Placer 1997 Feasibility Study Report. The assay protocol for the 1988 check assays is not known.

In 1989, an additional 306 samples were sent to the PDI Research Center for check analyses. Although the regression analysis performed in 1989 showed that "a high degree of confidence" could be placed in the assays, the relative difference plot in Figure 16.1 shows a systematic bias between 5 and 10% for the data corresponding to the inner quartile range. It should be noted, however, that the PDI assays consisted of two fire assays of the minus 150 mesh fraction. The average of the two was used. Thus, the PDI assays do not include the plus 150 mesh gold fraction. Studying the MRA lab data sheets indicated that an average of 8.0% of the MRA gold assay came from the plus 150 mesh fraction (the assay protocol is discussed further below). Taken in this context, it is probable that the 1989 check assays done by Placer Dome are biased low by only 2%, a level that is acceptable.

16.2   KENNECOTT CHECK ASSAYS

Check and duplicate assay data for some of the Kennecott drill holes was reviewed. These data came only from the work completed by Kennecott in 1993. Earlier check assay data was not available. A total of 90 check assays and 401 duplicate assays comprise the data. The original Kennecott assays were completed at Rocky Mountain labs in Salt Lake City, Utah, and the check assays were done at Skyline Labs in Tucson, Arizona.

The check assays show good agreement with a correlation coefficient of 0.99. The mean and median of the check assays are -3.5% and -2.1% of the original assay, respectively, but the relative difference plot does not show any systematic bias. The check assay plots are presented in Figure 16.2. Duplicate assays also show good agreement with a correlation coefficient of 0.92 and percentage differences at the mean and median of 1.87% and 4.0%. The relative difference plot for the duplicate assays shows local high grade bias to the duplicates (Figure

16.3). Although there appears to be a slight bias for the duplicate assays, the check assays compare well. Thus, the 1993 Kennecott data is of acceptable quality.

16.3   1996 DRILLING QUALITY CONTROL AND CHECK ASSAYS

During the 1996 northern extension drilling program, blind standard and blank samples prepared by the project staff were included with each sample shipment to Barringer Labs. If the standard assay was higher than one standard deviation of the expected value of the standard, the sample batch was sent for re-assay. Thirteen sample batches required new assaying. A memo outlining the QA/QC program is presented in Appendix V of the MSA/Placer 1997 Feasibility Study Report.

In addition to the standards and blanks, 213 pulps from the new drilling (approximately 5% of the samples) were sent to the Placer Research Center for check assays.   The assays compare well with a correlation coefficient of 0.99 and percent difference at the mean and median of 2.2% and 2.0%, respectively. These statistics and the relative difference plot show that the check assays of Placer Dome Research Center are systematically higher grade than the original Barringer assay by approximately 2% (Figure 16.4). This difference is insignificant and the assays from the 1996 drilling should be considered good quality.

16.4   ALAMOS GOLD 2003 DRILLING

Alamos Gold (AGI) drilled 13 underground holes in 2003. A discussion of the collection of the samples, security, sample preparation and check assays is presented here.

Core was collected daily from the underground drill site by the site geologist and brought to the secure core logging and storage area. All core storage facilities are locked when not being used by geologic personnel. Core was logged on site, using paper logs with later entry into digital Geolog format. Logging included descriptions of lithology, alteration, and oxidation type as well as core recovery, RQD, and fracture orientation. After completion of geological logging, geologists defined and labeled the intervals to be sampled, along with marking cut lines on the core. Core holes were consistently sampled on 5 ft (1.52 m) intervals with the exception of tops and bottoms of holes and intervals adjacent to missing samples. All core was digitally photographed prior to sampling, and then cut on site with a diamond saw. One half split was used for the sample, and the other split returned to the box and archived on site. Plastic sample bags were sealed after filling, and then placed in large sealed plastic bags for transport to Hermosillo. Samples awaiting shipment were kept in a locked facility.

Core samples were driven to Hermosillo by company personnel and shipped to the BSI Inspectorate sample prep lab in Durango, Mexico. The BSI Durango lab

crushed, split, and pulverized the sample prior to sending a representative pulp to their Reno, Nevada facilities. AGI QA/QC protocol included the submission of standards and blanks every 20th sample, and utilized the same standards and procedures as used for the Placer Dome Inc. 1996-1998 drilling programs. Assay results were received electronically and by certified hard copy assay certificate. Rejects are currently in the BSI Durango facility, whereas the split core is stored on site in a secure facility.

16.5   CHECKS BY AN INDEPENDENT CONSULTANT

The resource model for this report was constructed by an independent consultant, Resource Modeling Inc. The discussion of the drill data transfer and additional checking completed by RMI are included in Section 19.

17   ADJACENT PROPERTIES

The Salamandra Property (controlled by AGI) consists of the Mulatos deposit and eight satellite gold systems known as El Halcon, La Yaqui, Los Bajios, El Jaspe, Cerro Pelon, El Victor/San Carlos, La Dura, and El Carricito.

The Mulatos deposit consists of the Estrella pit for which a minable reserve estimate is completed and included in Section 19. Immediately north and northeast of Estrella are the Mina Vieja, Escondida, Gap and El Victor deposits. These deposits are in various stages of exploration including drilling and future work is intended to delineate both resources and reserves in these deposits.

Exploration on the satellite gold systems ranges from early stages of mapping and sampling to drill target selection.

Mineral rights for all claims on and around the Mulatos orebody are controlled by AGI. A majority of the Mulatos orebody is positioned on the Nuevo Mulatos claim; however, a number of other claims surround or are in close proximity to the Nuevo Mulatos claim and represent exploration potential. AGI controls the Salamandra claim block and several other large concessions, which are located mostly to the west of the Mulatos deposit. A total of 19,634.34 hectares of mineral concessions are controlled by Alamos.

18   MINERAL PROCESSING AND METALLURGICAL TESTING

18.1   RECOVERABILITY

Information concerning results of all test and operating results relating to the recoverability of the valuable component or commodity and amenability of the mineralization to proposed processing methods.

A weighted average gold extraction for all ore types has been estimated at 72.9%, (Up from 66% reported in the MSA/Placer 2000 Information Package).

This increase is mainly due to the elimination of the Mina Vieja and North Estrella mineralized zone from consideration, crush size reduced to P80 of 3/8" from 1/2" and additional sulfide ore column leach tests. Extraction formulas for the different ore types were changed from the Placer Dome Feasibility to yield the following extractions:

Oxide	96.4% (was 90.0%)
Mixed and Fracture < 1.6%S	82.9% (was 75.0%)
Sulfide and Fracture > 1.6%S	67.6% (was 56.2%)
Weighted Average	72.9% (was 66.0%)

This change is due to investigations by RDi and includes elimination of the 0.95 scale up factor used by PDI, the higher gold recovery in the south (Estrella) pit area and the crush size reduced to P80 of 3/8 inch from ½ inch.

Because many of the Placer column leach testes were terminated early, RDi believes the 0.95 scale factor is not appropriate.

18.2 METALLURGY

18.2.1 Introduction

In 2002 Minas de Oro National, S.A. de C.V. (MON) contracted Resource Development, Inc. (RDi) to review the metallurgical testwork undertaken by Placer Dome Division Research Center (PDDRC)(Appendix 3.1). The study indicated that the deportation of gold in the sulfide ore was unknown and the poor extraction of gold could be due to a combination of size dependence and solid solution of gold in pyrite. Based on these findings, MON decided to undertake additional test work at RDi, Polysius Research Center (Polysius) and Metcon Research Inc. (Metcon). The primary objectives of the additional testing were: (a) to determine by diagnostic testing the deportation of gold in sulfide ore; (b) to evaluate high pressure grinding roll (HPGR) comminution to see if ore fractures along grain boundaries enhanced gold recovery; and (c) to column test of finer crush sulfide ore. The testwork consisted of HPGR crushing tests, gravity tests and bottle roll and column leach tests on sulfide-bearing channel samples from the deposit.

The metallurgical review of PDDRC metallurgical test data also indicated that the gold in the sulfide ore from the south Estrella zone more readily liberated during crushing as compared to the gold in the ore from the north zone. This resulted in lower gold extraction from the north zone sulfides (Report No. 6, August 1996).

The south Estrella zone is the focus of this Estimation.

18.2.2 Recommendation for Gold Recovery

18.2.2.1 Gold Recovery Equations

Placer Dome Models

Placer Dome Inc. (PDI) developed models to project the recovery for each ore type. Metallurgical column test results from three test programs were used: Report No. 4 consisting of five composites; Report No. 5 (Phase III) consisting of eight composites, and Report No. 8 (Phase V) consisting of forty two composites. These reports are noted in section 3.16.2, Placer 1997 References. Data used to create model equations is presented in Table 3.1. Gold extraction for columns and bottle-roll tests, as well as those projected by the extraction equations are also shown.

Test results were grouped by oxidation type and, in the case of the fracture oxidation, by total sulfur content. During the data analyses it was noted that those fracture oxidation composites having a total sulfur content greater than 1.6% behaved similarly to the sulfide composites. The fracture composites containing less than 1.6% total sulfur behaved similarly to the mixed oxidation composites. Hence, the data was grouped into four categories: oxide, mixed and fracture oxidation less than 1.6% total sulfur; sulfide and fracture oxidation greater than 1.6% total sulfur; and south zone high copper sulfide.

For each category, a linear regression analysis was used to obtain a relationship predicting residue assay as a function of head grade. The data and regression results are shown in Figures 3.1 to 3.3.

PDI compensated for scale-up of laboratory column test results to heap leach results by multiplying recovery equations by 95%. The following recovery equations were obtained from their study:

Oxide % Recovery = 95 x [0.988 - 0.027/Au, g/T].
Mixed and Fracture <1.6% S % Recovery = 95 x [0.909 - 0.131/Au, g/T].
Sulfide and Fracture >1.6% S % Recovery = 95 x [0.634 - 0.098/Au, g/T].
South High Copper % Recovery = 95 x [0.203 - 0.100/Au, g/T].

The higher sulfur content fracture oxidation ore type occurs in the south portion of the deposit and in the lower elevations of the

north portion of the deposit. The lower sulfur content fracture oxidation ore type occurs in the upper portion of the north area of the deposit.

South high copper equation is for materials south of section 4200 N with copper values greater than 1000 ppm total copper. The high copper ore constitutes a minor portion of the ore deposit.

Gold recovery was projected by PDI's use of the above recovery equations to average 90%, 75%, 58% and 52% for oxide, mixed, silicified sulfide, and non-silicified sulfide, respectively, for an overall recovery of 63.5% for the project.

Table 3.1
Summary of Column Results and Recovery Model

Column	Composite	Description	Head Grade Total S (%)	Au (g/t)	Ag (g/t)	Cu (ppm)	Test Tail Au (g/t)	Ag (g/t)	Extraction Au Test	Bottle	Equations	Model	Ag Test
OXIDE													
35	96PM054B	Silicified Oxide Vuggy Bx	0.12	0.53	1.17	34	0.07	1.00	86.8%	89.8%	93.7%	89.0%	14.5%
29	96PM037A	Argillized Oxide Bx	0.15	1.04	0.86	37	0.05	0.60	95.2%	96.6%	96.2%	91.4%	30.2%
22	96PM029A	Argillized Oxide	0.17	1.98	2.05	50	0.02	0.99	99.0%	97.6%	97.4%	92.6%	51.7%
15	96PM019A	Weakly Silic./Argil. Oxide Bx	0.18	2.43	3.44	111	0.03	3.00	98.8%	96.7%	97.7%	92.8%	12.8%
32	96PM046A	Silicified Oxide Vuggy Int. Bx	0.02	12.29	1.36	24	0.18	0.90	98.5%	97.3%	98.6%	93.7%	33.8%
		Average	0.13	3.65	1.78	51	0.07	1.30	95.7%	95.6%	96.7%	91.9%	28.6%
MIXED 2 & FRACTURE <= 1.6 % S													
5	III-3 NZM	Silicified Mixed	3.14	0.55	8.07	89	0.20	6.80	63.4%	68.6%	66.9%	63.6%	15.7%
3	96PM017B	Silicified Spotty Oxidation	0.35	0.74	4.35	50	0.20	2.93	73.0%	82.2%	73.2%	69.5%	32.6%
18B	PM-018	Silicified Mixed	1.66	0.87	11.70	60	0.23	9.50	73.6%	86.0%	75.8%	72.1%	18.8%
24	96PM029C	Silicified Mixed2 Vuggy Bx	2.96	0.92	8.81	44	0.30	6.94	67.4%	73.0%	76.7%	72.8%	21.2%
13	96PM018C	Silicified Fracture Vuggy T-RF	1.46	0.93	3.93	69	0.19	3.00	79.6%	74.7%	76.8%	73.0%	23.7%
14	96PM018D	Silicified Mixed2 Vuggy T-RF	0.60	0.96	2.30	63	0.11	1.69	88.5%	78.7%	77.3%	73.4%	26.5%
42	96PM064D	Silicified Mixed2 Hard White	0.19	1.24	0.56	21	0.21	0.50	83.1%	79.3%	80.3%	76.3%	10.7%
17	96PM019C	Silicified Mixed2 SlVuggy	0.17	1.44	36.15	14	0.27	30.31	81.3%	91.5%	81.8%	77.7%	16.2%
11	96PM018A	Silicified Mixed2 Vuggy CBx	0.95	1.47	4.37	62	0.33	3.35	77.6%	74.6%	82.0%	77.9%	23.3%
36	96PM054C	Silicified Spotty Oxidation Bx	1.64	1.52	3.00	54	0.16	1.40	89.5%	92.4%	82.3%	78.1%	53.3%
10	96PM015E	Silicified Fract./Mixed2 Vuggy	0.11	1.62	1.55	20	0.27	1.09	83.3%	80.3%	82.8%	78.7%	29.7%
11	III-8 ArgMx	WklSil/Arg. S/Sil, Mx Vug Bx	1.39	1.84	3.39	59	0.12	2.50	93.7%	94.5%	83.8%	79.6%	26.2%
18	96PM019D	Silicified/Argillized Mixed2 Bx	0.46	1.87	16.76	74	0.15	13.66	92.0%	90.1%	83.9%	79.7%	18.5%
6	III-4 NZM	Silicified Mixed Hi Cu Zones	1.24	2.03	9.30	69	0.28	6.30	86.2%	91.4%	84.4%	80.2%	32.3%
37	96PM054D	Silicified Mixed2 Bx	0.10	2.03	0.69	10	0.37	0.50	81.8%	85.1%	84.4%	80.2%	27.5%
33	96PM046C	Silicified Mixed2 Intr. CBx	0.33	2.05	0.47	21	0.43	0.36	79.0%	80.0%	84.5%	80.3%	23.4%
17B	PM-017	Silicified Mixed	1.33	2.10	15.00	90	0.38	12.00	81.9%	89.4%	84.7%	80.4%	20.0%
7	III-4 NZM	Silicified Mixed Hi Cu Zones	1.24	2.14	10.62	78	0.37	7.50	83.0%	91.4%	84.8%	80.5%	29.4%
1	96PM002A	Silicified Mixed2	0.61	2.40	4.72	53	0.38	4.19	84.2%	90.2%	85.4%	81.2%	11.2%
22B	PM-022	Argillized Mixed	2.35	2.63	5.50	70	0.32	3.65	87.8%	93.5%	85.9%	81.6%	33.6%
12	96PM018B	Silicified Mixed2 Vuggy	0.74	3.72	2.57	50	0.62	2.00	83.3%	85.6%	87.4%	83.0%	22.2%
6	96PM015A	Silicified Mixed2 Contact Bx	0.41	6.39	17.47	138	0.64	13.16	90.0%	90.5%	88.8%	84.4%	24.7%
		Average	1.06	1.88	7.79	57	0.30	6.06	82.0%	84.7%	81.5%	77.5%	24.6%
SULFIDE & FRACTURE >1.6 % S													
16	96PM019B	Weakly Silic/Argillized Sulfide	5.77	0.50	2.18	63	0.32	2.00	36.0%	34.1%	43.8%	41.6%	8.3%
4	96PM017C	Moderately Silicified Sulfide	4.61	0.60	2.93	320	0.28	2.61	53.3%	74.5%	47.1%	44.7%	10.9%
9	96PM015D	Argillized Sulfide Low Grade	5.73	0.64	1.00	44	0.25	0.73	60.9%	50.0%	48.1%	45.7%	27.0%
2	96PM017A	Weakly Silic/Argillized Sulfide	5.57	0.64	2.45	113	0.35	1.97	45.3%	64.1%	48.1%	45.7%	19.6%
9	III-6 LGS	Silicified Sulfide	5.97	0.66	2.27	121	0.24	2.00	63.8%	66.9%	48.6%	46.2%	11.8%
25	96PM029D	Silicified Sulfide Bx	4.95	0.69	5.07	97	0.43	4.79	37.7%	20.5%	49.2%	46.7%	5.5%
30	96PM037D	Silic/Argillized Sulfide CBx	5.84	0.78	1.32	65	0.33	0.80	57.7%	61.6%	50.8%	48.3%	39.4%
40	96PM054G	Silicified Fracture Vuggy	2.40	0.86	18.84	158	0.40	14.80	53.5%	61.6%	52.0%	49.4%	21.4%
21B	PM-021	Argillized Sulfide	1.62	0.87	1.50	80	0.42	1.13	51.7%	72.5%	52.1%	49.5%	24.7%
41	96PM054H	Argillized Sulfide DF4 Copper	6.86	0.96	7.83	264	0.37	7.30	61.5%	66.4%	53.2%	50.5%	6.8%
34	96PM046E	Silicified Fracture Vuggy	2.98	1.05	16.75	45	0.41	13.90	61.0%	67.6%	54.1%	51.4%	17.0%
23	96PM029B	Silicified /Argillized Spotty	2.48	1.35	3.61	44	0.57	2.65	57.8%	70.5%	56.1%	53.3%	26.6%
26	96PM029E	Silicified Sulfide Bx	3.86	1.35	4.13	93	0.76	3.80	43.7%	59.4%	56.1%	53.3%	8.0%
27	96PM029F	Silicified Sulfide T-RF	4.10	1.43	2.32	82	0.99	2.00	30.8%	31.6%	56.5%	53.7%	13.8%
39	96PM054F	Silicified Fracture Vuggy CBx	1.63	1.47	17.75	127	0.72	14.00	51.0%	65.1%	56.7%	53.9%	21.1%
10	III-7 Arg Sul	Argillized Sulfide Low Copper	5.15	1.52	2.55	112	0.73	2.00	52.3%	67.4%	57.0%	54.1%	21.6%
4	III-2 SZS	Silicified Sulfide	3.73	1.52	1.46	108	0.26	1.00	82.9%	78.1%	57.0%	54.1%	31.7%
2	III-1 NZS	Silicified Sulfide	7.25	1.57	10.63	88	0.65	9.00	58.7%	65.7%	57.2%	54.3%	15.3%
5	96PM017D	Silicified Sulfide	2.57	1.62	0.61	114	0.68	0.50	58.0%	66.0%	57.4%	54.5%	18.0%
12	III-9 SulCu	Silic/Argillized Sulfide/Fracture	5.64	1.63	4.75	376	0.59	3.00	63.7%	69.6%	57.4%	54.5%	36.8%
31	96PM037E	Silicified/Argillized Sulfide Bx	4.93	1.66	2.99	60	0.57	2.40	65.7%	64.1%	57.5%	54.6%	19.7%
8	96PM015C	Argillized Sulfide Mod Copper	3.78	1.78	1.66	275	0.91	1.48	48.9%	51.6%	57.9%	55.0%	10.8%
20B	PM-020	Argillized Sulfide	3.82	1.78	1.40	100	0.64	1.13	64.0%	74.1%	57.9%	55.0%	19.3%
38	96PM054E	Silicified Fracture Vuggy CBx	0.99	1.92	1.42	18	0.86	0.50	55.2%	76.6%	58.3%	55.4%	64.8%
19	96PM022A	Silicified Fracture	2.83	2.02	2.33	49	0.77	1.40	61.9%	67.0%	58.5%	55.6%	39.9%
20	96PM022B	Silicified Fracture Vuggy	1.96	2.29	3.85	71	0.80	2.96	65.1%	69.1%	59.1%	56.2%	23.1%
		Average	4.12	1.28	4.75	119	0.55	3.84	55.5%	62.1%	54.1%	51.4%	21.7%
SOUTH ZONE SULFIDE HIGH COPPER													
8	III-5 SZS	Silic Sulfide S HiAu HiCu	7.88	7.33	7.43	1235	5.84	6.25	20.3%	19.7%	18.9%	18.0%	15.9%
		GRAND AVERAGE	2.53	1.88	5.71	106	0.49	4.51	69.8%	74.0%	69.1%	65.6%	23.5%

18.2.2.2 RDi Recommended Revised Gold Recovery Equations

PDI/PDDRC did extensive testwork on samples from the deposit. They ran over 75 column tests and several hundred bottle roll tests. The metallurgical testwork was reviewed by MON consultants and additional column testing was undertaken on South Zone ore.

The highlights of this review and testwork indicated the following:

- While estimating the recovery of the deposit, PDI had discounted it by 5% to compensate for uncertainties of scaling up laboratory column test results to actual heap leach results. Mr. V.G. Lofftus, PDI metallurgist, remarked in a memorandum dated May 26, 1995, "that discounting of the recovery was inappropriate. Recoveries should actually have been increased because the tests were cut off too soon". He suspected that the sulfide composite will continue to yield gold for a long time as the sulfides oxidize. This is consistent with what RDi saw in the columns currently testing at Metcon. These comments indicate that the PDI engineers may have underestimated the gold recovery for the project.

- The South Zone ore tended to give higher gold extraction than the North Zone ore at the same crush size. For example, Column Test 4 with South Zone ore gave 82.9% gold extraction as compared to 64% for the North Zone ore in Column Test 2 (Table 3.2). This observation was also noted by PDI engineers in their study of nine composite samples in August 1996. They remarked that "the extraction measurements show that the gold extraction is higher from South Zone ore (zonation effect) at both crush sizes with yields of 93% in 406 days at minus 1 inch and 83% in 81 days at minus 1/2 inch" as compared to North Zone sulfide composite ore. They further remarked "that fine crushing increased gold extraction from North Zone ore, but the effect is marginal for South Zone ore due to high gold extraction obtained at the minus 1 inch crush size".

- The gold in Mulatos ore has two component systems; a portion of gold leaches very quickly and the remaining gold leaches very slowly. Extraction of 93% of gold in 406 days

of leach time at minus 1 inch crush size confirms that given enough leach time, gold recovery can be improved. As stated earlier, Mr. Lofftus remarked that recovery should actually have been increased because the test were cut off too soon.

- The current Metcon column testing of sulfide ore from the South Zone indicates gold extraction of ±80% at 12.5 mm crush size. This is significantly higher than the recoveries used in developing the PDI projections.

- The current testwork at Metcon demonstrates finer crushing at ¼ inch results in a better recovery than ½ inch or 1 inch. It may be that the testwork done by PDI was done at low pH and terminated prematurely.

Table 3.2
Summary of Single Column Leach Tests
Placer Dome Report No. 6

Comp	Column	Crush	Days	Kg/t		G Au/t		Extraction %			Location
				NaCN	CaO	Feed	Tail	Au	Ag	Cu	
1	1	-1"	406	1.06	7.1	1.56	0.69	56	19	5	NZS
1	2	-1/2"	74	0.48	3.5	1.44	0.51	64	10	2	
2	3	-1"	406	1.26	6.6	1.46	0.10	93	12	24	SZS
2	4	-1/2"	81	0.63	4.1	1.52	0.26	83	31	13	
3	5	-1/2"	81	0.46	2.7	0.58	0.24	59	17	4	NZM
4	6	-1/2"	60	0.40	2.3	2.03	0.28	86	32	3	NZM
4	7	-1/2"	60	0.24	2.3	2.14	0.37	83	29	2	
5	8	-1/2"	180	4.00	3.0	7.34	5.73	22	19	6	SZS (Hi C)
6	9	-1/2"	81	0.52	2.6	0.66	0.24	63	12	9	LGS
7	10	-1/2"	90	0.97	5.0	1.52	0.73	52	21	29	Ar
8	11	-1/2"	60	0.56	4.7	1.84	0.12	93	26	12	ArM
9	12	-1/2"	81	0.66	5.2	1.63	0.59	64	37	15	SCu

Placer Dome Report No. 6 channel samples from the Buena Vista II, El Nopal, and El Cantil underground adits and core samples from the 1994 metallurgical drill program were used to investigate various aspects of heap leaching. Nine composites were assembled and used to conduct twelve shingle column tests and one three-stage multi-column test. Crush size, alteration, location, and copper content were investigated. Splits from each composite were taken for grinding and bottle-roll cyanidation tests. This series of test work is referred to as Phase III. Composite locations along with column leach results are shown on oxidation geology sections in Appendix III, PDDRC Report No. 6. Metallurgical composite descriptions are shown in Appendix V PDDRC Report No. 6.

The following methodology was used to update the recovery equations to reflect the current mining plans and the findings of the metallurgical review of the past testwork:

- There appears to be no justified reason to apply 5% correction factors to the recovery models for scale up considering the fact that the column tests were terminated too soon. The .95 factor has therefore been removed from all four equations. No additional modification was made to the oxide, mixed and fracture <1.6% S, and south zone high copper recovery models.

- Since the new plan calls for mining only the main (4275N to 4360N) and south zone (4075N to 4275N) areas at this time, the metallurgical data for north zone (4350N to 4560N) needed to be eliminated from the summary table given in Table 3.1. This was done by reviewing the drilling data and correlating it to the various mining areas and samples used for the column tests.

- The data for recovery projection was reduced from 26 columns tests for sulfide and fracture >1.6% S given in Table 3.1 to 11 tests undertaken with samples from south zone (Table 3.3). The average gold recovery in the columns was 60.9% and in bottle roll tests was 66.0%. Applying the same methodology used by PDI, the recovery was averaged from column and bottle roll tests. The gold recovery was projected to be 63.5% for sulfide ore in the south zone.

- The projected recovery is plus minus 8% higher than the equations developed by Placer Dome. The finer crush to 3/8 inch and longer leach time will have an additional 4% effect on gold recovery as indicated by Metcon column testwork on sulfide ore. They achieved 77.2% gold extraction at 0.5 inch crush size in 67 days leach cycle and 80.9% at 1/4 inch crush size. These recoveries are significantly higher than the recoveries used in our models.

The equation was modified by RDi by sorting Placer Dome's data, Table 3.1 Sulfide and Fracture >1.6%S, into those columns from the South Estrella zone only. This resulted in Table 3.3

Table3.3
Summary of Column Results and Recovery Model for South Zone Sulfide Ore

Column	Composite	Description	Head Grade				Test Tail		Au Extraction , %				Ag
			Total S(%)	Au g/T	Ag g/T	Cu ppm	Au g/T	Ag g/T	Test	Bottle	Equation	Model[1]	Extraction % Test
4	96PM017C	Modestly silicified sulfide	4.61	0.60	2.93	320	0.28	2.61	53.5	74.5	47.1	44.7	10.9
9	96PM015D	Argillized sulfide low grade	5.73	0.64	1.00	44	0.25	0.73	60.9	50.0	48.1	45.7	27.0
2	96PM017A	Weakly silic/argillized sulfide	5.57	0.64	2.45	113	0.35	1.97	45.3	64.1	48.1	45.7	19.6
9	III-6 LGS	Silicified sulfide	5.97	0.66	2.27	121	0.24	2.00	63.8	66.9	48.6	46.2	11.8
21B	PM-021	Argillized sulfide	1.62	0.87	1.50	80	0.42	1.13	51.7	72.5	52.1	49.5	24.7
4	III-2SZS	Silicified sulfide	3.73	1.52	1.46	108	0.26	1.00	82.9	78.1	57.0	54.1	31.7
5	96PM017D	Silicified sulfide	7.25	1.57	10.63	88	0.65	9.00	58.7	65.7	57.2	54.3	15.3
8	96PM015C	Argillized sulfide mod. Copper	3.78	1.78	1.66	275	0.91	1.48	48.9	51.6	57.9	55.0	10.8
19	96PM022A	Silicified fracture	2.83	2.02	2.33	49	0.77	1.40	61.9	67.0	58.5	55.6	39.9
20	96PM022B	Silicified fracture vuggy	1.96	2.29	3.85	71	0.80	2.96	65.1	69.1	59.1	56.2	23.1
CL-05	METCON	Sulfide	3.08	2.06	2.83	122	0.51	2.15	77.2	–	–	–	23.9
	AVERAGE		4.19	1.33	2.99	126	0.49	2.40	60.9	66.0	53.4	50.7	21.7
	MODIFIED MODEL								63.5	–	66.0	–	–

(Sulfide and Fracture > 1.6% S)
Note: 1   Original model developed by Placer Dome.

It can be seen that the test average recovery for the southern sulfide tests has risen to 63.5% from 58.8% in PDI's table. This and the higher recovery from ore crushed to P80 of 3/8 inch versus a P80 of ½ inch in Placer work, leads to the 0.734 factor in the RDi equation. The negative factor in the equation reduces 0.734 to approximately 0.635 or 63.5 %, based on head grade.

The following RDi revised recovery equations were used for the deposit:

Oxide % Recovery = [0.988 - 0.027/Au, g/T]

Mixed and Fracture <1.6% S% Recovery = [0.909 - 0.0131/Au, g/T]

Sulfide and Fracture >1.6% S% Recovery = [0.734 - 0.098/Au, g/T]

South Zone High Copper % Recovery = [0.203 - 0.100/Au, g/T]

Gold recovery will average 96.4%, 82.9% and 67.6% for oxide, mixed and sulfide ores. The overall recovery calculated for the project based on the actual proportion of each ore type is 72.9%

18.2.3 Metallurgical Test Program

In order to support the MSA/Placer 1997 Feasibility Study, twelve metallurgical test programs were conducted by Placer Dome Division Research Center (PDDRC) and Mineral Real de Angeles (MRA). These programs were undertaken to determine the economically optimum gold extraction process and process conditions for treating ores from the deposit and to obtain estimates for metal extraction and reagent consumption.

Initial metallurgical investigations were undertaken by MRA, PDDRC, and Hazen Research Inc. (HRI) in 1989 and 1990. In these investigations, conventional gold extraction processes such as milling followed by cyanide leaching, gravity concentration, heap leaching, and flotation were evaluated. A combination of milling and heap leaching ("split-flow") was also evaluated. In this process, crusher and/or SAG mill products were screened with the oversize fraction column-leached and the undersize treated by conventional bottle-roll testing.

In 1994, representative samples of the deposit were obtained by diamond drilling (eight metallurgy holes) Figure 3.4. Metallurgical

investigation of these samples was undertaken by PDDRC and HRI. Crushing, grinding, bottle-roll leach, column leach, flotation and "split-flow" tests were conducted. During 1995, additional column and bottle-roll leach tests were performed by PDDRC using core from the 1994 metallurgy drilling and from underground channel sampling. Crush size, alteration, location in the deposit, and copper content were investigated. In addition, a three-stage multi-column test was conducted.

As there are numerous rock types, alteration states, and gold and copper grades in the deposit, a comprehensive bottle-roll program was initiated in the fall of 1995 to identify which factors and to what extent these factors affected metallurgy. Composites from previous test programs had combined these different factors. Using coarse reject samples from geological core holes drilled in the 1994 campaign, 222 composites within the estimated pit limits were assembled. Testing of these composites continued into 1996.

Because previous column test program composites did not adequately represent the various ore types, an additional 11 core holes were drilled in 1996 to obtain samples for metallurgical column testing. Composites were selected from these holes plus part of one geotechnical hole for column leaching testing. A total of 42 composites were tested in 48 different column tests. Coarse reject from interval assaying was also subjected to bottle-roll testing for each composite. All test work was conducted at PDDRC. Coarse bulk samples were also taken from underground workings and submitted to Nordberg Inc., for crusher impact and abrasion testing.

Reverse circulation (RC) drilling in 1996 encountered new zones of mineralized material. Exploration in an area referred to as the "North Extension" located north of and adjacent to the Mulatos deposit defined additional minable reserves. Condemnation drilling in the waste dump area discovered a sulfide zone of ore-grade material. Metallurgical composites of the RC chips were assembled for both areas. Bottle-roll tests were conducted at PDDRC to determine the metallurgy of the various rock types encountered.

In 2002 MON contracted RDi to review the metallurgical testwork undertaken by PDDRC. Based on the findings, MON decided to undertake additional testwork at Polysis., RDi, and Metcon with the primary objective of identifying process options, which would enhance gold recovery from sulfide bearing ores. The testwork consisted of crushing tests, gravity tests, bottle roll and column leach tests on sulfide-bearing channel samples from the deposit.

18.2.4 Rock Types

The Mulatos deposit is a Au-Ag-Cu, high-sulfidation, acid-sulfate type epithermal system, hosted within an Oligocene rhyodacite flow/dome and breccia complex. Mulatos mineral deposits are particular in that they occur primarily in areas of massive pervasive silicification in volcanic host rocks. Quartz veins and quartz stockwork zones seldom occur. Geological and mineralogical details pertinent to metallurgy are shown in Appendix II, Volume 3A of the MSA Placer 1997 Feasibility Study Report.

Mulatos mineralization consists of two separate, yet contiguous, Au-Ag-Cu deposits. One is hosted within a southern rhyodacite flow dome, is generally located south of section 4350N, and is referred to as "South Zone" in the metallurgical test programs. The other deposit is hosted within the Buena Vista breccia complex, is generally located north of Section 4350N, and is referred to as the "North Zone" in the metallurgical test program.

18.2.4.1 Alteration

Silicic alteration occurs in the central part of the deposit and is the primary host for Au-Ag-Cu mineralization. Approximately 80% of the contained gold occurs in moderately to intensely silicified rocks. This alteration is subdivided into two major types: vuggy silica and pervasive silica alteration. Vuggy silica material contains the highest gold grades in the deposit. The degree of silicic alteration within the deposit is quite variable and is often mixed with varying degrees of argillic alteration. Argillic alteration is characterized by the presence of pyrophyllite, kaolinite, and/or alunite occurring as a halo around silicified zones.

18.2.3.2 Oxidation

Oxidation ranging from totally oxidized to fresh sulfide occurs within the deposit. The usual vertical sequence of oxide-mixed oxide/sulfide - sulfide does occur in a general sense at Mulatos, but because of high-angle vertical fault structures, oxidation has occurred locally in deeper zones within the deposit. Conversely, sulfide zones can be seen in surface or near-surface zones above mixed zones. Mixed zones contain both oxide and sulfide minerals in any proportion ranging from nearly all oxide to nearly all sulfide. Mixed zones frequently show up as leached wells in cross-section, generally along fault zones.

For purpose of geology and metallurgical modeling, oxidation has been divided into four categories: oxide, pervasively mixed, fracture-controlled mixed, and sulfide. Sections showing oxidation modeling along with pit outlines and metallurgical composites are located in Appendix IV Volume 3A of the MSA/Placer 1997 Feasibility Study Report.

Oxide rock type zones occur primarily near the surface, in the leaching zone, and are largely a result of surface weathering. Several deeper zones of oxidized material exist primarily in highly fractured areas where permeability is enhanced along major structural zones. Oxide ores make up about 7.7% of the reserve.

Pervasively mixed rock type zones, referred to as mixed-2, also occur in leaching zones and are characterized by weak to moderate pervasive oxidation in which the rock is generally oxidized but sulfide minerals remain. Fracture-controlled mixed rock type zones, referred to as mixed-1, are characterized by intense oxidation along narrow (1 mm to 1000 mm) fractures leaving a majority of the rock in the sulfide state. Pervasively mixed ores and fracture-controlled mixed ores make up about 24.9% of the reserve.

Sulfide zones generally occur in the deepest portion of the deposit and make up about 67.4% of the ore reserves. Sulfide zones contain no oxide minerals. There are some areas where sulfide zones out crop at the surface. In a general sense, mixed zone/sulfide zone interfaces occur closer to the surface in the southern Nopal/Nopalito block while interfaces are much deeper in the northern Buena Vista breccia block.

18.2.5 Mineralogy

Minerals observed in the deposit include: pyrite, enargite, chalcopyrite, chalcocite, molybdenite, gold, covellite, bornite, tetrahedrite-tennantite, marcasite, copper oxides, specularite, hematite, limonite, goethite, jarosite, pyrophyllite, kaolinite, alunite, montmorillonite, barite, chlorite, and epidote. Free gold is commonly found in hematite-filled fractures. Gold also occurs in pyrite and as gold-silver telluride and possibly as solid-solution in some copper sulfide minerals. Pyrite is by far the most common sulfide mineral.

Important minerals observed in the **oxide zones** include: hematite, limonite, jarosite, goethite, and copper oxides. In the geological

model, oxide zones contain only oxide minerals; no sulfide minerals are present.

Minerals found in the **mixed zones** include those described for the oxide zones as well as specular hematite and the sulfide minerals: pyrite, enargite, chalcopyrite, molybdenite, chalocite, covellite, bornite, tetrahedrite-tennanite, and marcasite. Free gold can sometimes be found in hematite-filled fractures.

Minerals found in the **sulfide zones** include: pyrite, enargite, chalcopyrite, molybdenite, gold, chalcocite, covellite, bornite, tetrahedrite-tennantite, and marcasite and specular hematite. Significant lower gold extractions are obtained in the copper sulfide zone in the southern block. Copper sulfide zones in the north block experience gold extractions comparable to those from low copper sulfide zones

Four programs of mineral microscopy work done in conjunction with metallurgical testing were performed by PDDRC. The first microscopy work was performed in 1988 by Vancouver Petrographics Ltd. of Fort Langley, British Columbia, on three MRA composite samples. The second was performed in 1990 by Comisión de Fomento Minero of Chihuahua, Mexico on one MRA composite head and residue. The third was conducted by Chamberlain Geological Associates of Victoria, British Columbia in 1994 on six selected core samples from the 1994 geological drilling program and two flotation concentrate leach residues. The fourth, conducted in 1996 by AMTEL of London, Ontario, was on selected column leach residues from the 1995 Phase III metallurgical column test program. Descriptions and summary of findings for each program are in Appendix II, Volume 3A of the MSA Placer 1997 Feasibility Study Report.

Observations and conclusions from the microscopy programs are:

- Native gold is the predominant gold-bearing mineral.

- Pyrite is the predominant sulfide mineral, which has been altered to hematite in the oxide and mixed zones.

- Gold is primarily in association with pyrite, occurring in two main modes; as free grains attached to pyrite and within pyrite. Figure 3.5 shows examples of "free" gold.

- About 15% of gold minerals are associated with iron oxides.

- Native gold has a low silver content. Figure 3.5 shows relative distributions of gold and silver in a native gold particle.

- Gold occurrence may be different in the high copper areas.

Four types of pyrite were identified: coarse grained, fractured (mylonitic), with dissolution features, and fine-grained. Figure 3.6 shows the four types of pyrite. Coarse-grained pyrite contains very little (<0.5 ppm) gold, while the others contain up to 45 ppm gold. Gold contained in any of these pyrite types is either dissolved in the crystal structure or it occurs as colloidal (<0.1 µm) micro-inclusions. The second photomicrograph in Figure 3.6 shows typical distribution of gold (seen as white specks) in a native gold and pyrite particle. As can be seen, the gold contained in pyrite is very finely dispersed.

The mineralogical work performed explains metallurgical responses observed during testing. The following mineralogical factors affect the metallurgy of Mulatos ores:

- Gold is associated with and included in iron oxides, which excludes flotation or gravity processing as options for oxide and mixed ores.

- A majority of the gold is "free" and readily cyanide soluble.

- Grains of "free" gold are relatively fine and attached to the surface of pyrite grains, making gravity separation of gold from pyrite difficult.

- The remaining gold is locked in sulfide minerals in a very fine state, making fine crushing or oxidation the only liberation options to recover this gold.

- Fine crushing or grinding and sufficient leach time, cyanide, and lime needs to be applied to dissolve the gold and silver. In two programs that looked at heap leach test residues, undissolved "free" gold was encountered.

- A majority of the silver appears to be associated with sulfide minerals and not electrum; this would explain the low (20%) silver extractions seen in the test work.

19   MINERAL RESOURCE AND RESERVE ESTIMATES

A mineral resource estimate is completed for Mulatos and a open pit reserve is determined for the south Estrella area of Mulatos. Table 19.1 shows the total block model area that has been tabulated for the mineral resource as a dark blue box on the map of the sample locations. The final pit limit used to define the open pit reserve is included on the same figure as a light blue line.

The mineral resource is shown by gold cutoff grades on Table 19.1

Table 19.1
Mulatos Resource

Gold Cutoff, g/t	Measured		Indicated		Total Measured + Indicated Resource		Inferred	
	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)
0.20	15,039	1.24	125,147	0.83	140,186	0.88	54,667	0.50
0.40	11,978	1.48	81,122	1.12	93,100	1.17	21,192	0.86
0.60	**9,089**	**1.80**	**53,127**	**1.46**	**62,216**	**1.51**	10,382	1.26
0.80	7,124	2.10	37,161	1.79	44,285	1.84	6,336	1.63
1.00	5,642	2.42	27,452	2.11	33,094	2.17	4,240	1.99

An open pit has been designed and the proven and probable reserves within the pit are summarized on Table 19.1A. The inferred column is for information only. It is not used in any way. The recovery and processing costs vary by ore type, thus a 'net of process' value has been calculated for each ore block in the model. The net of process value is defined as the value based on:

(block gold grade x recover x metal price) – (process + G&A costs).

The pit reserve has been tabulated using this net of process value. The internal cutoff grade is that grade that covers the process and general and administrative costs and recovery losses. The range of this cutoff on a gold cutoff basis at $350/oz gold and the assumed costs and recoveries range from a low of 0.34 g/t gold for the oxide ore type of the reserve to a high of 0.63 g/t gold in the silicified sulfide ore type.

Table 19.1A,

South Estrella Pit Reserve – Based on Mine Production Schedule

	Proven		Probable		Proven + Probable	
Ore Type	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)
Oxide	324	1.18	2,478	1.12	2,802	1.12
Mixed, Non-silicified	184	1.72	1,025	1.56	1,209	1.58
Mixed, Silicified	1,865	1.85	5,998	1.59	7,863	1.65
Sulfide, Non-silicified	1,466	1.60	6,990	1.45	8,456	1.48
Sulfide, Silicified	3,409	2.01	12,628	1.76	16,037	1.81
Total	7,248	1.84	29,119	1.59	36,367	1.64
Total Pit Tonnage = 87,937						

This reserve is the sum of the mine production schedule, which used a declining cutoff grade over time.

The economics used to calculate the net process value are included in Section 19.3.

19.1   RESOURCE MODEL – DATA BASE AND ADDITIONAL CHECKING

The resource model for the Estrella area of Mulatos was developed by the independent consulting firm of RMI (Mike Lechner, R.G.). The drill hole, geologic and topographic information was provided to RMI by AGI for the resource estimate. RMI did many checks on the data prior to making a resource estimate. This section describes the data transfer, checking and statistics of the data base used for the resource estimate.

19.1.1 Database

The Mulatos drill hole database contains information that was collected by four companies: Minera Real de Angeles (MRA), Kennecott, Placer Dome Incorporated (PDI), and Alamos Gold Incorporated (AGI). Approximately 60 percent of the drill hole data were collected by Placer Dome during their involvement with the project from 1993 to 2000. Most of the MRA data are located within the main Mulatos deposit while a significant number of the Kennecott drill holes are located in the El Victor area northeast of the main Mulatos deposit.

In addition to drill hole assay data, other key information such as topography, density, geotechnical, and metallurgical information were collected by Placer Dome and used in this study. Placer Dome's last

geologic interpretation of lithology, alteration, and oxidation were used in developing a resource model. These geologic units were used primarily for specific gravity and ore type assignments

19.1.2 Data Transfer

All of the historical drill hole data were obtained from PDI, who stored the data in ASCII Geolog files. The information stored in those files was imported into acQuire™, a relational database manager that is commonly used in the mining industry. There were two sets of Geolog files, Old Geolog and New Geolog, each with different formats and data structures. The underground channel and muck sample data were stored in ASCII CSV files and were also imported into acQuire™. Significant diligence was required in mapping the various data fields from the Geolog files to acQuire™ to avoid errors.

The drill hole data were then imported into MineSight® binary drill hole files. Basic descriptive statistics (number of meters, length weighted mean grade, and standard deviation) were tabulated from the gold assays stored in the MineSight® drill hole files at four different cutoff grades. These same statistics were then tabulated from the raw data stored in the Geolog files and then compared with those generated from the MineSight® files. The statistical parameters from each data source were identical indicating that the data transfer was successful.

Alamos Gold Incorporated drilled 15 underground core holes in late 2003. The assay results for 13 of these drill holes were available for estimating gold resources. The data for these holes was obtained as ASCII CSV files from the laboratory and loaded into MineSight®.

19.1.3 Sample Data

The total Mulatos drill hole database is comprised of six basic types of sample data: surface core, underground core, reverse circulation (RVC), surface airtrack, underground channel samples, and underground muck samples. Table 19.2 summarizes the sample database by sample type, the number of meters of each data type, and the percentage of each data type. The database type code is also shown for each data type. About twelve percent of the data shown in Table 19.2 are located well beyond the limits of the resource model used in for this study.

Table 19.2
Sample Data – Total Project Area

Type	Type Code	Number	Meters	Percentage
Surface Core	1	183	33,837.14	28.9%
U/G Core	2	13	1,565.08	1.3%
RVC	3	424	74,139.55	63.3%
U/G Channel	4	17	1,649.93	1.4%
Airtrack	5	34	5,725.68	4.9%
U/G Muck	6	10	229.28	0.2%
Grand Total		681	117,146.66	100.0%

Table 19.2 A tabulates the number of sample types and meters of data that were within the resource model that is the subject of this report.

Table 19.2 A
Sample Data – Resource Model Area

Type	Type Code	Number	Meters	Percentage
SurfaceCore	1	159	29,878.33	29.0%
U/G Core	2	13	1,565.08	1.5%
RVC	3	360	63,842.81	62.0%
U/G Channel	4	17	1,649.93	1.6%
Airtrack	5	34	5,725.68	5.6%
U/G Muck	6	10	229.28	0.2%
Grand Total		593	102,891.11	100.0%

The data summarized in Table 19.2A show that about 30 percent of the database consists of diamond drill core data and around 60 percent of the database is comprised of reverse circulation drilling data. The airtrack drill holes and underground muck samples were not used to estimate gold, silver, or copper grades. The underground channel samples only represent about two percent of the data that were used to estimate mineral resources. Not all of the data shown in Table 19.2 were assayed. Assay statistics are summarized in Section 19.1.9.

Table 19.3 summarizes the drill hole data by company that were used for estimating gold resources.

Table 19.3
Drill Hole Data by Company

Company	No. Drill Holes	No. Meters
Alamos Gold Inc.	13	1,565.08
Kennecott	69	15,305.53
Minera Real de Angeles	130	22,254.85
Placer Dome Inc.	381	63,765.65
Grand Total	593	102,891.11

Figure 19.1 is a plan map showing the distribution of the sample data for a portion of the Mulatos project area. The thick blue rectangle represents the resource model boundary. The topographic contour interval is 25 meters and a 500-meter grid was used. The ultimate design pit limit is shown in light blue.

The average drill hole spacing within the ultimate design pit is about 26 meters. Table 19.4 summarizes the number of holes per bench and the average drill hole spacing within the ultimate pit. The average spacing was calculated by taking the square root of the pit area divided by the number of drill holes within the area.

Table 19.4
Average Drill Hole Spacing

Bench Elevation	No. Drill Holes	Pit Area (m^2)	Ave. Spacing (m)
1425	40	18,716	22
1395	97	66,213	26
1365	168	124,016	27
1335	204	150,581	27
1305	248	165,891	26
1275	280	172,197	25
1245	263	172,113	26
1215	212	139,833	26
1185	121	90,625	27
1155	70	53,689	28
1125	36	26,738	27
1101	3	2,452	29
Averages (Area Weighted)	202	98,589	26

19.1.4 Drill Hole Surveys

In the late 1980's, Mineral Real de Angeles (MRA) established a coordinate grid system across the Mulatos project site that consisted of permanent survey monuments. After taking over the project in 1994, Placer Dome Exploration (PDX) commissioned Jose Ramos from Asesores Technicos Mineros S.A. de C.V. to survey MRA's triangulation net. The results from this survey indicated that MRA's survey grid was adequate for a mineral exploration program. PDX continued to use Ramos for surveying drill hole collar locations. Ramos also re-surveyed the underground workings to check the accuracy of the MRA's surveys. According to PDX's 1997 feasibility study, Ramos was able to relocate MRA's surveys within +/- 1.5 meters.

The drill hole collar elevation for every hole in the entire database was compared with the provided topographic surface. An elevation difference of greater than 1.5 meters was found for 19 drill holes. Most of these drill holes were located well beyond the limits of the mineralized area. Ground surveys were conducted in July 2003 with a high precision GPS instrument in order to establish the correct elevation for these 19 drill holes. Most of the drill sites had been reclaimed so it was difficult to definitively establish the original collar elevation for 13 of the 19 holes. The elevation for six of these drill holes were located and re-surveyed. Based on this field study, the collar elevation of 14 drill holes was changed. Eight of the holes were located within the mineralized zone, but most of these holes had a minimal elevation difference of 1.5 to 2.2 meters, although hole 96NE145 did have an elevation difference of 8.07 meters relative to the topographic surface. The elevation of 5 of the 19 holes was left unchanged as the difference in elevations was determined to be a function of reclamation disturbance. Table 19.5 summarizes the drill holes that were found to have elevation errors and shows the corrected elevations.

Table 19.5
Drill Hole Collar Elevation Changes

Hole ID	Initial Collar Location			Topo Elevatio	Elevati on Diff	Corrected Elevation	Note	Comments
	Easting	Northing	Elev.					
K-48	2248.50	4250.90	1331.90	1367.03	-35.13	1296.77	1	Collar lowered. Hole well outside of resource area
96WD075	2091.70	3979.86	1368.62	1386.30	-17.68	1352.00	2	Collar lowered. Hole well outside of resource area
96WD076	2161.27	3818.78	1328.98	1343.85	-14.87	1315.20	2	Collar lowered. Hole well outside of resource area
96NE145	1735.00	4975.00	1075.00	1083.07	-8.07	1066.93	1	Collar lowered - Hole covered with debris
98EV015	3252.08	5866.67	1097.40	1105.14	-7.74	1089.66	1	Collar lowered. Hole well outside of resource area
96WD142	3522.00	4106.00	956.00	961.00	-5.00	951.00	1	Collar lowered. Hole well outside of resource area
98EV016	3295.44	5867.34	1088.55	1093.20	-4.65	1083.90	1	Collar lowered. Hole well outside of resource area
96NE059	1730.79	4907.80	1083.17	1085.41	-2.24	1080.93	1	Collar lowered - Difference due to reclamataion
M-11	1673.67	4099.00	1259.55	1261.48	-1.93	1257.62	1	Collar lowered - Difference due to reclamataion
96NE056	1730.89	4905.20	1083.15	1085.06	-1.91	1081.24	1	Collar lowered - Difference due to reclamataion
96NE049	1706.65	4851.99	1110.10	1111.93	-1.83	1108.27	1	Collar lowered - Difference due to reclamataion
M-127A	1906.52	4092.95	1446.70	1448.48	-1.78	1445.60	2	Collar lowered - Difference due to reclamataion
98EI002	2219.13	4874.82	1291.63	1293.38	-1.75	1290.80	2	Collar lowered - Difference due to reclamataion
98EI016	2266.49	5037.82	1273.98	1275.55	-1.57	1274.50	2	Collar raised - Difference due to reclamataion
M-127B	1906.04	4093.90	1446.77	1448.30	-1.53	OK	3	Did not modify - Difference due to reclamation
K-95	1694.46	4158.05	1265.69	1269.20	-3.51	OK	3	Did not modify - Difference due to reclamation
M-107G	1851.98	4455.75	1312.47	1314.70	-2.23	OK	3	Did not modify - Difference due to reclamation
M-110	1780.56	4413.68	1255.05	1257.13	-2.08	OK	3	Did not modify - Difference due to reclamation
96NE047	1708.45	4851.83	1110.09	1111.96	-1.87	OK	3	Did not modify - Difference due to reclamation

Notes: 1 - Drill hole collar correction based on topo surface elevation
2 - Drill hole collar correction based on GPS ground survey
3 - Difference between topo surface and collar elevation due to reclamation ground disturbance

From the provided data it was determined that down-hole surveys were collected from 75 drill holes. According to Placer Dome's 1997 feasibility study report, down-hole surveys were obtained for 24 core holes during the 1993-94 drilling campaign. These surveys were made using a Sperrysun instrument.   Placer surveyed all core holes from 1996 onward. Based on the available survey data, the holes do not seem to deviate much. However it should be noted that approximately 12 holes have one or more survey intervals that show a pronounced deviation in azimuth relative to the adjacent survey readings. These surveys may be a result of erroneous readings. All of the surveyed hole inclinations were reasonable. Table 19.6 summarizes the drill holes that have suspect azimuth readings.

Table 19.6
Down-hole Survey Deviations

Borehole ID	Survey Depth	Degrees Azimuth Change/Meter
97RE001	40.50	9.08
97RE001	60.00	26.59
97RE001	81.00	10.55
97RE008	41.00	5.28
97RE008	133.00	6.35
97RE009	144.77	8.22
97RE009	202.69	6.03
97RE020	104.00	7.44
97RE021	27.50	7.60
97RE031	41.76	21.81
97RE031	104.00	18.68
97RE031	123.75	22.62
97RE034	59.50	9.62
97RE034	69.50	23.49
97RE034	100.00	5.97
97RE034	110.00	5.16
97RE036	79.50	24.92
97RE036	89.50	7.80
PD-47	64.01	7.03
PD-47	200.25	6.10
PD-50	178.61	6.09
PDM-86	145.39	7.26

The down-hole survey deviations shown in Table 19.6 are not considered to be material, since the deposit will be mined by open pit methods and the location of the ore is not as critical as an underground operation. However, the location of some mineralized horizons may be somewhat displaced because of erroneous survey readings.

19.1.5 Drill Hole Orientations

About 43 percent of the holes within the model area were drilled vertically. The remaining 57% were drilled primarily as steep westerly and easterly directed angle holes that were designed to intersect the mineralized system at acute angles. Table 19.7 summarizes the drilling data for holes within the resource model area. All of the "holes" shown in Table 19.7 with "flat" orientations represent about 1,500 meters of underground channel samples and 13 core underground core holes (about 1,600 meters) that were drilled in late 2003.

Table 19.7
Drill Hole Orientations

Inclination - Orientation	Number	Meters	Percentage
Vertical Downward Hole	252	43,736.41	42.5%
Steep Downward Northerly Angle Hole	33	6,243.75	6.1%
Steep Downward Northeasterly Angle Hole	1	200.00	0.2%
Steep Downward Easterly Angle Hole	81	14,753.18	14.3%
Steep Downward Southeasterly Angle Hole	6	1,155.56	1.1%
Steep Downward Southerly Angle Hole	38	6,248.27	6.1%
Steep Downward Southwesterly Angle Hole	2	258.70	0.3%
Steep Downward Westerly Angle Hole	137	26,223.50	25.5%
Steep Downward Northwesterly Angle Hole	2	409.05	0.4%
Shallow Downward Easterly Angle Hole	1	218.40	0.2%
Shallow Downward Southwesterly Angle Hole	1	106.10	0.1%
Flat Northerly Hole	3	180.90	0.2%
Flat Northeasterly Hole	6	502.65	0.5%
Flat Easterly Hole	2	212.40	0.2%
Flat Southeasterly Hole	3	148.68	0.1%
Flat Southerly Hole	6	275.40	0.3%
Flat Southwesterly Hole	6	455.68	0.4%
Flat Westerly Hole	7	923.08	0.9%
Shallow Upward Northeasterly Angle Hole	4	407.60	0.4%
Shallow Upward Southwesterly Angle Hole	2	231.80	0.2%
Total	593	102,891.11	100.0%

19.1.6 Assay Verifications

In 1994, PDI contracted Froidevaux, Srivastava and Schofield (FSSI) to verify the Mulatos database. FSSI reviewed all of the MRA, Kennecott, and PDI data using a team approach. The results of that verification effort were summarized in a report entitled "Mulatos Project Database Compilation and Verification". PDI loaded the verified data into a Paradox database. Starting in 1996, PDI compiled, managed, and verified their data on site using project personnel.

The accuracy of assays in the electronic drill hole database was verified for this study by selecting a group of drill holes that contained significant mineralized intersections and comparing the values against the original assay certificates. Fifty-eight drill holes totaling nearly 11,000 meters of drilling or about 11% of the drill holes used for estimating gold resources were examined. A total of 5,864 gold assay records were checked and 5 errors were found (0.09%), which is acceptable for a resources estimation. Eleven of the 58 drill holes that were selected did not have signed assay certificates or had assay results that were not printed on an independent lab stationery. Gold was the principal element that was checked, but where available,

silver, copper, sample numbers, down-hole from depth and to depths were all compared against the assay certificates. Table 19.8 summarizes the data that were checked.

Table 19.8
Assay Verification Summary

Data Source	No. Holes	No. Assays	% of Total	No. Meters	% of Total
Signed Certificates	47	5,864	10%	8,932	9%
Unsigned Certificates	11	852	2%	2,027	2%
Total	58	6,716	12%	10,959	11%

Forty-one out of the 58 drill holes that were examined had no errors. Several minor errors were discovered during the verification program. Table 19.9 summarizes the errors and error rate that were discovered during the database check. No errors were discovered for the unsigned certificates but those 852 assays were not used in the denominator to determine the error rate, as those certificates could not be verified as being official. The number of gold, silver, and copper assays that were found in the signed certified assay copies varied for each metal.

Table 19.9
Database Errors

Metal	No. Assays Checked	No. Errors Found	Error Rate
Gold	5,864	5	0.09%
Silver	5,294	7	0.13%
Copper	3,816	3	0.08%

19.1.7 Quality Assurance/Quality Control

According to PDI's 1997 feasibility study, about 10% of all drill hole assays were sent out for check assay prior to May 1994. After that date PDI began sending 20% of all samples to Bondar Clegg and Rocky Mountain Geochemical for check analysis.

Check assays for the MRA assays were analyzed at four commercial labs in 1988; Comision de Fomento Minero (CFM) in Hermosillo, Sonora, Skyline Labs in Tucson, Arizona, Cortez Mines in Nevada, and the Placer Dome Research Center in Vancouver, British Columbia. In 1989, an additional 306 MRA samples were re-assayed at the PDI Research Center in Vancouver. Figure 19.2 (taken from PDI's 1997 feasibility study) compares 263 MRA original assays with check assays at PDI's lab in Vancouver.

There were only 90 check assays that were available for the Kennecott drilling data plus 401 duplicate pulp assay results. The original Kennecott assays were completed at the Rocky Mountain Geochemical lab in Salt Lake City, Utah, and the check assays were analyzed at Skyline Labs in Tucson, Arizona. The mean grade of the 90 check assays was about 3.5 percent lower than the original assays but the relative difference plot shows no systematic bias. Figure 19.3 (taken from PDI's 1997 feasibility study) compares 90 Kennecott original assays with the Skyline check assays. Figure 19.4 compares 401 Kennecott duplicate pulp assay results.

Figure 19.5 compares 213 Placer Dome Research Center check assays with the original assays that were analyzed by Barringer for PDI's 1996 drilling program.

19.1.8 Assay Adjustment

Based on the conclusions that were derived from work completed by FSSI and PDI on sample reliability for various drilling campaigns, it was decided that some of the assays in the database needed to be adjusted. Assays from a portion of the 1988 MRA and 1996 PDI Phase 1 program were adjusted based on a statistical review of check assays that were completed for those drilling programs.

19.1.8.1 MRA 1988 Campaign

In PDI's 1997 feasibility study an analysis of check assays for MRA's 1988 drilling program indicated that the assays may be biased as much as 15 to 20%. A recent review of the check assay data revealed that a single high-grade assay from the Nopal underground workings (25.5 g/t Au) was biasing the global statistics. The check assays for this particular sample averaged 6.8 g/t Au. By removing this outlier, the overall global bias dropped to 10 percent.

As mentioned in Section 19.1.7, four laboratories were used by Placer Dome to verify the 1988 MRA assays. Table 19.10 compares basic statistical parameters for various check assay campaigns with the original MRA assays that were completed by Comision de Fomento Minero (CFM) in Hermosillo, Sonora.

Table 19.10
1988 MRA Check Assay Comparison

Data Set	Valid N	Mean gpt	Med	Min	Max	Lower Quartile	Upper Quartile	Quartile Range	Std Dev	Corr Coef
AUF_CFM	83	4.86	1.42	0.23	78	0.57	2.73	2.16	12.07	
CHK1_CFM	83	4.88	1.38	0.14	74	0.54	2.72	2.18	12.00	0.993
	%diff	0.39								
AUF_CFM	104	2.43	1.415	0.13	13.3	0.635	2.65	2.015	2.87	
CHK_SKY	104	2.18	1.3	0.01	17.7	0.5	2.5	2	2.91	0.896
	%diff	-10.28								
AUF_CFM	107	2.26	1.55	0.13	13.3	0.73	2.6	1.87	2.39	
CHK1_CTZ	107	1.99	1.33	0.07	15.08	0.52	2.26	1.74	2.44	0.967
	%diff	-12.13								
AUF_CFM	99	2.34	1.58	0.13	13.3	0.73	2.7	1.97	2.46	
CHK2_CTZ	99	2.07	1.44	0.07	15.29	0.55	2.19	1.64	2.55	0.967
	%diff	-11.66								
AUF_CFM	103	2.27	1.57	0.13	13.3	0.73	2.6	1.87	2.41	
CHK1_PDI	103	2.14	1.38	0.04	17.8	0.62	2.4	1.78	2.77	0.964
	%diff	-5.75	-12.10							
ALL	avg_%diff	-9.96								

Notes:
AUF CFM = gold fire assay Comision de Fomento Minero (original assay)
CHK1 CFM = gold fire assay Comision de Fomento Minero (check assay)
CHK_Sky = Skyline check assay
CHK1 CTZ = check assay #1 at Placer Dome's Cortez lab in Nevada
CHK2 CTZ = check assay #2 at Placer Dome's Cortez lab in Nevada
CHK1_PDI = check assay at Placer Dome's lab in Vancouver

Based on a review of relative difference plots for the check assay data shown in Table 19.10, it was determined that the original assays were conditionally biased. The mean gold grades for the original and various check assay programs were calculated at five different gold grade ranges and compared with one another. Table 19.11 summarizes percent difference between the two data sets for each grade range, which was the factor that was used to reduce about 2,300 assays contained in 44 1988 MRA drill holes that were assayed by CFM.

Table 19.11
1988 MRA Assay Adjustment Factors

Au Grade Range (g/t)	No. Samples	No. Meters	Au Reduction Factor
0.0 to 0.4	1,103	3,269	-35.22
0.4 to 0.8	729	2,171	-29.99
0.8 to 1.2	368	1,108	-16.65
1.2 to 4.0	105	319	-9.3
> 4.0	39	117	-0.4
Total	2,344	6,984	-10.0

19.1.8.2   MRA 1989 Campaign

Check assays for MRA's 1989 drilling campaign were done by the Placer Dome Research Center. These check assay data were examined and one anomalous sample was discarded from the study. In that sample the check assay value was 100 times greater than the original (i.e. 61 g/t vs. 0.65 g/t). It was believed that the check assay might have been a transcription error. Without the one anomalous sample, the 1989 MRA check assays had less than a 5% difference in mean grade than the original samples. Table 19.12 summarizes the check assay statistics for the 1989 MRA drilling.

Table 19.12
1989 MRA Check Assay Comparison

	Valid N	Mean gpt	Med	Min	Max	Lower Quartile	Upper Quartile	Quartile Range	Std Dev	Corr Coef
AUF	266	2.04	0.90	0	39	0.4	1.8	1.4	3.99	
CHK1_PDI	266	1.92	0.83	0.01	35.4	0.34	1.69	1.35	3.78	0.968
	%diff	-5.88								
AUF	266	2.04	0.90	0	39	0.4	1.8	1.4	3.99	
CHK2_PDI	266	1.96	0.83	0.01	38.2	0.35	1.7	1.35	4.11	0.987
	%diff	-3.90								
ALL	avg_%diff	-4.89								

Given the closer agreement between the check assays and the original data it was decided that the 1989 MRA assays would not be factored.

19.1.8.3   Kennecott Campaign

Data analysis in the 1997 Placer Dome feasibility study indicated that the Kennecott and MRA reverse circulation assays were significantly higher than the nearby Placer Dome core hole assays. A review of the 1997 work showed that the comparison of drilling type and campaign was done on samples that were up to 12 meters apart for the MRA data and up to 9 meters away for the Kennecott data. Given the variability of insitu grades in an epithermal hosted gold system the validity of the 1997 study is suspect given the separation distance of the sample pairs. The measures of spatial correlation of Au grades show that at these separation distances, a large proportion of the total variability of gold grade had already been attained (0.65); thus, there is no real reason to believe that samples of different drill types compared at these distances should be well correlated. The 1988 MRA samples were adjusted according to the factors shown in Table 19.11.

Check and duplicate assay data for some of the Kennecott drill holes were reviewed. These data came only from the work completed by Kennecott in 1993; earlier check assay data were not available. The original Kennecott assays were completed at Skyline Labs in Tucson, Arizona (SKY) and check assays were done at Rocky Mountain Geochemical in Salt Lake City, Utah (ROCK). A total of 90 check assays and 401 duplicate assays comprise the data. The check assays show good agreement with a correlation coefficient of 0.988 and a regressed line of slope 1.05. Descriptive statistics for the check and duplicate sample assays are presented in Table 19.13 and Table 19.14.

Table 19.13
1993 Kennecott Check Assay Comparison

	Valid N	Mean gpt	Median	Min	Max	Lower Quartile	Upper Quartile	Quartile Range	Std.Dev.	Corr Coef
GPT_SKY	90	2.66	1.58	0.34	14.57	1.03	3.26	2.23	2.82	
GPT_ROCK	90	2.76	1.63	0.24	15.87	1.03	3.15	2.12	2.99	0.988
	%diff	3.72	3.16							

The mean and median Rocky Mountain check assays were slightly higher grade than the initial Skyline assays (i.e. 3.72% and 3.16%). A relative difference plot does not show any bias to grades less than approximately 3 g/t. Above that level, the Rocky Mountain check assays show a slightly higher average grade (+4%) to the original Skyline assays.

Table 19.14
1993 Kennecott Duplicate Assay Comparison

	Valid N	Mean	Median	Min	Max	Lower Quartile	Upper Quartile	Quartile Range	Std.Dev.	Corr Coef
Split1	402	0.88	0.27	0.03	32.91	0.07	0.69	0.62	2.46	
Split2	402	0.89	0.28	0.00	35.80	0.11	0.69	0.59	2.69	0.922
	%diff	1.84	4.81							

Duplicate assays from the 1003 Kennecott drilling program also show good agreement with a correlation coefficient of 0.92, a regressed line of slope 1.01, and percentage differences at the mean and median of 1.84% and 4.81%. The relative difference plot for the duplicate assays shows a systematic high-grade bias of approximately 5% for the second split. Some of this difference is most likely the result of the natural short-scale variability (nugget effect).

Given the relatively close agreement between the check and duplicate assays for the 1993 Kennecott drilling program it was decided not to factor the assays.

19.1.8.4   PDI 1996 Phase 1 Campaign

During 1994, FSSI performed a preliminary check assay study and revealed that the SGS/XRAL assays that had been completed prior to May 1994 for Placer Dome's Phase 1 drilling campaign were 5 to 10% higher than a series of Bondar Clegg check assays. The FSSI study also showed that samples below 0.5 g/t Au were as much as 20% higher than the Bondar Clegg check assays, indicating a conditional bias. FSSI recommended that all Phase 1 samples be sent for re-assay and that higher-grade material be assayed using a gravimetric finish. No samples were sent for re-assay in 1994.

Approximately 8,500 Phase 1 samples were sent out for check assay in 1996 and comparisons between the original

and check assays confirmed the bias that was initially recognized in the 1994 FSSI study. The samples were re-assayed by Barringer Labs of Reno, Nevada. The mean gold grade for those samples averaged about 13% lower than the original SGS/XRAL assay. The check assays for those 8,500 samples were used as the final gold assay in the database, replacing the original SGS/XRAL values.

Because there were 34 Phase 1 assays (about 3,900 assays) that had not been re-assayed, it was decided to perform a statistical analysis of the Phase 1 assays and formulate appropriate "adjustment factors" for the remaining Phase 1 assays. The re-assayed Phase 1 assays were merged with the original SGS/XRAL assays and various statistical comparisons were made. Of these samples, 8,503 had assays above detection limits and were used in the comparative study. The basic descriptive statistics from each data source are presented in Table 19.15.

Table 19.15
Phase 1 Check Assays vs. Original Assays

	Valid N	Mean gpt	Med	Min	Max	Lower Quartile	Upper Quartile	Quartile Range	Std Dev	Corr Coef
Original	8503	0.846	0.510	0.0	168.0	0.210	1.06	0.850	2.273	
Check	8503	0.747	0.437	0.0	185.2	0.172	0.924	0.752	2.413	0.99
	%diff	13.2								

It was noted that the bias between the original SGS/XRAL assays and subsequent check assays was conditional with a more significant difference at lower grade thresholds. For this reason, the SGS/XRAL and Barringer assay pairs were subdivided into five grade range classes and the percent difference was calculated. Table 19.16 summarizes the percent difference between the two data sets for each grade range, which was the factor that was used to reduce the remaining Phase 1 assays for 34 drill holes.

Table 19.16
1996 PDI Phase 1 Assay Adjustment Factors

Au Grade Range (g/t)	No. Samples	No. Meters	Au Reduction Factor
0.0 to 0.4	1,517	2,312	-22.9
0.4 to 0.8	904	1,378	-15.1
0.8 to 1.2	415	632	-13.4
1.2 to 4.0	869	5,840	-10.0
>4.1	221	337	0.0
Total	3,926	10,499	-7.0

19.1.8.5   Resulting Assay Adjustments

Assays from 78 drill holes totaling about 6,300 meters of
drilling were factored downward. The mean grade for
these data was reduced by about 8 percent. Table 19.17
summarizes the effect of factoring the assays for the 1988
MRA and 1996 PDI Phase 1 drilling programs.

Table 19.17
PDI Phase 1 and 1988 MRA Assay Adjustment Results

Drilling Campaign	No. Holes	No. Assays	No. Meters	Unfactored		Factored		Percent Difference
				Au (g/t)	G * T (g/t-m)	Au (g/t)	G * T (g/t-m)	
Phase 1	34	3,925	5,981.72	1.350	8,077.53	1.255	7,508.61	-7.0%
1988 MRA	44	2,344	6,984.40	0.832	5,811.85	0.749	5,232.03	-10.0%
Total	78	6,269	12,966.12	1.071	13,889.38	0.983	12,740.64	-8.3%

19.1.9 Assay Statistics

Basic descriptive statistics were calculated for gold assays by
lithology, alteration, oxidation, and by sample type and are shown in
Table 19.18 through Table 19.21. The number of meters of data, the
length weighted mean grades, grade-thickness products, standard
deviations, and coefficient of variation are summarized at four gold
cutoff grades. The uncapped and capped statistics are also shown in
the various tables.

Similar statistics were calculated for silver and the summary by
lithology is shown in Table 19.22.   The copper assay statistics by
lithology are shown in Table 19.23. Table 19.24 shows the sulfur
assay statistics by lithology.

Table 19.18
Gold Assay Statistics – By Lithology

Unit	Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV
					Uncapped Statistics Above Cutoff					Capped Statistics Above Cutoff			
All	0.00	98,504	40%	0.91	89,325	4.1%	3.65	4.03	0.86	85,168	4.3%	2.07	2.40
	0.25	58,806	20%	1.46	85,661	7.8%	4.65	3.19	1.39	81,503	8.2%	2.55	1.84
	0.50	39,520	18%	1.99	78,672	14.4%	5.59	2.81	1.89	74,514	15.1%	2.99	1.58
	1.00	21,391	22%	3.08	65,835	73.7%	7.43	2.41	2.88	61,678	72.4%	3.78	1.31
Overburden	0.00	1,011	71%	0.32	328	8.6%	0.68	2.11	0.32	328	8.6%	0.68	2.11
	0.25	295	10%	1.02	300	11.0%	0.96	0.95	1.02	300	11.0%	0.96	0.95
	0.50	198	10%	1.33	264	20.3%	1.03	0.78	1.33	264	20.3%	1.03	0.78
	1.00	101	10%	1.95	197	60.1%	1.14	0.58	1.95	197	60.1%	1.14	0.58
PM	0.00	2,504	97%	0.04	110	56.9%	0.21	4.73	0.04	110	56.9%	0.21	4.73
	0.25	75	2%	0.64	47	13.5%	1.03	1.63	0.64	47	13.5%	1.03	1.63
	0.50	30	1%	1.10	33	15.8%	1.53	1.39	1.10	33	15.8%	1.53	1.39
	1.00	4	0%	4.29	15	13.8%	2.80	0.65	4.29	15	13.8%	2.80	0.65
TQ	0.00	10,588	55%	0.91	9,596	3.6%	3.24	3.57	0.86	9,144	3.8%	2.36	2.74
	0.25	4,780	11%	1.94	9,253	4.6%	4.62	2.38	1.84	8,801	4.8%	3.26	1.77
	0.50	3,583	13%	2.46	8,815	10.4%	5.23	2.12	2.33	8,362	10.9%	3.63	1.56
	1.00	2,216	21%	3.53	7,821	81.5%	6.42	1.82	3.33	7,368	80.6%	4.33	1.30
DF2	0.00	950	73%	0.24	224	18.9%	0.55	2.34	0.24	224	18.9%	0.55	2.34
	0.25	257	14%	0.71	182	21.8%	0.90	1.27	0.71	182	21.8%	0.90	1.27
	0.50	126	11%	1.05	133	30.5%	1.19	1.13	1.05	133	30.5%	1.19	1.13
	1.00	26	3%	2.51	65	28.8%	2.05	0.81	2.51	65	28.8%	2.05	0.81
DF3	0.00	9,938	37%	1.29	12,814	1.9%	4.14	3.21	1.23	12,225	2.0%	2.71	2.20
	0.25	6,304	13%	1.99	12,569	3.6%	5.07	2.54	1.90	11,981	3.8%	3.21	1.69
	0.50	5,039	20%	2.40	12,103	11.5%	5.59	2.33	2.29	11,515	12.1%	3.49	1.53
	1.00	3,002	30%	3.54	10,629	83.0%	7.02	1.98	3.34	10,041	82.1%	4.20	1.26
DF4	0.00	6,860	69%	0.34	2,330	20.4%	1.18	3.48	0.34	2,327	20.5%	1.17	3.44
	0.25	2,156	17%	0.86	1,854	17.1%	2.01	2.34	0.86	1,851	17.2%	1.98	2.31
	0.50	1,001	9%	1.45	1,455	18.2%	2.84	1.95	1.45	1,451	18.3%	2.79	1.93
	1.00	372	5%	2.76	1,029	44.2%	4.34	1.57	2.76	1,026	44.1%	4.27	1.55
RF	0.00	29,156	35%	1.01	29,540	4.0%	3.27	3.23	0.97	28,312	4.1%	2.27	2.34
	0.25	19,069	23%	1.49	28,367	8.0%	3.96	2.66	1.42	27,140	8.4%	2.70	1.89
	0.50	12,508	19%	2.08	26,000	13.5%	4.78	2.30	1.98	24,773	14.1%	3.19	1.61
	1.00	6,871	24%	3.20	22,015	74.5%	6.23	1.94	3.03	20,788	73.4%	4.01	1.33
Volc-1	0.00	6,299	27%	1.00	6,324	3.4%	2.99	2.98	0.97	6,126	3.5%	2.05	2.11
	0.25	4,609	24%	1.33	6,110	8.8%	3.44	2.60	1.28	5,912	9.1%	2.32	1.81
	0.50	3,067	23%	1.81	5,552	16.4%	4.13	2.28	1.75	5,354	16.9%	2.73	1.57
	1.00	1,598	25%	2.82	4,515	71.4%	5.54	1.96	2.70	4,317	70.5%	3.52	1.30
Volc-2	0.00	8,007	23%	0.92	7,383	3.6%	1.94	2.11	0.91	7,313	3.7%	1.75	1.91
	0.25	6,192	28%	1.15	7,113	10.9%	2.16	1.88	1.14	7,043	11.0%	1.93	1.70
	0.50	3,971	24%	1.59	6,307	18.6%	2.59	1.63	1.57	6,237	18.7%	2.30	1.46
	1.00	2,029	25%	2.43	4,938	66.9%	3.41	1.40	2.40	4,868	66.6%	2.98	1.24
Volc-3	0.00	3,060	55%	0.52	1,600	7.9%	1.82	3.48	0.51	1,567	8.1%	1.56	3.04
	0.25	1,383	22%	1.06	1,473	15.1%	2.60	2.44	1.04	1,440	15.4%	2.20	2.12
	0.50	715	12%	1.72	1,231	16.1%	3.49	2.03	1.67	1,198	16.4%	2.92	1.75
	1.00	340	11%	2.86	973	60.9%	4.81	1.68	2.77	941	60.0%	3.96	1.43
Volc-4	0.00	17,461	25%	1.04	18,246	3.2%	5.87	5.62	0.95	16,666	3.5%	1.87	1.96
	0.25	13,096	24%	1.35	17,667	8.4%	6.75	5.00	1.23	16,087	9.2%	2.09	1.70
	0.50	8,917	25%	1.81	16,133	16.6%	8.14	4.50	1.63	14,553	18.1%	2.43	1.49
	1.00	4,633	27%	2.83	13,111	71.9%	11.20	3.96	2.49	11,531	69.2%	3.13	1.26
Undefined	0.00	2,670	78%	0.31	832	12.7%	1.23	3.94	0.31	826	12.8%	1.16	3.75
	0.25	590	8%	1.23	726	9.4%	2.39	1.95	1.22	720	9.5%	2.24	1.83
	0.50	365	6%	1.77	647	14.4%	2.91	1.64	1.76	641	14.5%	2.71	1.54
	1.00	199	7%	2.65	527	63.4%	3.72	1.40	2.62	522	63.2%	3.43	1.31

Table 19.19
Gold Assay Statistics – By Alteration

			Uncapped Statistics Above Cutoff					Capped Statistics Above Cutoff				
Unit	Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All	0.00	98,504	40%	0.91	89,325	4.1%	3.65	4.03	0.86	85,168	4.3%	2.07	2.40
	0.25	58,806	20%	1.46	85,661	7.8%	4.65	3.19	1.39	81,503	8.2%	2.55	1.84
	0.50	39,520	18%	1.99	78,672	14.4%	5.59	2.81	1.89	74,514	15.1%	2.99	1.58
	1.00	21,391	22%	3.08	65,835	73.7%	7.43	2.41	2.88	61,678	72.4%	3.78	1.31
Arg-1	0.00	33,389	78%	0.22	7,272	25.5%	0.84	3.85	0.22	7,191	25.7%	0.70	3.26
	0.25	7,489	14%	0.72	5,421	21.6%	1.67	2.31	0.71	5,340	21.9%	1.37	1.92
	0.50	2,955	6%	1.30	3,847	18.3%	2.55	1.96	1.27	3,766	18.5%	2.05	1.61
	1.00	993	3%	2.54	2,519	34.6%	4.12	1.63	2.46	2,438	33.9%	3.22	1.31
Arg-2	0.00	17,296	22%	0.94	16,176	3.2%	2.43	2.60	0.91	15,819	3.3%	1.90	2.07
	0.25	13,479	26%	1.16	15,655	10.3%	2.71	2.33	1.14	15,299	10.6%	2.10	1.85
	0.50	8,945	28%	1.56	13,986	21.1%	3.25	2.08	1.52	13,629	21.6%	2.48	1.63
	1.00	4,115	24%	2.57	10,571	65.4%	4.59	1.79	2.48	10,215	64.6%	3.42	1.38
Silicified	0.00	29,932	25%	1.10	32,943	2.8%	5.24	4.77	1.02	30,419	3.0%	2.16	2.13
	0.25	22,414	24%	1.43	32,017	7.9%	6.03	4.22	1.32	29,493	8.6%	2.42	1.84
	0.50	15,319	24%	1.92	29,412	15.5%	7.24	3.77	1.76	26,888	16.8%	2.82	1.61
	1.00	8,095	27%	3.00	24,297	73.8%	9.83	3.27	2.69	21,773	71.6%	3.64	1.35
Vuggy Silica	0.00	17,773	13%	1.85	32,929	1.1%	4.32	2.33	1.79	31,733	1.1%	3.13	1.75
	0.25	15,425	18%	2.11	32,568	3.5%	4.59	2.17	2.03	31,372	3.6%	3.29	1.62
	0.50	12,300	23%	2.56	31,427	9.0%	5.04	1.97	2.46	30,231	9.4%	3.56	1.45
	1.00	8,187	46%	3.47	28,449	86.4%	5.97	1.72	3.33	27,252	85.9%	4.10	1.23
Undefined	0.00	112	100%	0.05	6	100.0%	0.04	0.75	0.05	6	100.0%	0.04	0.75
	0.25	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
	0.50	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00

Table 19.20
Gold Assay Statistics – By Oxidation

Unit	Cutoff (g/t)	Uncapped Statistics Above Cutoff							Capped Statistics Above Cutoff				
		Total Meters	Inc. Percent	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV
All	0.00	98,504	40%	0.91	89,325	4.1%	3.65	4.03	0.86	85,168	4.3%	2.07	2.40
	0.25	58,806	20%	1.46	85,661	7.8%	4.65	3.19	1.39	81,503	8.2%	2.55	1.84
	0.50	39,520	18%	1.99	78,672	14.4%	5.59	2.81	1.89	74,514	15.1%	2.99	1.58
	1.00	21,391	22%	3.08	65,835	73.7%	7.43	2.41	2.88	61,678	72.4%	3.78	1.31
Oxide	0.00	7,582	62%	0.51	3,880	8.9%	2.86	5.59	0.47	3,527	9.8%	1.64	3.53
	0.25	2,878	16%	1.23	3,535	11.4%	4.55	3.70	1.11	3,182	12.5%	2.54	2.30
	0.50	1,668	12%	1.85	3,094	16.3%	5.90	3.18	1.64	2,740	17.9%	3.23	1.97
	1.00	751	10%	3.28	2,462	63.5%	8.57	2.62	2.81	2,109	59.8%	4.55	1.62
Mixed-1	0.00	16,799	26%	1.31	21,991	2.2%	3.53	2.70	1.27	21,254	2.3%	2.58	2.04
	0.25	12,409	20%	1.73	21,504	5.6%	4.02	2.32	1.67	20,768	5.8%	2.90	1.73
	0.50	9,052	21%	2.24	20,281	11.7%	4.61	2.06	2.16	19,544	12.2%	3.26	1.51
	1.00	5,466	33%	3.24	17,698	80.5%	5.71	1.76	3.10	16,961	79.8%	3.92	1.26
Mixed-2	0.00	14,939	23%	1.33	19,905	1.8%	3.99	2.99	1.27	18,989	1.9%	2.52	1.98
	0.25	11,482	20%	1.70	19,553	5.5%	4.48	2.63	1.62	18,637	5.7%	2.78	1.71
	0.50	8,502	22%	2.17	18,463	11.7%	5.13	2.36	2.06	17,547	12.3%	3.11	1.51
	1.00	5,216	35%	3.09	16,128	81.0%	6.38	2.06	2.92	15,212	80.1%	3.72	1.28
Sulfide	0.00	59,072	46%	0.74	43,543	5.7%	3.67	4.98	0.70	41,392	6.0%	1.78	2.54
	0.25	32,037	20%	1.28	41,067	9.7%	4.92	3.84	1.21	38,916	10.2%	2.29	1.89
	0.50	20,297	18%	1.81	36,834	16.7%	6.12	3.37	1.71	34,683	17.6%	2.76	1.62
	1.00	9,958	17%	2.97	29,547	67.9%	8.59	2.89	2.75	27,396	66.2%	3.66	1.33
Undefined	0.00	112	100%	0.05	6	100.0%	0.04	0.75	0.05	6	100.0%	0.04	0.75
	0.25	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
	0.50	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00
	1.00	0	0%	0.00	0	0.0%	0.00	0.00	0.00	0	0.0%	0.00	0.00

Table 19.21
Gold Assay Statistics – By Sample Type

Unit	Cutoff (g/t)	Total Meters	Inc. Percent	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Au (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV
			Uncapped Statistics Above Cutoff						Capped Statistics Above Cutoff				
Totals	0.00	98,504	40%	0.91	89,325	4.1%	3.65	4.03	0.86	85,168	4.3%	2.07	2.40
	0.25	58,806	20%	1.46	85,661	7.8%	4.65	3.19	1.39	81,503	8.2%	2.55	1.84
	0.50	39,520	18%	1.99	78,672	14.4%	5.59	2.81	1.89	74,514	15.1%	2.99	1.58
	1.00	21,391	22%	3.08	65,835	73.7%	7.43	2.41	2.88	61,678	72.4%	3.78	1.31
Surface Core	0.00	27,491	39%	0.91	25,107	4.3%	5.23	5.73	0.83	22,902	4.7%	1.97	2.37
	0.25	16,788	21%	1.43	24,020	8.2%	6.64	4.64	1.30	21,815	9.0%	2.41	1.85
	0.50	11,025	19%	1.99	21,959	14.5%	8.14	4.09	1.79	19,753	15.9%	2.85	1.59
	1.00	5,849	21%	3.13	18,319	73.0%	11.05	3.53	2.76	16,114	70.4%	3.65	1.32
U/G Core	0.00	1,565	6%	2.26	3,531	0.4%	4.52	2.00	2.18	3,413	0.4%	3.72	1.71
	0.25	1,464	10%	2.40	3,518	1.7%	4.64	1.93	2.32	3,400	1.7%	3.81	1.64
	0.50	1,306	23%	2.65	3,459	7.7%	4.85	1.83	2.56	3,340	7.9%	3.96	1.55
	1.00	943	60%	3.38	3,188	90.3%	5.54	1.64	3.26	3,069	89.9%	4.47	1.37
RVC	0.00	62,319	45%	0.78	48,639	4.9%	2.43	3.11	0.76	47,517	5.1%	1.88	2.47
	0.25	34,434	19%	1.34	46,232	9.0%	3.16	2.35	1.31	45,110	9.2%	2.39	1.83
	0.50	22,402	17%	1.87	41,858	15.7%	3.81	2.04	1.82	40,736	16.1%	2.84	1.56
	1.00	11,636	19%	2.94	34,226	70.4%	5.06	1.72	2.84	33,104	69.7%	3.65	1.28
U/G Channels	0.00	1,491	8%	1.81	2,697	0.7%	5.02	2.77	1.68	2,509	0.7%	3.12	1.85
	0.25	1,376	17%	1.95	2,679	3.6%	5.20	2.67	1.81	2,491	3.8%	3.21	1.77
	0.50	1,126	28%	2.29	2,583	11.2%	5.69	2.48	2.13	2,395	12.1%	3.47	1.63
	1.00	705	47%	3.24	2,280	84.5%	7.02	2.17	2.97	2,092	83.4%	4.16	1.40
Airtrack	0.00	5,422	16%	1.60	8,661	1.6%	4.55	2.85	1.50	8,137	1.7%	2.98	1.98
	0.25	4,531	20%	1.88	8,522	4.6%	4.93	2.62	1.77	7,998	4.9%	3.19	1.81
	0.50	3,455	25%	2.35	8,127	11.3%	5.56	2.37	2.20	7,603	12.0%	3.54	1.61
	1.00	2,073	38%	3.45	7,151	82.6%	6.97	2.02	3.20	6,627	81.4%	4.29	1.34
U/G Muck	0.00	215	1%	3.21	689	0.0%	3.80	1.18	3.21	689	0.0%	3.80	1.18
	0.25	213	4%	3.23	689	0.4%	3.80	1.18	3.23	689	0.4%	3.80	1.18
	0.50	206	9%	3.34	686	2.3%	3.83	1.15	3.34	686	2.3%	3.83	1.15
	1.00	185	86%	3.62	671	97.3%	3.93	1.09	3.62	671	97.3%	3.93	1.09

Table 19.22
Silver Assay Statistics – By Lithology

Unit	Cutoff (g/t)	Total Meters	Inc. Percent	Mean Ag (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV	Mean Ag (g/t)	grd-thk (g/t-m)	Inc. Percent	Std. Dev.	CV
					Uncapped Statistics Above Cutoff					Capped Statistics Above Cutoff			
All	0.00	98,504	60%	0.86	85,168	12.5%	2.07	2.40	3.02	228,618	1.5%	11.54	3.83
	0.50	39,520	18%	1.89	74,514	15.1%	2.99	1.58	4.49	225,213	2.4%	13.96	3.11
	1.00	21,391	19%	2.88	61,678	43.4%	3.78	1.31	5.40	219,612	27.3%	15.36	2.84
	5.00	2,322	2%	10.66	24,741	29.0%	7.54	0.71	14.43	157,193	68.8%	27.69	1.92
Overburden	0.00	1,011	80%	0.32	328	19.6%	0.68	2.11	0.68	555	13.7%	1.48	2.19
	0.50	198	10%	1.33	264	20.3%	1.03	0.78	1.78	479	11.1%	2.20	1.23
	1.00	101	10%	1.95	197	54.5%	1.14	0.58	2.56	417	50.2%	2.54	0.99
	5.00	3	0%	6.00	18	5.6%	0.17	0.03	8.27	139	25.0%	4.01	0.48
PM	0.00	2,504	99%	0.04	110	70.5%	0.21	4.73	0.39	874	33.3%	1.54	3.96
	0.50	30	1%	1.10	33	15.8%	1.53	1.39	2.45	583	7.2%	4.20	1.72
	1.00	4	0%	4.29	15	1.6%	2.80	0.65	3.61	520	29.5%	5.08	1.41
	5.00	2	0%	6.68	13	12.1%	0.85	0.13	13.25	262	30.0%	8.50	0.64
TQ	0.00	10,588	66%	0.86	9,144	8.5%	2.36	2.74	2.20	16,921	2.4%	10.51	4.77
	0.50	3,583	13%	2.33	8,362	10.9%	3.63	1.56	3.75	16,520	2.9%	13.68	3.64
	1.00	2,216	18%	3.33	7,368	40.8%	4.33	1.30	4.58	16,025	31.4%	15.24	3.33
	5.00	339	3%	10.74	3,640	39.8%	7.23	0.67	14.00	10,716	63.3%	30.72	2.19
DF2	0.00	950	87%	0.24	224	40.7%	0.55	2.34	0.87	522	5.4%	3.17	3.65
	0.50	126	11%	1.05	133	30.5%	1.19	1.13	2.39	494	10.5%	5.06	2.12
	1.00	26	2%	2.51	65	15.8%	2.05	0.81	3.84	439	34.1%	6.46	1.68
	5.00	5	0%	6.51	29	13.1%	0.00	0.00	15.58	261	50.0%	10.78	0.69
DF3	0.00	9,938	49%	1.23	12,225	5.8%	2.71	2.20	4.60	38,602	1.0%	18.32	3.98
	0.50	5,039	20%	2.29	11,515	12.1%	3.49	1.53	6.71	38,216	1.1%	21.93	3.27
	1.00	3,002	25%	3.34	10,041	41.7%	4.20	1.26	7.66	37,791	19.4%	23.42	3.06
	5.00	484	5%	10.22	4,944	40.4%	6.90	0.68	19.66	30,313	78.5%	39.27	2.00
DF4	0.00	6,860	85%	0.34	2,327	37.6%	1.17	3.44	1.49	7,508	4.4%	3.88	2.60
	0.50	1,001	9%	1.45	1,451	18.3%	2.79	1.93	2.44	7,178	7.6%	4.86	1.99
	1.00	372	5%	2.76	1,026	26.2%	4.27	1.55	3.26	6,605	43.1%	5.67	1.74
	5.00	34	0%	12.13	416	17.9%	9.69	0.80	10.90	3,367	44.8%	11.72	1.07
RF	0.00	29,156	57%	0.97	28,312	12.5%	2.27	2.34	2.89	69,920	1.4%	8.81	3.05
	0.50	12,508	19%	1.98	24,773	14.1%	3.19	1.61	4.05	68,907	2.4%	10.30	2.54
	1.00	6,871	21%	3.03	20,788	42.7%	4.01	1.33	4.75	67,232	32.5%	11.15	2.35
	5.00	794	3%	10.93	8,686	30.7%	7.85	0.72	12.47	44,507	63.7%	20.27	1.63
Volc-1	0.00	6,299	51%	0.97	6,126	12.6%	2.05	2.11	3.29	11,866	0.3%	9.73	2.96
	0.50	3,067	23%	1.75	5,354	16.9%	2.73	1.57	4.04	11,829	2.7%	10.66	2.64
	1.00	1,598	23%	2.70	4,317	44.6%	3.52	1.30	4.90	11,504	31.5%	11.75	2.40
	5.00	142	2%	11.18	1,586	25.9%	7.28	0.65	14.30	7,767	65.5%	21.87	1.53
Volc-2	0.00	8,007	50%	0.91	7,313	14.7%	1.75	1.91	4.66	23,242	0.3%	8.30	1.78
	0.50	3,971	24%	1.57	6,237	18.7%	2.30	1.46	5.36	23,181	1.4%	8.70	1.62
	1.00	2,029	24%	2.40	4,868	47.8%	2.98	1.24	6.07	22,867	24.2%	9.11	1.50
	5.00	131	2%	10.51	1,373	18.8%	7.61	0.72	12.99	17,246	74.2%	12.62	0.97
Volc-3	0.00	3,060	77%	0.51	1,567	23.5%	1.56	3.04	0.97	2,342	9.8%	2.90	2.99
	0.50	715	12%	1.67	1,198	16.4%	2.92	1.75	2.07	2,113	13.8%	4.22	2.03
	1.00	340	10%	2.77	941	35.6%	3.96	1.43	3.49	1,789	34.5%	5.59	1.60
	5.00	38	1%	10.16	383	24.5%	8.62	0.85	12.52	981	41.9%	10.14	0.81
Volc-4	0.00	17,461	49%	0.95	16,666	12.7%	1.87	1.96	3.91	52,331	0.6%	16.36	4.18
	0.50	8,917	25%	1.63	14,553	18.1%	2.43	1.49	5.06	52,037	2.2%	18.51	3.66
	1.00	4,633	25%	2.49	11,531	48.4%	3.13	1.26	6.08	50,910	22.6%	20.37	3.35
	5.00	331	2%	10.48	3,463	20.8%	7.68	0.73	15.50	39,059	74.6%	35.35	2.28
Undefined	0.00	2,670	86%	0.31	826	22.3%	1.16	3.75	1.62	3,934	6.5%	5.55	3.42
	0.50	365	6%	1.76	641	14.5%	2.71	1.54	4.19	3,676	4.2%	8.65	2.06
	1.00	199	7%	2.62	522	40.2%	3.43	1.31	5.57	3,512	23.8%	9.87	1.77
	5.00	20	1%	9.47	189	22.9%	7.63	0.81	14.21	2,576	65.5%	15.20	1.07

Table 19.23
Copper Assay Statistics – By Lithology

Unit	Cutoff (ppm)	Uncapped Statistics Above Cutoff					Capped Statistics Above Cutoff		
		Total Meters	Inc. Percent	Mean Cu (ppm)	Std. Dev.	CV	Mean Cu (ppm)	Std. Dev.	CV
All	0	62,327	34%	221	1,239	5.61	207	804	3.87
	50	41,230	54%	320	1,514	4.74	299	975	3.26
	250	7,755	5%	1,225	3,343	2.73	1,115	2,056	1.84
	370	4,564	7%	1,875	4,238	2.26	1,687	2,527	1.50
Overburden	0	639	75%	39	36	0.91	39	36	0.91
	50	162	25%	85	42	0.50	85	42	0.50
	250	2	0%	282	0	0.00	282	0	0.00
	370	0	0%	0	0	0.00	0	0	0.00
PM	0	1,077	83%	32	61	1.88	32	61	1.88
	50	188	16%	102	120	1.17	102	120	1.17
	250	11	1%	455	303	0.67	455	303	0.67
	370	3	0%	910	220	0.24	910	220	0.24
TQ	0	6,076	39%	120	221	1.85	120	221	1.85
	50	3,728	52%	174	269	1.54	174	269	1.54
	250	575	4%	578	512	0.89	578	512	0.89
	370	310	5%	822	596	0.72	822	596	0.72
DF2	0	510	56%	66	109	1.65	66	109	1.65
	50	227	42%	110	151	1.37	110	151	1.37
	250	14	1%	535	405	0.76	535	405	0.76
	370	6	1%	835	454	0.54	835	454	0.54
DF3	0	6,414	32%	271	2,202	8.12	226	843	3.74
	50	4,379	54%	381	2,658	6.97	314	1,009	3.21
	250	903	5%	1,443	5,728	3.97	1,118	2,026	1.81
	370	589	9%	2,052	7,015	3.42	1,553	2,397	1.54
DF4	0	3,688	22%	600	2,245	3.74	557	1,648	2.96
	50	2,885	50%	759	2,516	3.32	703	1,837	2.61
	250	1,031	6%	1,930	3,946	2.04	1,775	2,766	1.56
	370	795	22%	2,412	4,377	1.82	2,211	3,013	1.36
RF	0	18,987	27%	272	1,297	4.77	257	935	3.64
	50	13,811	57%	363	1,511	4.17	341	1,084	3.18
	250	2,928	5%	1,299	3,105	2.39	1,199	2,145	1.79
	370	1,896	10%	1,844	3,748	2.03	1,688	2,534	1.50
Volc-1	0	3,518	37%	90	88	0.98	90	88	0.98
	50	2,203	58%	128	92	0.72	128	92	0.72
	250	179	5%	310	180	0.58	310	180	0.58
	370	17	0%	618	475	0.77	618	475	0.77
Volc-2	0	4,834	23%	275	1,086	3.94	266	905	3.40
	50	3,738	60%	348	1,226	3.53	335	1,019	3.04
	250	819	8%	1,168	2,447	2.10	1,111	1,990	1.79
	370	444	9%	1,906	3,138	1.65	1,801	2,502	1.39
Volc-3	0	2,103	63%	69	322	4.64	69	322	4.64
	50	788	35%	143	517	3.62	143	517	3.62
	250	51	2%	916	1,858	2.03	916	1,858	2.03
	370	16	1%	2,300	2,859	1.24	2,300	2,859	1.24
Volc-4	0	13,092	36%	128	395	3.08	128	395	3.08
	50	8,384	56%	186	484	2.60	186	484	2.60
	250	1,095	5%	644	1,237	1.92	644	1,237	1.92
	370	403	3%	1,244	1,895	1.52	1,244	1,895	1.52
Undefined	0	1,390	47%	264	1,872	7.08	222	1,120	5.05
	50	738	43%	478	2,550	5.34	397	1,515	3.81
	250	147	5%	1,987	5,460	2.75	1,583	3,124	1.97
	370	84	6%	3,260	6,964	2.14	2,553	3,864	1.51

Table 19.24
Total Sulfur Assay Statistics – By Lithology

Unit	Cutoff (%)	Total Meters	Inc. Percent	Mean S (%)	Std. Dev.	CV
			Uncapped Statistics Above Cutoff			
All	0.0	36,275	15%	3.73	2.83	0.76
	0.5	30,918	7%	4.34	2.63	0.61
	1.0	28,436	48%	4.65	2.50	0.54
	5.0	10,902	30%	6.98	2.33	0.33
Overburden	0.0	128	74%	0.53	0.79	1.48
	0.5	34	18%	1.31	1.22	0.93
	1.0	11	8%	2.70	1.36	0.50
	5.0	0	0%	0.00	0.00	0.00
PM	0.0	0	0%	0.00	0.00	0.00
	0.5	0	0%	0.00	0.00	0.00
	1.0	0	0%	0.00	0.00	0.00
	5.0	0	0%	0.00	0.00	0.00
TQ	0.0	4,016	13%	3.89	2.56	0.66
	0.5	3,489	8%	4.44	2.29	0.52
	1.0	3,187	42%	4.79	2.07	0.43
	5.0	1,495	37%	6.41	1.62	0.25
DF2	0.0	355	41%	2.00	2.01	1.00
	0.5	210	6%	3.26	1.71	0.53
	1.0	187	40%	3.57	1.56	0.44
	5.0	44	12%	5.54	0.69	0.12
DF3	0.0	4,253	7%	4.57	2.55	0.56
	0.5	3,950	5%	4.90	2.33	0.48
	1.0	3,740	40%	5.14	2.17	0.42
	5.0	2,046	48%	6.55	1.72	0.26
DF4	0.0	1,701	0%	6.67	3.30	0.49
	0.5	1,701	1%	6.67	3.30	0.49
	1.0	1,689	31%	6.71	3.27	0.49
	5.0	1,154	68%	8.11	3.02	0.37
RF	0.0	11,703	5%	4.00	2.09	0.52
	0.5	11,077	3%	4.22	1.94	0.46
	1.0	10,676	64%	4.35	1.85	0.43
	5.0	3,149	27%	6.49	1.65	0.25
Volc-1	0.0	2,487	34%	2.11	2.38	1.12
	0.5	1,632	15%	3.12	2.38	0.76
	1.0	1,257	33%	3.84	2.26	0.59
	5.0	425	17%	6.44	1.51	0.23
Volc-2	0.0	3,708	10%	4.49	3.84	0.86
	0.5	3,328	6%	4.98	3.76	0.75
	1.0	3,093	48%	5.30	3.70	0.70
	5.0	1,320	36%	8.61	3.36	0.39
Volc-3	0.0	421	28%	2.55	2.45	0.96
	0.5	302	10%	3.46	2.31	0.67
	1.0	261	45%	3.90	2.19	0.56
	5.0	72	17%	6.58	1.80	0.27
Volc-4	0.0	7,370	31%	2.40	2.57	1.07
	0.5	5,115	11%	3.37	2.53	0.75
	1.0	4,268	42%	3.90	2.45	0.63
	5.0	1,154	16%	7.00	2.26	0.32
Undefined	0.0	133	39%	3.46	4.57	1.32
	0.5	81	10%	5.59	4.79	0.86
	1.0	67	18%	6.58	4.67	0.71
	5.0	43	32%	8.75	4.50	0.51

19.1.10　　　Grade Capping

High-grade outlier values were identified for the gold, silver, and total copper populations. Thresholds for capping these high-grade values were determined by examining cumulative probability distribution plots of the raw assays for each metal. In addition, the distribution of grades was examined by deciles to gauge how much metal was contained for each segment of the population.

Gold

Raw gold assay grades were found to be well behaved below 35 g/t, but became somewhat erratically distributed above that grade. Figure 19.9 is a histogram constructed from raw gold assays that were transformed using the cumulative normal distribution function. Assay above 35 g/t are believed to be erratically distributed and those values should be reduced to minimize the over-estimation of gold resources.

The distribution of raw gold assays was also analyzed by decile ranges by sorting the grades by ascending order and summarizing basic descriptive statistics of each decile bin. The number of samples in each decile along with other statistical parameters is shown in Table 19.25. This table shows that about 57 percent of the gold metal is contained in 10 percent of the samples. Table 19.25 also breaks down the data by one percent increments and shows that about 23 percent of the gold is contained in one percent of the data.

Table 19.25
Gold Deciles

Decile Range	No. of Samples	Min Grade	Mean Grade	Max Grade	G*T Product	% of Total
0 - 10	5,860	0.00	0.01	0.03	130	0.1
10 - 20	5,860	0.03	0.04	0.06	343	0.4
20 - 30	5,860	0.06	0.10	0.14	963	1.1
30 - 40	5,860	0.14	0.18	0.23	1,837	2.1
40 - 50	5,860	0.23	0.28	0.34	2,787	3.1
50 - 60	5,860	0.34	0.41	0.48	4,137	4.6
60 - 70	5,860	0.48	0.58	0.69	5,891	6.6
70 - 80	5,860	0.69	0.86	1.05	8,703	9.7
80 - 90	5,860	1.05	1.38	1.85	13,931	15.6
90 - 100	5,860	1.85	5.07	325.50	50,604	56.7
Total	58,600	0.00	0.91	325.50	89,325	100.0

Decile Range	No. of Samples	Min Grade	Mean Grade	Max Grade	G*T Product	% of Total
90 - 91	586	1.85	1.92	2.00	1,933	2.2
91 - 92	586	2.00	2.08	2.17	2,134	2.4
92 - 93	586	2.18	2.27	2.38	2,335	2.6
93 - 94	586	2.38	2.51	2.64	2,481	2.8
94 - 95	586	2.64	2.80	2.96	2,743	3.1
95 - 96	586	2.96	3.22	3.51	3,193	3.6
96 - 97	586	3.51	3.92	4.32	3,943	4.4
97 - 98	586	4.32	4.86	5.53	4,850	5.4
98 - 99	586	5.53	6.76	8.60	6,649	7.4
99 -100	586	8.60	20.86	325.50	20,343	22.8
Sub-total	5,860	1.85	5.07	325.50	50,604	56.7

Table 19.26 summarizes metal loss at various potential capping limits. The number of raw samples that would be capped, mean grade after capping, standard deviation, coefficient of variation, and metal loss are shown in the table. The last column shows how much of the total gold metal in the entire population is contained in samples above each cutoff.

Table 19.26
Gold Capping Sensitivity

Cap Grade	No. Capped	Mean Au (g/t)	Std. Dev.	Coefficient of Variation	% Metal Loss	% Metal > Cap Grade
None	0	0.907	3.653	4.03	0.0%	100.0%
51	29	0.876	2.291	2.61	3.4%	6.2%
49	31	0.875	2.268	2.59	3.5%	6.4%
47	33	0.874	2.244	2.57	3.6%	6.6%
45	36	0.873	2.220	2.54	3.7%	6.8%
43	42	0.872	2.194	2.52	3.9%	7.3%
41	46	0.870	2.165	2.49	4.1%	7.6%
39	50	0.868	2.136	2.46	4.2%	7.9%
37	55	0.867	2.104	2.43	4.4%	8.3%
35	*60*	*0.865*	*2.071*	*2.40*	*4.7%*	*8.6%*
33	69	0.862	2.035	2.36	4.9%	9.1%
31	78	0.860	1.997	2.32	5.2%	9.7%
29	89	0.857	1.954	2.28	5.5%	10.4%
27	99	0.854	1.909	2.24	5.8%	10.9%
25	105	0.850	1.863	2.19	6.2%	11.2%

Table 19.27 lists 60 gold assays that were cut back to 35 g/t. The assays are sorted be decreasing original gold grade. The source of each sample is also shown in the table. The 10 airtrack assays shown in Table 19.27 were not used for estimating gold resources.

Table 19.27
List of Capped Gold Assays

DH-ID	From	To	Length (m)	Au (g/t)	Data Type	DH-ID	From	To	Length (m)	Au (g/t)	Data Type
98EI014	59.66	64.77	5.11	325.50	Surface Core	K-81	129.54	131.06	1.52	50.40	RVC
PD-4	124.97	126.49	1.52	185.21	Surface Core	P-58	94.49	96.01	1.52	48.30	RVC
K-15	169.16	170.69	1.53	144.69	RVC	NOPAL-C	93.66	95.18	1.52	48.20	U/G Channel
P-19	134.11	135.64	1.53	143.20	RVC	PDM-83	45.72	47.24	1.52	46.00	Surface Core
PD-36	45.72	47.24	1.52	142.90	Surface Core	NP03-06	70.37	71.90	1.53	45.96	U/G Core
P-33	19.81	21.34	1.53	134.02	Airtrack	K-1	109.73	111.25	1.52	45.26	RVC
96PM018	100.58	102.11	1.53	126.00	RVC	M-6	123.00	126.00	3.00	44.92	Surface Core
PD-36	7.01	7.62	0.61	115.20	Surface Core	96PM022	65.53	67.06	1.53	44.80	RVC
NOPAL-C	89.09	90.00	0.91	108.00	U/G Channel	P-37	33.53	35.05	1.52	44.34	Airtrack
NOPAL-C	90.00	90.61	0.61	108.00	U/G Channel	K-63	147.83	149.35	1.52	43.89	RVC
M-134B	4.50	7.50	3.00	94.09	RVC	PD-36	94.49	96.01	1.52	43.80	Surface Core
P-37A	86.87	88.39	1.52	86.58	Airtrack	K-63	149.35	150.88	1.53	43.20	RVC
P-37	53.34	54.86	1.52	83.72	Airtrack	98EE004	111.80	113.00	1.20	42.75	Surface Core
PD-36	132.59	134.11	1.52	83.70	Surface Core	PD-6	32.00	33.53	1.53	42.00	Surface Core
PD-4	126.49	128.02	1.53	79.59	Surface Core	P-30	76.20	77.72	1.52	41.61	Airtrack
P-39	106.68	108.20	1.52	79.36	RVC	PD-50	117.35	118.87	1.52	41.28	Surface Core
K-63	144.78	146.30	1.52	75.43	RVC	K-72	156.97	158.50	1.53	40.46	RVC
P-23	91.44	92.96	1.52	75.30	Airtrack	PD-15	80.77	82.30	1.53	40.00	Surface Core
NOPALITO-C	36.00	37.50	1.50	73.08	U/G Channel	96WD103	70.10	71.63	1.53	39.93	RVC
96NE057	44.20	45.72	1.52	65.93	RVC	96PM046	16.76	18.29	1.53	39.70	RVC
P-37	39.62	41.15	1.53	65.84	Airtrack	M-116A	88.75	91.75	3.00	39.00	RVC
P-33	7.62	9.14	1.52	65.32	Airtrack	PD-45	164.59	166.12	1.53	38.88	Surface Core
NP03-04	47.12	48.64	1.52	65.14	U/G Core	K-28	208.79	210.31	1.52	38.74	RVC
P-37	32.00	33.53	1.53	61.32	Airtrack	96PM046	7.62	9.14	1.52	37.70	RVC
K-63	146.30	147.83	1.53	60.34	RVC	PD-43	178.31	179.83	1.52	37.30	Surface Core
NP03-01	26.00	27.53	1.53	55.48	U/G Core	P-84	141.73	143.26	1.53	36.36	RVC
PD-20	198.12	199.64	1.52	55.22	Surface Core	K-63	143.26	144.78	1.52	36.00	RVC
P-55A	124.97	126.49	1.52	52.85	RVC	P-30	71.63	73.15	1.52	35.74	Airtrack
M-134E	0.00	5.00	5.00	51.21	RVC	K-58	167.64	169.16	1.52	35.31	RVC
NP03-06	71.90	73.43	1.53	50.55	U/G Core	NPT03-06	64.35	66.95	2.60	35.21	U/G Core

Note: The 10 airtrack assays were not used for grade estimation.

Silver

Thirty-five silver assays were capped at 150 g/t based on a review of cumulative probability plots. The highest silver assay in the drill hole database was 1,148 g/t. Capping at 150 g/t removed about 3.9% of the available silver metal from the database. Table 19.28 summarizes the sensitivity of capping silver assays at various cutoff grades.

Table 19.28
Silver Capping Sensitivity

Cap Grade	No. Capped	Mean Ag (g/t)	Std. Dev.	Coefficient of Variation	% Metal Loss	% Metal > Cap Grade
None	0	3.0	11.5	3.83	0.0%	100.0%
225	24	2.9	8.5	2.89	2.6%	5.7%
210	25	2.9	8.3	2.85	2.8%	5.8%
195	27	2.9	8.2	2.79	3.0%	6.1%
180	31	2.9	8.0	2.74	3.3%	6.6%
165	34	2.9	7.8	2.69	3.6%	6.9%
150	*35*	*2.9*	*7.6*	*2.63*	*3.9%*	*7.0%*
135	42	2.9	7.4	2.57	4.2%	7.6%
120	52	2.9	7.2	2.51	4.7%	8.4%
105	69	2.9	7.0	2.43	5.2%	9.7%
90	86	2.8	6.7	2.35	5.9%	10.8%

Copper

Forty-five total copper assays were capped at 15,000 ppm based on a review of cumulative probability plots. The highest copper assay in the drill hole database was 93,200 ppm. Table 19.29 summarizes the sensitivity of capping copper assays at various cutoff grades.

Table 19.29
Copper Capping Sensitivity

Cap Grade	No. Capped	Mean Cu (ppm)	Std. Dev.	Coefficient of Variation	% Metal Loss	% Metal > Cap Grade
None	0	221	1,239	5.61	0.0%	100.0%
30,000	10	215	987	4.58	2.6%	5.8%
27,500	13	215	968	4.51	2.9%	6.7%
25,000	18	214	943	4.41	3.3%	8.0%
22,500	20	213	915	4.30	3.8%	8.6%
20,000	25	211	884	4.18	4.4%	9.6%
17,500	36	210	847	4.04	5.2%	11.7%
15,000	*45*	*207*	*804*	*3.87*	*6.2%*	*13.2%*
12,500	62	204	751	3.68	7.6%	15.7%
10,000	84	200	685	3.42	9.5%	18.3%

19.1.11   Assay Composting

19.1.11.1 Coordinate Limits of the Geologic Model

The drill hole assays were composited into three-meter lengths by the fixed length method. Most of the raw assay intervals were sampled on 1.52-meter lengths. The fixed length method of compositing differs from bench composting in that the drill hole intervals are systematically combined into three-meter lengths starting at the drill hole collar and continuing down the hole in lieu of creating composites relative to artificial horizontal datum. The fixed-length method assures that nearly all of the samples are of uniform length regardless of their orientation. Similarly the fixed-length method results in fewer short composites than bench compositing. Usually the last composite in a drill hole is the only one that is less than the selected three-meter length using the fixed-length method.

19.1.11.2 Dilution/Ore Loss

To determine what may be an optimal bench height a study was undertaken to quantify dilution and ore loss by looking at composites of various lengths. Ten separate drill hole composites were generated with lengths from 1.5 meters to 15 meters in 1.5-meter increments. The proportion of ore and waste were tracked at three different gold cutoff grades (0.40, 0.60, and 0.80 g/t) for each composite length so that the amount of internal dilution and potential ore loss could be calculated. By definition, dilution can only be measured for composites above a given cutoff and by contrast, ore loss can only be described for composites that are below a cutoff grade. Figure 19.10 is a graph that illustrates the amount of internal dilution that is incurred with increasing composite length.

19.2   RESOURCE MODEL

The Mulatos resource model was constructed using 3m x 3m x3m blocks and later combined to 6m x 6m x 6m blocks. The final model limits in the local grid system are:

- North-South, 3800N to 5300N, 250 rows in the model
- East-West, 1500E to 2400E, 150 columns in the model

- Elevation, 900 to 1452, 92 benches or tiers in the model

The gold, silver, copper and sulfur grades were estimated into the blocks from the composite database using inverse distance estimation that respected various geologic constraints.

19.2.1 Variography

For this study anisotropy vectors were determined for gold, silver, and copper by interpreting correlograms using 3-meter-long drill hole composites. Ed Isaaks Sage2001 variogram modeling package was used to generate 37 directional correlograms from which a search ellipse was constructed using a least squares regression routine. In addition to anisotropy vectors, search range distances were also obtained for each ellipse. MineSight® variogram modeling tools were used to develop a search ellipse for total sulfur.

Gold grade and gold indicator correlograms were calculated and analyzed. The final search ellipse that was selected for gold zones 1 and 5 was based on a grade correlogram. Figure 19.12 shows the gold grade correlogram that was constructed using a nested spherical model.

Silver grade and indicator correlograms were calculated and interpreted using Sage2001. The major axis of the search ellipse for silver trends about N30W. Figure 19.13 shows the relationship of the gold and silver search ellipses.

Copper grade and indicator correlograms were interpreted using Sage2001 software. A N65W trending search ellipse was indicated from a copper grade correlogram.

Directional variograms (correlograms) were calculated for total sulfur at 45° increments at 0, 45, 90 and 135 degrees on the horizontal plane using 6-meter-long composite data. A vertical tolerance angle of (+/-) 5° was used from the horizontal plane. Initially oxide, mixed, and sulfide populations were analyzed, but due to a lack of data it was decided to combine oxide and mixed domains for the spatial analysis.

In order to determine the continuity of sulfur in different horizontal directions and to establish the strike direction, the directional correlogram values were contoured on a horizontal plane. These contours were used to aid in the selection of the ranges for modeling the correlograms. The total sulfur correlograms in sulfide material indicated a strike direction of about N30E. The oxide and

mixed variograms indicated a strike direction of approximately N-S. The vertical continuity was determined using the down-hole variograms.

19.2.2 Geologic Constraints

Various methods were used to control or constrain the estimate of gold, silver, copper, and total sulfur block model grades. In the case of gold, mineralization was seen to cross cut various lithologic and alteration boundaries so the grade envelope approach was thought to be one of the best ways of constraining the estimate of block grades. The following sections describe the methods that were used for each metal

19.2.2.1 Optimization Parameters

Five distinct gold grade domains were created for constraining the estimate of gold resources. Three of the domains were high-grade, more structurally controlled zones that had been identified by surface and underground mapping. The primary gold domain contains the bulk of the deposit and is characterized by dissemeninated and stockwork-type mineralization. This domain was constructed by drawing gold grade contours in plan view on three-meter-spaced horizontal level plans using a 0.25 g/t gold cutoff grade. Alteration, surface and underground mapping were used as a guide in drawing the grade envelopes along with gold composite grade values. In the absence of mapped geologic control the contours were drawn mid-way between "ore" and "waste" holes. The envelope was typically drawn between 25 to 30 meters outboard of mineralized perimeter drill holes. The last domain was the volume of material located outside of the 0.25 g/t gold envelope. Table 19.30 summarizes the five gold domains that were developed for the resource model.

Table 19.30
Gold Zone Domains

AUZON Block Code	Description
1	0.25 g/t gold grade contours
2	Northwest trending high-grade strucure
3	North-south trending high-grade strucure
4	Northeast trending high-grade strucure
5	Default area outside of 0.25 g/t envelope

The high-grade structures were drawn on 3-meter-spaced level plans using all available geologic data and drill hole assay information. Once the polygonal outlines were drawn in plan, they were linked together forming three-dimensional wireframes that were used for coding both drill hole composites and model blocks.

The 0.25 g/t gold grade envelopes were drawn at mid-bench level horizons and were then extruded vertically 1.5 meters bi-directionally to form wireframes that were used to code the drill holes and model blocks. By default, all other uncoded drill holes and model blocks were assigned AUZON code 5.

19.2.2.2   Silver Discriminator Domains

A probabilistic approach was taken to define two silver grade populations. An indicator cutoff grade of 3 g/t was selected after reviewing silver grade histograms and visually inspecting section and plan views of silver composite grades. A field in the composite file was set to "1" if the composited silver grade was above a 3 g/t cutoff grade. All other composites were set to "0". The silver indicator was then interpolated and all blocks with an estimated value of 0.50 or greater were flagged as a higher-grade population. Blocks with an estimated value of less than 0.50 were considered to be another lower-grade population. This indicator method essentially discriminated the silver population into two categories. The composites were back tagged from the flagged model blocks for subsequent silver grade estimation by domain.

19.2.2.3 Copper Discriminator Domains

Two copper domains were also constructed using the same principal that was previously described for silver domains. In the case of copper, a 250 ppm indicator threshold was used to discriminate the copper distribution into high and low-grade populations.

19.2.3 Topographic Model

A topographic surface was constructed using the provided topographic contour data. The model was then coded with the percentage of each block located below the surface or how much of the block contained non-air.

19.2.4 Density Model

Density values were loaded to the block model based on a review of average specific gravity values by various geologic combinations (lithology, alteration, and oxidation). Table 2.21 summarizes the specific gravity values that were loaded to the block model.

19.2.5 Grade Estimation

Grades (except for sulfur) were estimated for 3-meter by 3-meter by 3-meter blocks using 3-meter-long drill hole composites. The philosophy behind using this block size was that it provided for greater resolution between ore and waste contacts. Once the block grades were estimated they could be combined or regularized into different selective mining units or SMU's. The amount of internal dilution and ore loss could then be calculated for each SMU.

Since gold is the most important metal in the Mulatos deposit, more effort was expended in the estimate of the metal. Minimal sample data were used to estimate the gold grade for each block in order to minimize the amount of grade smoothing that is typical of most grade estimates.

19.2.5.1 Gold

Block grades were estimated for each of the five gold zone domains using multiple runs for each zone using increasingly longer search distances. The method used for all gold domains was inverse distance to the third power. The number of eligible samples varied by

estimation run. Table 19.31 summarizes the key parameters that were used to estimate gold resources.

Table 19.31
Gold Estimation Parameters

Gold Zone	Run No.	Composite Selection				Composite Search/Ellipse Ranges (m)							Ellipse Orientation (°)		
		Min	Max	Max/hole	Au Zone	X	Y	Z	Max Proj.	Major	Minor	Vert.	ROTN	DIPN	DIPE
1	1	1	3	1	1 & 2	3	3	1.5	3	118	66	76	20	19	-36
1	2	1	3	1	1 & 2	15	20	10	18	118	66	76	20	19	-36
1	3	1	3	1	1 & 2	30	40	15	36	118	66	76	20	19	-36
1	4	2	6	2	1 & 2	60	75	25	75	118	66	76	20	19	-36
2	1	1	3	1	2	3	3	3	3	90	90	20	245	-75	0
2	2	1	3	1	2	15	20	20	18	90	90	20	245	-75	0
2	3	1	3	1	2	30	40	40	36	90	90	20	245	-75	0
2	4	2	6	2	2	50	70	70	60	90	90	20	245	-75	0
2	5	2	6	2	1,2,3,4	75	75	75	75	90	90	20	245	-75	0
3	1	1	3	1	3	3	3	3	3	90	90	20	270	-75	0
3	2	1	3	1	3	20	20	20	18	90	90	20	270	-75	0
3	3	1	3	1	3	40	40	40	36	90	90	20	270	-75	0
3	4	2	6	2	3	70	70	70	60	90	90	20	270	-75	0
3	5	2	6	2	1,2,3,4	75	75	75	75	90	90	20	270	-75	0
4	1	1	3	1	4	3	3	3	3	90	90	20	300	-75	0
4	2	1	3	1	4	20	20	20	18	90	90	20	300	-75	0
4	3	1	3	1	4	40	40	40	36	90	90	20	300	-75	0
4	4	2	6	2	4	70	70	70	60	90	90	20	300	-75	0
4	5	2	6	2	1,2,3,4	75	75	75	75	90	90	20	300	-75	0
5	1	1	3	1	5	3	3	1.5	3	118	66	76	20	19	-36
5	2	1	3	1	5	15	20	10	18	118	66	76	20	19	-36
5	3	1	3	1	5	30	40	15	36	118	66	76	20	19	-36
5	4	2	6	2	5	60	75	25	60	118	66	76	20	19	-36

Notes:
Max Proj. = maximum projection distance of composite grades
ROTN = rotation about the Z-axis using the "left-hand rule"
DIPN = rotation about the X-axis using the "right-hand rule"
DIPE = rotation about the Y-axis using the "left-hand rule"

The gold zones and Au zone codes shown in Table 19.31 refer to the domains discussed in Section 19.2.2.1. Blocks that were estimated were flagged and therefore not eligible to be estimated by subsequent runs that used longer search ranges. Other parameters not shown in Table 19.31 are only gold composites from core, reverse circulation, and underground channel samples were used, the composites had to be at least 1.5 meters long, and the estimate was weighted by the length of the samples. Because fixed length composites were used only 1.6 percent of the gold composites were between 1.5 and 2.99 meters long.

The search ellipse that was used for gold zones 1 and 5 was obtained from modeling a gold grade correlogram. For all of the estimation runs for gold zones 1 and 5 the full maximum search ellipse was used, but each run limited the effective search range by only allowing composites to be within certain X, Y, and Z distances from each block.

No variograms could be established for gold zones 2, 3, and 4 so ellipses were oriented in the plane of those zones. A fifth estimation run was also used for those zones to ensure that all blocks were estimated.

The number of composites and the distance to the closest drill hole composite were captured for each estimation run. These data were later used to classify the resources.

19.2.5.2   Silver

The estimate of silver block grades was constrained by using an indicator or "discriminator" approach as described in Section 19.2.2.2. This process consists of two distinct steps. First the higher-grade population was defined using a simple probabilistic function where a 3 g/t silver indicator was interpolated using the inverse distance squared method. Interpolated blocks having a value greater than or equal to 0.50 were flagged as having a high probability of being in excess of 3 g/t. The 3-meter-long composites were then back tagged from the block model and used in a multi-pass interpolation plan where longer search distances were used for each successive run. Block grades were estimated using the inverse distance cubed method. Silver grades for blocks within the high-grade flagged population were estimated using composites located both inside and outside of the flagged population. Grades were estimated for blocks outside of the high-grade population using only composites that were located outside of the flagged population. In addition to estimating silver grades using capped assay data, an outlier restriction was imposed. Silver composites in excess of 60 g/t could only be projected 30 meters. Table 19.32 summarizes the interpolation parameters that were used for estimating the silver indicator and silver block grades.

Table 19.32
Silver Estimation Parameters

Ag Zone	Run No.	Composite Selection				Composite Search/Ellipse Ranges (m)							Ellipse Orientation (º)		
		Min	Max	Max/hole	Ag Zone	X	Y	Z	Max Proj.	Major	Minor	Vert.	ROTN	DIPN	DIPE
Indicator		3	12	3	n/a	70	70	30	60	100	75	25	330	0	0
1	1	1	3	1	1 & 2	6	6	3	6	100	75	25	330	0	0
1	2	1	6	2	1 & 2	40	40	15	30	100	75	25	330	0	0
1	3	2	6	2	1 & 2	70	70	25	60	100	75	25	330	0	0
2	1	1	3	1	2	6	6	3	6	100	75	25	330	0	0
2	2	1	6	2	2	40	40	15	30	100	75	25	330	0	0
2	3	2	6	2	2	70	70	25	60	100	75	25	330	0	0

Notes:

Ag Zone = 1 is high-grade population, 2 is low-grade population

Max Proj. = maximum projection distance of composite grades

ROTN = rotation about the Z-axis using the "left-hand rule"

DIPN = rotation about the X-axis using the "right-hand rule"

DIPE = rotation about the Y-axis using the "left-hand rule"

19.2.5.3 Copper

The approach for estimating copper grades was identical to the one used for the estimation of silver grades (i.e. inverse distance squared for the indicator and inverse distance cubed for grade). In the case of copper, a 250-ppm indicator was selected for the "discriminator". Like silver, an outlier restriction was used for estimating copper grades. Copper composites in excess of 5000 ppm were only projected a maximum distance of 9 meters. Table 19.33 summarizes the parameters that were used for estimating block copper grades.

Table 19.33
Copper Estimation Parameters

Cu Zone	Run No.	Composite Selection				Composite Search/Ellipse Ranges (m)							Ellipse Orientation (º)		
		Min	Max	Max/hole	Cu Zone	X	Y	Z	Max Proj.	Major	Minor	Vert.	ROTN	DIPN	DIPE
Indicator		3	12	3	n/a	70	70	30	60	100	75	25	295	0	0
1	1	1	3	1	1 & 2	6	6	3	6	100	75	25	295	0	0
1	2	1	6	2	1 & 2	40	40	15	30	100	75	25	295	0	0
1	3	2	6	2	1 & 2	70	70	25	60	100	75	25	295	0	0
2	1	1	3	1	2	6	6	3	6	100	75	25	295	0	0
2	2	1	6	2	2	40	40	15	30	100	75	25	295	0	0
2	3	2	6	2	2	70	70	25	60	100	75	25	295	0	0

Notes:

Cu Zone = 1 is high-grade population, 2 is low-grade population

Max Proj. = maximum projection distance of composite grades

ROTN = rotation about the Z-axis using the "left-hand rule"

DIPN = rotation about the X-axis using the "right-hand rule"

DIPE = rotation about the Y-axis using the "left-hand rule

19.2.5.4 Sulfur

Interpolation was completed in two inverse distance squared passes for each oxidation domain. In each pass, an ellipsoidal search was used to select the composites for the interpolation of the blocks. Only the composites that had the same REDOX (oxidation) domain code as the block were used during in the interpolation plan. For example, only oxide composites were used to interpolate sulfur for oxide blocks, and only the sulfide composites were used to interpolate the sulfide blocks. The intent with the first pass for each domain was to make sure that all blocks were interpolated. The maximum search distance for this run was 300 meters for the major axis of the ellipsoid. The length of the minor and vertical axis was adjusted according to the ratios from the respective correlograms. The second pass had a maximum search distance of 75 meters. That run essentially overwrote most of the blocks that were estimated from the first pass. Outlier restriction was used to minimize smearing high sulfur values. Oxide composites in excess of 12% total sulfur were only projected a maximum of 15 meters. Mixed oxidation composites greater than 6.5% total sulfur were projected a maximum of 15 meters. Sulfide composites in excess of 8.5% were only projected a maximum distance of 15 meters.

19.2.6 Model Verification

As a test to ensure that the gold grade model was globally unbiased, a nearest neighbor model was constructed. The nearest neighbor model used the same search strategy as the inverse distance cubed model. The maximum projection distance for the composite data was 60 meters. Table 19.34 compares the inverse distance cubed grade model with the nearest neighbor model grade at a zero cutoff grade for different volumes and resource categories.

Table 19.34
Nearest Neighbor Grade Comparison at a 0.0 g/t Au Cutoff

Volume Examined	Au - ID^3 (g/t)	Au - NN (g/t)	% Difference
All Blocks Within 36m of Drilling	0.4409	0.4539	-2.86%
All Blocks Inside Au Zones 1-4	0.9265	0.9332	-0.72%
M+I Blocks Inside Au Zones 1-4	0.9351	0.9421	-0.74%
M+I+I Blocks Inside Au Zones 1-5	0.9265	0.9335	-0.75%

Note:
Refer to Table 2.40 for Au zone description
M+I = Measured + Indicated Resource
M+I+I = Measured + Indicated + Inferred Resource

There is about a 3 percent variance for all model blocks within 36 meters of drilling. This is probably a result of the soft/hard boundary conditions that were used for estimating gold grades for gold zone 1 (0.25 g/t envelope). The inverse distance cubed grade is about 3 percent lower than the nearest neighbor model. The two grades are very comparable for the other categories shown in Table 19.34 indicating that the model is not globally biased.

The block grades were also verified by a visual inspection by comparing composite drill hole and estimated block gold grades. Figure 19.14 is a plan map showing the lines of section for three east-west and one north-south block model cross sections. The ultimate design pit is shown in Figure 19.14 as faint grey lines and in all cross section and plan maps as a light brown line. Figure 19.15 through Figure 19.20 contain various cross sectional and level views of the block model. The drill hole composites are shown on all sections and plans with the same gold color-coding as the model blocks.

A comparison was made between the 3-meter-long drill hole composite grades and the 3m x 3m x 3m resource model blocks. Mean block grades and tonnes were summarized for all blocks inside of the ultimate design pit at a variety of gold cutoffs. Similarly, drill hole composite statistics were tabulated for data located within the pit and 30-meters outboard of the pit shell. Figure 19.22 is a histogram comparing the amount of drilling and the amount of model tonnes above each gold cutoff grade. Figure 19.23 shows the average grade of the composites and model blocks for the same volume at different cutoff grades.

Based on the nearest neighbor grade comparisons with the inverse distance cubed model, statistical distribution of grades, and a

visual examination of the block grades, the resource model seems to be satisfactory for mine planning purposes.

19.2.6.1   Selective Mining Unit Study

As mentioned in Section 19.2.5, grades were interpolated into 3-meter by 3-meter by 3-meter blocks. This selective mining unit (SMU) is probably impractical for a bulk tonnage operation but it allows better resolution of grade contacts in the modeling process. After the grades block grades were estimated, three gold indicator flags were set to 0 or 1 based on whether the 3m x 3m x 3m block grade was less than or greater than the selected cutoff grade. The indicators that were chosen were 0.25, 0.50, and 0.75 g/t. Then the 3m x 3m x 3m block gold grades and indicators were "composited" or regularized into a variety of larger SMU's. Figure 19.24 graphically illustrates the process of regularization by showing eight 3m x 3m x 3m blocks being averaged or regularized into a single 6m x 6m x 6m block. In this case, the larger SMU contains 50% internal dilution at a 0.50 cutoff grade, 62.5% dilution at a 0.70 cutoff grade, and 75% dilution at the 0.80 cutoff grade.

Fourteen different SMU's were created. Table 19.35 shows the dimensions for each SMU along with the cubic meters of each block and how many times larger each block is relative to the original 3m x 3m x 3m blocks.

Table 19.35
SMU Block Dimensions

East-West (meters)	North-South (meters)	Bench Height (meters)	Volume (m^3)	Volume Increase
3	3	3	27	0
6	6	3	108	4
9	9	3	243	9
12	12	3	432	16
15	15	3	675	25
6	6	6	216	8
9	9	6	486	18
12	12	6	864	32
15	15	6	1,350	50
9	9	9	729	27
12	12	9	1,296	48
15	15	9	2,025	75
12	12	12	1,728	64
15	15	12	2,700	100
15	15	15	3,375	125

Resources were calculated for the 14 SMU's listed in Table 19.35 for all blocks that were within 36 meters of drilling data. These resources were then compared with the original 3m x 3m x 3m block model resources to calculate dilution and gold metal loss for each SMU. Figure 19.25 shows the amount of tonnage dilution at four different cutoff grades relative to the initial 3m x 3m x 3m block model. There is a step function in the dilution curves that mark the change in bench height. In general, dilution is primarily a function of bench height, although larger plan dimensions do contribute to diluting block grades.

Figure 19.26 shows how the block gold grade decreases with increasing SMU size. Again, there is a step function in the curves each time the bench height is changed. At a 0.6 g/t gold cutoff grade the average block grade decreases about 5% when going from a 3m x 3m x 3m to a 6m x 6m x 6m block. The grade decreases by 8-10 percent for 9-meter high benches.

Figure 19.27 shows the percentage of gold that would be "lost" by going from the initial 3m x 3m x 3m blocks to various SMU's.

Figure 19.28 better illustrates the effect that bench height plays with the "loss" of gold ounces with increasing bench height. A plan block size of 15m x 15m was kept constant and five bench heights are shown in the figure. At reasonable economic cutoff grades there is about 2 percent gold loss for every three meters of increased bench height.

Based on this SMU study, a bench height of six meters seems appropriate. While a 3-meter bench allows for greater selectivity, that height is very impractical and the added costs for mining such a short bench would probably not be offset by recovered metal. At reasonable economic cutoff grades, the percentage of gold ounces that would be unrecoverable accelerates with bench heights in excess of 6 meters. So the cost reduction in mining the higher benches do not justify accepting the dilution that will occur by mining higher benches. For that reason, the final block size that was selected for pit optimization was 6m x 6m x 6m.

19.2.7 Resource Classification

Gold resources were categorized in the initial 3m x 3m x 3m block model by using the distance to drilling method. The distance to the closest drill hole composite was captured using a nearest neighbor interpolation method. The criteria for measured resources were kept conservative, as the property has no recorded history of open pit mining. Measured resources were defined for blocks that are within six meters of an exploration drill hole. In an open pit operation the last opportunity for selecting the ore/waste boundary is based on blasthole assays. Typically, the blasthole spacing for an operation like the one intended for the Mulatos deposit is somewhere between 5 and 7 meters. Therefore, within the gold grade envelopes, blocks were classified as measured if they were located within what may be a typical blasthole spacing. Measured resources were not assigned to blocks located outside of the four gold grade zones. The distance that distinguishes indicated from inferred resources was determined by analyzing the spatial continuity of gold based on a gold correlogram. The distance (range) corresponding to 80% of the total variance was selected as the maximum allowable distance from drilling data for indicated resources. The correlation of gold values beyond 80% of the variogram variance becomes more tenuous and is subject to how the practitioner modeled the variogram. The maximum allowable range for inferred resources was obtained from the correlogram in

a similar manner, only the range was found at 90% of the total variance. Refer to Figure 19.12, which shows the ranges corresponding to 80% and 90% of the gold correlogram variance. Table 19.36 summarizes the distances to drilling for the various gold zones for each resource category. The inferred column is for reference only. It is not used.

Table 19.36
Resource Classification Criteria (3m x 3m x 3m Model)

Gold Zone	Distance to Closest Drill Hole (m)					
	Measured		Indicated		Inferred	
	Min	Max	Min	Max	Min	Max
1	0	6	7	36	37	74
2	0	6	7	36	37	74
3	0	6	7	36	37	74
4	0	6	7	36	37	74
5	n/a	n/a	0	18	19	36

The final resource model was regularized from the 3m x 3m x 3m model into a 6m x 6m x 6m configuration. The same method criteria was used for classifying resources in the 6m x 6m x 6m model as shown in Table 19.37. Additional resource classes were created in the regularized model to account for internal dilution that was tracked. Table19.37 summarizes the resource classification criteria that were used for the 6m x 6m x 6m block model.

Table 19.37
Resource Classification Criteria (6m x 6m x 6m Model)

Resource Class	Model Code	Distance to Closest Drill Hole (m)			
		Gold Zones 1 - 4		Gold Zone 5	
		Min	Max	Min	Max
Measured < 50% Dilution	1	0	6	n/a	n/a
Indicated < 50% Dilution	2	7	36	0	18
Inferred < 50% Dilution	5	37	74	19	36
Measured > 50% Dilution	3	0	6	n/a	n/a
Indicated > 50% Dilution	4	7	36	0	18
Inferred > 50% Dilution	6	37	74	19	36

To account for some past production that occurred at the northern end of the Mulatos deposit (Mina Viejo), a volume of approximately 260,000 tonnes was classified as an inferred resource so that material would not be available for subsequent pit optimization runs. There was no accurate historical information about the location, size, and orientation of some of the historical

underground workings. A three-dimensional wireframe was constructed from project northing 4825 to 4925. A ten-meter high stope height was drawn for this 100-meter long stope shape and used to code the resource model blocks. The stope block is located several hundred meters north of the current ultimate pit design.

9.2.5   Model

Block model resources were tabulated for the 6m x 6m x 6m model at a variety of gold cutoff grades. Table 19.38 summarizes measured, indicated, and inferred resources for the Mulatos deposit using the resource classification criteria outlined in Section 19.2.7. The inferred column is for information only. It is not used in the resourse tabulation.

Table 19.38
Mulatos Gold Resources

Gold Cutoff, g/t	Measured		Indicated		Total Measured + Indicated Reserve		Inferred	
	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)
0.20	15,039	1.24	125,147	0.83	140,186	0.88	54,667	0.50
0.40	11,978	1.48	81,122	1.12	93,100	1.17	21,192	0.86
0.60	9,089	1.80	53,127	1.46	62,216	1.51	10,382	1.26
0.80	7,124	2.10	37,161	1.79	44,285	1.84	6,336	1.63
1.00	5,642	2.42	27,452	2.11	33,094	2.17	4,240	1.99

KT = tonnes x 1000 (thousand metric tons)

19.3   OPEN PIT RESERVES

The open pit presented in this document is for the southern Estrella area. Other mineralization is present north of Estrella and those areas need further investigation and evaluation before being incorporated into a reserve.

The highlights of the Estrella open pit are:

> Mineable Ore Reserves
> > Tonnes   36,367,000
> > Average Gold Grade   1.636 g/t
> > Average Gold Recovery   72.9%
>
> Strip Ratio, Life of Mine   1.4 to 1.0

19.3.1 Open Pit Optimization

The final pit design is based on a modified floating cone algorithm geometry using a gold price of $350/oz. The block model of the deposit was developed by Mike Lechner of Resource Modeling Inc. and is described in the previous sections of this report volume. The process recoveries, process costs and general and administrative (G&A) costs estimates for input to the pit definition analysis are provided by M3. IMC provided the mining cost input. The slope angles are based on work completed by Golder Associates for a previous feasibility study.

The Estrella deposit contains three ore types defined as oxide, mixed and sulfide with the mixed and sulfide ore types further sub-divided into silicified and non-silicified. There is an additional ore type called high copper because of the elevated copper grades. The gold recovery in this type is low, costs are high, and there are relatively very few tonnes of this material in the Estrella deposit. The high copper zone was excluded from any economic consideration for the pit definition or any ore production schedules. The high copper tonnage is treated as waste in any tabulation of pit reserves.

Only the measured and indicated resource is used for input to the pit definition and tabulation of reserves.

19.3.1.1   Optimization Parameters

Table 19.39 summarizes the economic parameters used for the base case floating cones. The gold recovery,

lime and cyanide consumptions vary with ore type, thus a 'net of process' variable was added to the model that takes these variables into account when the net value of a model block is determined. The net of process value is defined as a value of the block based on the gold grade times recovery times the metal price, less the process and G&A costs:

Net of process =
gold price x gold grade x gold recovery – (process fixed costs + lime consumption x price
+ cyanide consumption x price + G&A costs)

19.3.1.2 Gold Price

The base case gold price for the definition of the final pit limits is $350/oz gold. This price is below the 2003 yearly average of $363/oz, and $350/oz gold does reflect a premium of about $35/oz over the 3 year average (January 2001 through December 2003) of $315/oz. Pit definition runs were completed at gold prices ranging from $100/oz to $400/oz. The lower gold priced cones were used as a guide for the design of the early mining phases and the $400/oz cone was to define any potential for pit expansion at higher gold prices

Table 19.39
Economic Parameters for Floating Cone Evaluation

Mining Cost per Total Tonne	$0.80 plus lift cost
Additional Mining Cost for Haulage below 1302 Bench	$0.005 per bench
Fixed Process Costs: Oxide and Mixed ore types	$1.67/tonne processed
Sulfide ore types	$1.87/tonne processed
Liner Cost	$0.49/tonne processed
G&A Cost	$0.71/tonne processed
Cyanide Unit Cost	$1.40/kg
Lime Unit Cost	$0.075/kg
Cyanide and Lime Consumption Rates	Variable by ore type
Gold Recovery	Variable by ore type
Royalties	None
Gold Price, base case	$350/oz
Sensitivities	$100/oz to $400/oz
Silver Price, silver not included in pit definition economics	$0.00
Overall slope angle	45 degrees

19.3.1.3 Gold Recovery

The gold recoveries by ore type are shown in Table 19.40. These are based on metallurgical test work that is discussed in other sections of this report. The recoveries shown in Table 19.40 are for the Estrella area. The areas north of Estrella were evaluated on a preliminary basis using the same gold recoveries for the oxide and mixed ore types but a lower sulfide recovery was applied. The northern areas are not included in the pit reserves for this report.

Table 19.40
Gold Recoveries for Pit Definition

Ore Type	Overall Recovery
Oxide	$(0.988 - (0.027/\text{gold grade}))$
Mixed, silicified and non-silicified	$(0.909 - (0.131/\text{gold grade}))$
Sulfide, silicified and non-silicified	$(0.734 - (0.098/\text{gold grade}))$
Sulfide in north zones	$(0.634 - (0.098/\text{gold grade}))$
High Copper ore type	No gold recovery assigned

19.3.1.4 Regent Consumption

The lime and cyanide consumption rates are variable by the ore type. The rates shown in Table 19.41 were used for the pit definition runs. These consumptions are the same as used in the 1997 Placer Dome feasibility study and may differ from the consumption rates currently being proposed for the project, discussed in the metallurgical test work section of this report.

Table 19.41
Lime and Cyanide Consumptions by Ore Type

Ore Type	Cyanide Consumption, kg/tonne	Lime Consumption, kg/ton
Oxide	0.15	6.0
Mixed, non-silicified	0.20	6.0
Mixed, silicified	0.18	6.0
Sulfide, non-silicified	0.31	7.0
Sulfide, silicified	0.15	7.0

19.3.1.5   Operating Cost

The operating costs include mining, ore crushing and processing and general and administrative (G&A) costs. The mining cost is estimated for the cone runs to be $0.80/tonne mined plus an additional haulage cost of $0.005/tonne per bench below the 1302 bench. This is based on the assumption that all the ore would be hauled to a crusher dump pocket higher than the 1302 bench. The initial designs of the waste dump were also higher than the 1302 bench elevation. Example equivalent mining costs would be: $0.80/t for the 1302 bench and higher, $0.835/t on the 1260 bench, $0.885/t on the 1200 bench, $0.935/t on the 1140 bench, and $0.985/t on the 1098 bench (the lowest bench in the final pit design).

The process cost is a combination of fixed costs (vary by ore type) and variable costs (based on cyanide and lime consumption rates). To correctly account for these, a process cost by ore type was assigned to each block in the model. The total process cost per tonne of ore included the G&A cost since it is treated as a cost per tonne processed cost. Table 19.42 shows the costs by ore type.

Table 19.42
Process and G&A Costs by Ore Type

Ore Type	Fixed Process $/t	Liner Cost, $/t	Cyanide Cost (1), $/t	Lime Cost (2), $/t	G&A Cost $/t	Total Cost, $/t
Oxide	1.67	0.49	0.21	0.45	0.71	3.53
Mixed, non-silicified	1.67	0.49	0.28	0.45	0.71	3.60
Mixed, silicified	1.67	0.49	0.25	0.45	0.71	3.57
Sulfide, non-silicified	1.97	0.49	0.44	0.52	0.71	4.13
Sulfide, silicified	1.97	0.49	0.21	0.52	0.71	3.90

1) Cyanide cost = $1.40/kg x consumption rate in Table 19.41
2) Lime cost = $0.075/kg x consumption rate in Table 19.41

Using the total costs shown in Table 19.42 and the variable gold recovery formulas in Table 19.40, a net of process value was assigned to each block for the oxide, mixed and sulfide ore types. This value changed with gold price.

Net of process =
gold price x gold grade x gold recovery – (process fixed costs + lime consumption x price
 + cyanide consumption x price + G&A cost)

19.3.1.6 Pit Slopes

Golder Associates completed a slope angle evaluation for a feasibility study developed by Placer Dome in 1997. This study recommended inter-ramp slope angles of 55° on the west, 51° on the northeast and southeast pit sectors and 48° on the east high wall. There would be haulage ramps on the east, north and south walls of the final pit, and the overall slope angle for the cone runs was selected at 45°. The west side of the pit daylights for the majority of the pit, and using the same 45° overall slope angle for this side did not impact the cone results.

19.3.1.7 Cut Off Grades

The internal cutoff grade (covering the cost of process plus G&A) varies with ore type and the breakeven cutoff grade (mining, processing and G&A) varies with both ore type and depth in the pit. Table 19.43 shows cutoff grades by ore type using a gold price of $350/oz. Tables in this report will report tonnage tabulations based on the 'net of process' cutoff grade. The internal cutoff grade for this variable is $0.00 and the breakeven using the base $0.80/t mining cost is $0.80. Using the net of process value for cutoff grades simplifies the reporting of the different ore types that have different costs and recoveries.

Table 19.43
Cutoff Grades by Ore Type

Cutoff	Oxide	Mixed, silicified	Mixed, Non-silicified	Sulfide, silicified	Sulfide, non-silicified
Internal	0.34	0.50	0.49	0.63	0.61
Breakeven, 1302 bench	0.42	0.57	0.57	0.73	0.70
Breakeven, 1200 bench	0.42	0.58	0.58	0.74	0.71
Breakeven, 1098 bench	0.43	0.59	0.59	0.75	0.73

Gold Price = $350/oz

19.3.1.8   Optimized Reserves

Floating cones were run using the base case economic parameters shown in Table 19.39 for the $350/oz gold price using only the measured and indicated resource to provide economic benefit to the cone economics. This cone was restricted to the Estrella area only and became the basis of the pit design. Other cones were run using the base case parameters and lower gold prices to define the starting mining phases. Table 19.44 shows the results of the $350/oz and lower gold prices using the net of process value as the cutoff grade for defining ore tonnage. The net of process cutoff grade used is $0.10/t, which is just above the internal cutoff grade of $0.00/t. Figure 19.29 shows the outlines of these cones on the 1260 bench.

Table 19.44
Summary of Floating Cones at Selected Gold Prices

Base Case Parameters							
Measured and Indicated Resource to Credit Cone Economics and Tabulation							
Estrella Area Only							
No Discounting							
Gold Price $/oz	Ore ktonnes	Ore, Gold Grade, g/t	Ore, Recovered Gold Grade, g/t	Ore, (1) Net of Process Value, $/t	Waste ktonnes	Total ktonnes	Waste / Ore Ratio
$350	37,213	1.63	1.18	$9.48	47,278	84,491	1.27
$300	33,080	1.73	1.27	$8.38	43,196	76,276	1.31
$250	28,203	1.88	1.39	$7.33	41,506	69,709	1.47
$200	21,972	2.10	1.57	$6.29	35,847	57,819	1.63
$150	15,358	2.46	1.87	$5.21	33,766	49,124	2.20
$125	10,683	2.75	2.12	$4.74	27,308	37,991	2.56
$100	3,653	3.42	2.72	$4.99	8,877	12,530	2.43

1) Net of Process Value Calculated Using Gold Price for Respective Cone Run

19.3.1.9   Sensitivities

Sensitivity cones to the base case parameters were run for the following cases:

- Bench discounting to incorporate time value of money on the final pit wall
- Including the inferred material for cone economics
- Increase the gold price to $400/oz.

The inclusion of bench discounting for the $350/oz cone produced very little impact on the cone geometry that was to be used for the final pit design. The block values of revenue and costs were discounted at 2% per bench (10% per year if 5 benches per year are mined on average). This gives an indication of the impact of delaying the revenue from mining the lower ore benches after stripping the upper waste benches. In the case of the Estrella pit, the impact is small because there is some ore along much of the pit wall.

The inclusion of the inferred material for crediting the cone economics had a minimal impact. There is not a large amount of inferred resource in the Estrella area.

Raising the gold price to $400/oz added about 10% to the ore above cutoff. It did not generate a large enough step out to create another mining phase.

Table 19.45 summarizes the sensitivity cone runs.

Table 19.45
Summary of Sensitivity Cone Runs

Gold Price $/oz	Ore ktonnes	Ore, Gold Grade, g/t	Ore, Recovered Gold Grade, g/t	Ore, (1) Net of Process Value, $/t	Waste ktonnes	Total ktonnes	Waste / Ore Ratio
Base Case Cone Run							
$350	37,213	1.63	1.18	$9.48	47,278	84,491	1.27
Discounted Cone Run							
$350	36,850	1.63	1.18	$9.49	43,539	80,389	1.18
Base Case Cone Parameters with Inferred Material Included							
$350	37,710	1.63	1.18	$9.47	54,738	92,448	1.45
Base Case Cone Parameters with Inferred Material Included and $400/oz Gold Price							
$400	41,771	1.54	1.12	$10.51	67,409	109,180	1.61

1) Net of Process Value Calculated Using Gold Price for Respective Cone Run

19.3.2 Open Pit Design

The final pit design was based on the $350/oz gold floating cone. Table 19.46 shows the key open pit design parameters.

Table 19.46
Open Pit Design Parameters

Haul Road Width	25 meters
Haul Road Grade	10%
Mining Bench Height	6 meters

Number of Stacked Benches:
   Internal phases (not final walls)
   Final phase
	
	2 (12m high bench)
	5 (30m high bench)

Catch Bench Width:
   Between every 12m stacked bench
   Between every 30m stacked bench
	
	6 meters
	10 meters

Interramp Slope Angles and Bench Face Angles:
   West Side
   Northeast and Southeast Sides
   East Side
	Face/Interramp Angles
	70° / 55°
	65° / 51°
	60° / 48°

Nominal Minimum Mining Phase Width	80 meters

Figure 19.30 shows the final pit design. There are two main exits from the final pit, one at the 1320 elevation, which is used for all material above the 1230 elevation, and the second exit is at the 1230 elevation for material below that elevation. Waste storage facilities and the crushing facility are anticipated to be located south of the pit, therefore all exits were located on the southern side of the pit. The final pit is approximately 800 meters long in the north-south direction and 475 meters wide in the east-west direction. The pit bottom is at the 1098 elevation. The highest wall is about 264 meters in the northeast corner of the pit. The total area disturbed by the pit is about 34 hectares. Other pit exits are at 1248, 1390 and at Puerto del Aire.

The ore types included in the pit reserve are: oxide, mixed (silicified and non-silicified) and sulfide (silicified and non-silicified). Table 19.47 summarizes the ore tonnage within the final pit at the internal cutoff grade by ore type.

Table 19.47
South Estrella Pit Reserve – Based on Mine Production Schedule

	Proven		Probable		Proven + Probable	
Ore Type	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)
Oxide	324	1.18	2,478	1.12	2,802	1.12
Mixed, Non-silicified	184	1.72	1,025	1.56	1,209	1.58
Mixed, Silicified	1,865	1.85	5,998	1.59	7,863	1.65
Sulfide, Non-silicified	1,466	1.60	6,990	1.45	8,456	1.48
Sulfide, Silicified	3,409	2.01	12,628	1.76	16,037	1.81
Total	7,248	1.84	29,119	1.59	36,367	1.64
Total Pit Tonnage = 87,937						

This reserve is the sum of the mine production schedule, which used a declining cutoff grade over time.

20 OTHER RELAVENT DATA AND INFORMATION

All information relative to the estimation of the Mulatos resources and the Estrella pit proven and probable reserves have been presented in previous sections.

21 INTERPETATION AND CONCLUSION

The interpretation of the results have been presented in the previous sections. In summary, the Mulatos resources and the Estrella pit reserves are shown in Tables 21.1 and 21.2. The inferred column is for information only. It is not to be used.

Table 21.1
Mulatos Resource

Gold Cutoff, g/t	Measured		Indicated		Total Measured + Indicated Reserve		Inferred	
	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)
0.20	15,039	1.24	125,147	0.83	140,186	0.88	54,667	0.50
0.40	11,978	1.48	81,122	1.12	93,100	1.17	21,192	0.86
0.60	9,089	1.80	53,127	1.46	62,216	1.51	10,382	1.26
0.80	7,124	2.10	37,161	1.79	44,285	1.84	6,336	1.63
1.00	5,642	2.42	27,452	2.11	33,094	2.17	4,240	1.99

KT = tonnes x 1000 (thousand metric tons)

Table 21.2
South Estrella Pit Reserve – Based on Mine Production Schedule

	Proven		Probable		Proven + Probable	
Ore Type	KT	Au (g/t)	KT	Au (g/t)	KT	Au (g/t)
Oxide	324	1.18	2,478	1.12	2,802	1.12
Mixed, Non-silicified	184	1.72	1,025	1.56	1,209	1.58
Mixed, Silicified	1,865	1.85	5,998	1.59	7,863	1.65
Sulfide, Non-silicified	1,466	1.60	6,990	1.45	8,456	1.48
Sulfide, Silicified	3,409	2.01	12,628	1.76	16,037	1.81
Total	7,248	1.84	29,119	1.59	36,367	1.64
Total Pit Tonnage = 87,937						

This reserve is the sum of the mine production schedule, which used a declining cutoff grade over time.

The areas north of the pit designed for Estrella are mineralized and presently being further defined by AGI. As additional information is incorporated into the resource estimate, this area should be evaluated for inclusion into the reserve base. The potential to increase both the resource and reserve on the property controlled by AGI is good.

22   RECOMMENDATIONS

M3 recommends a development decision now for the Estrella Pit portion of the Mulatos Deposit at an ore production rate of 10,000 MTPD. The details are contained herein and in the June 2004 Feasibility Study M3 completed for Alamos Gold.

A drilling program for the rest of the Mulatos deposit and the most promising of the seven surrounding deposits should be planned, to fill in the more promising areas. If additional reserves are developed, they can be mined and processed by the facilities recommended for the Estrella Pit, after its reserves are exhausted.

23   REFERENCES

1. June 1, 2004 Alamos Gold, Mulatos Feasibility Study Prepared by M3 Engineering.

2. May 19, 2004 M3 issued a Technical Report "The Estrella Pit Resource and Reserves Mulatos Sonora, Mexico" for Alamos Gold.

3. September 10, 2002 Preliminary Assessment and Scoping Study for Estrella Development alternative for Mulatos Deposit. Prepared for National Gold by Pincock Allen and Holt.

4. January 26, 2001 Qualifying report on the Salamander Gold property prepared for National Gold by Behre Dolbear and Company Ltd.

5. January 2000 Mulatos Project information package prepared by Minera San Augusto, S.A. de C.V. (Placer Dome)

6. June 1997 Feasibility Study Mulatos Project prepared by Placer Dome and M3 Engineering.

24   DATE THIS REPORT IS ISSUED: June 17, 2004, as revised on July 14, 2004.

25   ADDITIONAL REQUIREMENTS

The section addressed additional requirements for technical reports on development properties and production properties. This is based on the M3 Feasibility study, Alamos Gold Inc., "Mulatos Feasibility Study Phase One – Estrella Pit" dated June 1, 2004.

25.1   MINING OPERATIONS

Information and assumptions concerning the mining method, metallurgical processes and production forecast are described below.

Mulatos is a disseminated gold deposit hosted with a dacitic dome and breccia complex. Gold occurs in oxide, mixed oxide/sulfide and sulfide ore types, with pyrite as the primary sulfide mineral. The deposit is amenable to cyanidation in all ore types, but gold extraction decreases with decreasing levels of oxidation at the same size.

The Estrella pit portion of the Mulatos ore reserve is estimated at 36.37 Mt grading 1.637 g/t Au. Mining will be by conventional open pit methods using a 12.3 m3 hydraulic shovel and 90 t haul trucks. The open pit design has a waste to ore ratio of 1.42:1. Based on Estrella Pit ore reserves only, the mine life will be 10.5 years. It is anticipated that potential adjacent pits will keep the project in operation longer.

Ore will be crushed to a nominal size of 80% passing 9.5 mm (3/8") in a three-stage crushing circuit at the rate of 10,000 t/d for a total of 3,500,000 t/a. Crushed ore will then be heap leached using weak cyanide solution. Gold recovery will average 72.9% by a carbon plant and refinery.

The crushing plant and main mine equipment will come from the 15,000 t.p.d. Kennecott rawhide (Nevada) operation that suspended operations in late 2003 due to depleted reserves

The ore production schedule and mine major equipment fleet follow.

Ore Production Schedule by Ore Type

Year	Oxide		Mixed		Sulfide		Total	
	Ktonnes	Gold, g/t	Ktonnes	Gold, g/t	Ktonnes	Gold, g/t	Ktonnes	Gold, g/t
PP	363	1.452	54	2.746	101	1.887	518	1.672
1	562	1.575	482	2.195	1,937	2.080	2,981	2.004
2	363	1.174	1,042	2.194	2,095	1.891	3,500	1.906
3	137	0.856	775	1.881	2,588	1.735	3,500	1.733
4	262	0.854	1,580	1.619	1,658	1.957	3,500	1.722
5	638	0.927	1,088	1.466	1,774	1.818	3,500	1.546
6	200	0.759	823	1.118	2,477	1.720	3,500	1.524
7	53	1.211	769	1.519	2,678	1.486	3,500	1.489
8	128	0.856	1,033	1.464	2,339	1.594	3,500	1.529
9	82	0.488	885	1.623	2,533	1.423	3,500	1.452
10	12	0.495	472	1.360	3,016	1.657	3,500	1.613
11	2	0.567	69	1.316	1,297	1.380	1,368	1.375
Total	2,802	1.122	9,072	1.639	24,493	1.694	36,367	1.636

The 36.4 Mt of scheduled reserves is 7.3 Mt @ 1.84 g/t Au of proven and 29.1 Mt @ 1.64 g/t Au of probable reserve.

Mine Major Equipment Fleet

Mine Major Equipment Fleet Requirement												
						Time Period						
Equipment Type	PP	1	2	3	4	5	6	7	8	9	10	11
Blasthole Drill	2	2	2	2	2	2	2	2	2	2	2	2
Cat 992G Wheel Loader	2	2	2	2	2	2	2	2	2	2	2	2
Cat 777D Haul Truck	4	6	6	6	6	6	6	6	8	8	8	8
Cat D10R Track Dozer	2	2	2	2	2	2	2	2	2	2	2	2
Cat 14H Motor Grader	2	2	2	2	2	2	2	2	2	2	2	2
Cat 769D Water Truck	1	1	1	1	1	1	1	1	1	1	1	1
Cat 988G Wheel Loader	0	1	1	1	1	1	1	1	1	1	1	1
Cat 773D Haul Truck	0	2	2	2	2	2	2	2	2	2	2	2
IR ECM720 Rock Drill	1	1	1	1	1	1	1	1	1	1	1	1
Cat 325CL Excavator	1	1	1	1	1	1	1	1	1	1	1	1
TOTAL	15	20	20	20	20	20	20	20	22	22	22	22

25.2 RECOVERABILITY

Information concerning results of all test and operating results relating to the recoverability of the valuable component or commodity and ammeniability of the mineralization to proposed processing methods.

A weighted average gold extraction for all ore types has been estimated at 72.9%, (Up from 66.2% reported in the MSA/Placer 2000 Information Package).

This increase is mainly due to the elimination of the Mina Vieja and North Estrella mineralized zone from consideration, crush size reduced to P80 of 3/8" from 1/2" and additional sulfide ore column leach tests. Extraction formulas for the different ore types were changed to yield the following extractions:

Oxide	96.4% (was 90.0%)
Mixed and Fracture < 1.6%S	82.9% (was 75.0%)
Sulfide and Fracture > 1.6%S	67.6% (was 56.2%)
Weighted Average	72.9% (was 66.0%)

This change is due to investigations by RDi and includes elimination of the 0.95 scale up factor used by PDI, the higher gold recovery in the south (Estrella) pit and crush size reduced to P80 of 3/8 inch from ½ inch.

Because many of the Placer column leach tests were terminated early, RDi believes the 0.95 scale factor is not appropriate.

25.3 **METALLURGY**

25.3.1 Introduction

In 2002 Minas de Oro National, S.A. de C.V. (MON) contracted Resource Development, Inc. (RDi) to review the metallurgical testwork undertaken by Placer Dome Division Research Center (PDDRC)(Appendix 3.1of the 1997 Feasibility Study). The study indicated that the deportation of gold in the sulfide ore was unknown and the poor extraction of gold could be due to a combination of size dependence and solid solution of gold in pyrite. Based on these findings, MON decided to undertake additional test work at RDi, Polysius Research Center (Polysius) and Metcon Research Inc. (Metcon). The primary objectives of the additional testing were: (a) to determine by diagnostic testing the deportation of gold in sulfide ore; (b) to evaluate high pressure grinding roll (HPGR) comminution to see if ore fractures along grain boundaries enhanced gold recovery; and (c) to column test of finer crush sulfide ore. The testwork consisted of HPGR crushing tests, gravity tests and bottle roll and column leach tests on sulfide-bearing channel samples from the deposit.

The metallurgical review of PDDRC metallurgical test data also indicated that the gold in the sulfide ore from the south Estrella zone more readily liberated during crushing as compared to the gold in the ore from the north zone. This resulted in lower gold extraction from the north zone sulfides (Report No. 6, August 1996).

The south Estrella zone is the focus of the June 1, 2004 Mulatos Phase One Estrella Pit Feasibility Study.

25.3.2 Recommendation for Gold Recovery

25.3.2.1 Gold Recovery Equations

Please refer to Section 18 of this report for more detail.

25.4 **MARKETS**

Information concerning the markets for the issuer's production and the nature and material terms of any agency relationship.

Alamos expects to ship Gold/silver bullion by air from Matarachi, Sonora to an as yet unknown refiner. This is a common arrangement in Mexico

and elsewhere. A transportation, security, smelting and refining charge has been allowed for.

25.5   CONTRACTS

A discussion of whether the terms of the mining, concentrating, smelting, refining, transportation, handling, sales and hedging and forward sales contracts or arrangements, rates or charges are within market parameters.

M3 has done many similar gold heap leach projects in Mexico in the last 14 years. The cash cost operating cost estimated for Mulatos of $6.54/tonne (U.S.) is typical. Labor costs are lower in Mexico than elsewhere in North America and Chile, but are rising.

Gold- Silver bullion smelting and refining contracts are common and mostly similar. M3 has used the terms of similar contracts as Alamos does not yet have one. The security of Gold Bullion production and transportation is a concern in Mexico and so a little more money has been allowed. To the best of M3's knowledge, Alamos does not yet have any gold sales or hedging contracts.

25.6   ENVIRONMENT CONSIDERATIONS

A discussion of bond posting, remediation and reclamation.

As yet, a bond is not required in Mexico.

Mexican Environmental Norms (Laws) are similar to EPA, Canadian and World Bank criteria. There is a slight difference in attitude to enforcement. Mexican authorities review the project for flaws but do not take any responsibility.

The Feasibility Study was prepared with the above standards in mind and also with the banks newly developed "Equitorial Principles".

Placer Dome was very concerned with the potential for "Acid Mine Drainage" and so is Alamos and M3. The main mineral in the deposit is Pyrite. In fact, the ore is classified according to its average pyrite content; high being over 1.7%.

The Placer Dome Feasibility and the Alamos Feasibility rely on reclamation of the mine dump on a continuous basis while operating, and that which cannot be reclaimed immediately after. Both studies provide for 1 meter of cover to prevent water penetration.

Placer Dome considered mine dump in various areas to the east and south of the pit. Alamos did not prefer these areas as they are close (1.5 km) and upgradiant from the Mulatos River. Instead Alamos decided to put the mine dump in the same drainage as the pit, the Mulatos wash. It too drains into the Mulatos River but is further away. As well, unaffected upstream Mulatos wash runoff water is piped under the dump and around the pit in the Alamos plan.

The heap leach pad is the same as in the Placer Dome Feasibility and will be reclaimed in the same way by placing cover over it, after it reaches maximum height and it's decontamination by rising.

The other concern is the proximity of the village of Mulatos. Placer Dome and Kennecott had provided to move it in their 1997 Feasibility Study, although it was a much debated issue. Alamo has decided to leave it as is, for this Estrella Pit development. The difference is that Placer Dome was planning to mine the "Mina Viejo" area and Alamos is not. The Mina Viejo outcrop is between the village and the Alamos 'Estrella" pit.

25.7 TAXES

A description of the nature of taxes, royalties and other government levies or interests applicable to the mineral project or to production and to revenues or income from the mineral project.

The Mexican Federal income tax rate applicable in 2005 will be 32%, similar to the U.S. Capital write-off may be rapid or prolonged. The Feasibility Study assumes rapid depreciation.

A 10% tax on profits is distributable to company employees. For that reason, this and most Mexican mines have subcontracted labor. The subcontractor pays the tax. An allowance has been made when calculating subcontract labor rates for this provision.

A tax on the consumption of water, from any source has been included. A variable rebate from the Federal Government on diesel oil has been taken.

Although "NAFTA" now covers most equipment and material importation to Mexico, some is still subject to duty. Therefore, Alamos will apply for a "Pitex" exemption. The main conditions of this tax is the covenant by Alamos to export much of the gold and also to eventually export the equipment imported.

25.8 SLIDING SCALE

The Salamandra Property is subject to a Royalty for Technical Expertise Agreement dated March 23, 2001 between Minas de Oro Nacional and Minera San Augusto (the "RTE Agreement"). Pursuant to the RTE Agreement, the Company is obliged to pay to Tenedoramex and Kennecott, as assignees, the following royalty from the date of Commencement of Commercial Production (as defined in the RTE Agreement) until such time as the first 2,000,000 ounces of gold have been mined, processed and sold (or deemed sold) from the Salamandra Property:

(a)   2% of the Net Smelter Returns except gold and silver (as defined in the RTE Agreement) in respect of all Products (as defined in the RTE Agreement) mined and sold (or deemed sold) by Minas de Oro Nacional from the Salamandra Property; and

(b)   the applicable percentage based upon the Gold Price (as defined in the RTE Agreement) as published in the Wall Street Journal for the calendar quarter in which the royalty is payable of the Net Smelter Returns in respect of all Silver and Gold Products (as defined in the RTE Agreement) mined and sold (or deemed sold) by Minas de Oro Nacional from the Salamandra Property as follows:

Gold Price Range	Net Smelter Return Royalty 100% Basis
US$0.00/oz to US$299.99/oz	1.0%
US$300.00/oz to US$324.99/oz	1.5%
US$325.00/oz to US$349.99/oz	2.0%
US$350.00/oz to US$374.99/oz	3.0%
US$375.00/oz to US$399.99/oz	4.0%
US$400.00/oz or higher	5.0%

The term "Products" (as defined in the RTE Agreement) means ores, minerals, or other commercially valuable products, except any fraction therof comprising or deemed to comprise Gold and Silver Products, mined from the Salamadra Property.

The term "Gold and Silver Products"(as defined in the RTE Agreement") means ores, minerals, or other commercially valuable products containing

gold or silver mined from the Salamandra Property, provided that where such products contain a combination of gold and silver and other commercially viable metals or minerals, Gold and Silver Products shall be deemed to comprise on that fraction of such products as represents the proportionate commercial value of the gold and silver contained in such products, with the remaining fraction of such products deemed to be Products.

25.9 CAPITAL AND OPERATING COSTS

Capital and operating cost estimates with major components set out in tabular form.

Capital Cost (In US Dollars)

Initial Capital Cost $72,202,000
Sustaining Capital $10,284,000
Note: The initial capital is less than that shown in the table entitled "Total Project Cost Summary Sheet", because spare parts are shown as working capital.

Post Construction Capital Costs

A schedule for capital cost expenditure during the production period is presented. An estimated $10,284,000 in post-construction capital expenditures are anticipated. Major cost items include the following:

- Two additional 87 tonne haul trucks;

- Two 50 tonne utility trucks and one utility load in one year;

- Two replacement 87 tonne trucks in year 8;

- Major mine and other equipment rebuilds;

- Miscellaneous requirements.

Total Project Cost Summary Sheet

Plant Area	Description	Man-hours	Plant Equipment	Material	Labor	Subcontract	Construction Equipment	Total
	DIRECT COST							
100	Mining Capital	0	$14,925,000	$0	$0	$3,323,000	$0	$18,248,000
200	Crushing	154,133	$3,628,802	$631,961	$1,572,020	$2,772,626	$676,617	$9,282,026
300	Heap Leaching	299,802	$603,133	$1,201,422	$3,013,688	$2,968,684	$168,583	$7,955,510
400	Carbon Handling & Refinery	45,718	$1,297,410	$750,801	$485,218	$211,518	$76,229	$2,821,177
500	Ancillaries	87,737	$217,740	$1,052,428	$826,453	$3,342,271	$38,127	$5,477,019
600	Water System	40,448	$474,568	$688,932	$414,675	$391,159	$82,990	$2,052,324
700	Electrical	6,927	$1,625,000	$317,075	$103,900	$240,000	$0	$2,285,975
800	Camp	21,170	$0	$213,750	$227,550	$1,054,167	$0	$1,495,467
900	Townsite	28,376	$164,549	$233,056	$308,110	$76,133	$19,364	$801,213
	Subtotal DIRECT COST	684,311	$22,936,203	$5,089,425	$6,951,614	$14,379,557	$1,061,909	$50,418,710

NOTES: 1. Indirect Field Cost includes allocations as follows:
field payroll burden and overhead (included in labor); field supervision,
field supervisory burden, and support (included in labor); freight (included in direct cost);
and the estimated contractor field overhead cost (included in labor & unit rates).

2. Contractors' fee included in labor rate or unit cost, EPCM fee is $1,000,000.

3. Engineering included at 5% of Total Constructed Cost.

4. Home Office Services By Construction Manager/Engineer included at 2%
of Total Constructed Cost.

5. Construction Management Field Services included at 4% of Total Constructed Cost.

6. Contractor commissioning crew, refurbishment and vendor representatives are included
at $250,000. Spare Parts included at 5% of Plant Equipment .

7. Contingency included at 15% of Total Constructed Cost for process & infrastructure and 10% for mining

8. Added Owners Cost allocated by Owner for land acquisition, permitting and environmental
studies, owner's project administrative costs, mine development cost, operator training
cost, and all other Owner's Costs are included at $6,000,000.

9. All costs are in first quarter 2004 US dollars with 1.5% escalation added for six months.

10. Total Project Cost is projected to be in the range of -10% to +15%.

11. Initial reagents and consumables are not included.

Note: Construction Man-hours do not include subcontract hours.

IVA is not included as it is reimbursable.

TOTAL DIRECT FIELD COST	$50,418,700
TOTAL INDIRECT FIELD COST (1)	0
FEE - CONTRACTOR/EPCM (2)	1,000,000
TOTAL CONSTRUCTED COST	$51,418,700
ENGINEERING (3)	2,570,900
HOME OFFICE SERVICES BY CM/ENGR (4)	1,028,400
FIELD SERVICES (5)	2,056,700
TOTAL CONTRACTED COST	$57,074,700
COMMISSIONING AND SPARE PARTS (6)	1,396,800
CONTINGENCY (7)	7,712,598
ADDED OWNER'S COST (8)	6,080,000
TOTAL CONTRACTED AND OWNER'S COST	$72,264,098
ESCALATION (9)	1,083,961
TOTAL EVALUATED PROJECT COST (10)	$73,348,059

Average Cash Operating Cash Cost

Year 2 Of Operation

Processing Units Base Rate (tons/year ore)	3,500,000	
Mine Cost Area	**Total**	
	Annual Cost - $	**$/ton ore**
Mining Operations		
Drilling	$ 662,000	$ 0.1891
Blasting	1,558,000	0.4451
Loading	801,000	0.2289
Hauling	2,404,000	0.6869
Roads and Dumps (Auxiliary)	1,218,000	0.3480
Mining General	1,891,700	0.5405
Subtotal Mining	**$ 8,534,700**	**$ 2.4385**
Processing Operations		
Crushing Plant	$ 4,037,300	$ 1.1535
Heap Leaching	5,934,700	1.6956
Carbon Handling - Refinery	993,300	0.2838
Subtotal Processing	**$ 10,965,300**	**$ 3.1329**
Supporting Facilities		
Laboratory	$ 391,500	0.1119
Environmental Department	139,400	0.0398
General and Administrative	2,864,500	0.8184
Subtotal Supporting Facilities	**$ 3,395,400**	**$ 0.9701**
Total Mine Site Operating Cost	**$ 22,895,400**	**$ 6.5415**

25.10 ECONOMIC ANALYSIS

An economic analysis with cash flow forecasts on an annual basis using proven mineral reserves and probable mineral reserves only and sensitivity analysis with variants in metal prices.

Prices (In US Dollars)

Gold Price:	$350/oz
Silver Price:	$6.00/oz
Exchange Rate*:	NP$10= $1.00 U.S.

* This exchange rate was adopted in 2003. Since then the Peso has devalued to NP 11.4 = $ 1.00 U.S. However costs in northern Mexico follow the U.S. dollar, so little error is indicated.

Capital Cost (In US Dollars)

Initial Capital Cost $72,202,000
Sustaining Capital $10,284,000
Note: The initial capital is less than that shown on the table entitled "Total Project Cost Summary Sheet" because spare parts are shown as working capital.

Project Economics with 50% Alamos Equity (In US Dollars)

Net Present Value (NPV) at 0% $90,758,350
Net Present Value (NPV) at 7% $45,530,616
Net Present Value (NPV) at 10% $32,794,704
Internal Rate of Return   <u>23.3%</u>

Cash Cost $174.50/oz
Total Cost $239.20/oz
Operating Cost $6.54/tonne

The Project Economics Presume:
- 50% Equity Contributed by Alamos
- A sliding scale net smelter royalty (NSR) paid to "Placer/Kennecott"
- All Mexican labor is subcontracted
- After Mexican Taxes

Sensitivity Results

<table>
<tr><th colspan="7" align="center">Sensitivity Analysis</th></tr>
<tr>
<th>Case</th>
<th>Gold Price ($/oz)</th>
<th>Cumulative Net Cash Flow</th>
<th>NPV @7% (M$)</th>
<th>NPV @10% (M$)</th>
<th>Payback (yr)</th>
<th>IRR (%) CFI</th>
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<tr>
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<tr>
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<td>Capital Cost +10%</td>
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Payback
The payback of the capital cost occurs in 4 years (Please see the Financial Model Rev. 11).

Mine Life
The reserves will be exhausted in 10 ¼ years. Further drilling is expected to expand the pit beyond the Estrella area. In addition eight nearby deposits may ultimately be developed.

10,000 Metric Tons per Day

Production Statistics and Financial Analysis

Gold Price = $350   Silver Price = $6

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Days per Year		350	350	350	350	350	350	350	350	350	350	350	350	350	137				

Mine Production Statistics

Ore - Oxide

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Beginning (kt)	2,787,000	2,787,000	2,787,000	2,787,000	2,424,000	1,862,000	1,499,000	1,362,000	1,107,000	473,000	273,000	226,000	96,000	14,000	2,000	0
Mined (kt)	2,787,000	0	0	363,000	562,000	363,000	137,000	255,000	634,000	200,000	47,000	130,000	82,000	12,000	2,000	0
End (kt)	0	2,787,000	2,787,000	2,424,000	1,862,000	1,499,000	1,362,000	1,107,000	473,000	273,000	226,000	96,000	14,000	2,000	0	0
Contained Gold - grams	3,110,388	0	0	527,076	885,150	426,162	117,272	221,085	588,986	151,800	34,357	111,410	40,016	5,940	1,134	0
Contained Gold - ounces	99,999	0	0	16,945	28,458	13,701	3,770	7,108	18,936	4,880	1,105	3,582	1,287	191	36	0
Au Grade - g/t	1.1160	0.0000	0.0000	1.4520	1.5750	1.1740	0.8560	0.8670	0.9290	0.7590	0.7310	0.8570	0.4880	0.4950	0.5670	0.0000
Au Grade -oz/s.ton	0.0326	0.0000	0.0000	0.0424	0.0460	0.0343	0.0250	0.0253	0.0272	0.0222	0.0214	0.0250	0.0143	0.0145	0.0166	0.0000
Au Recovery Factor				96.97%	97.14%	96.51%	95.68%	95.73%	95.91%	95.26%	95.08%	95.57%	93.24%	93.33%	94.00%	
Contained Silver - grams	4,473,140	0	0	435,600	843,000	649,770	435,660	420,750	1,071,460	278,000	35,720	192,400	86,100	21,960	2,720	0
Contained Silver - ounces	143,811	0	0	14,005	27,102	20,890	14,006	13,527	34,447	8,938	1,148	6,186	2,768	706	87	0
Ag Grade - g/t	1.6050	0.0000	0.0000	1.2000	1.5000	1.7900	3.1800	1.6500	1.6900	1.3900	0.7600	1.4800	1.0500	1.8300	1.3600	
Ag Grade -oz/s.ton	0.0469	0.0000	0.0000	1.0909	1.3636	1.6273	2.8909	1.5000	1.5364	1.2636	0.6909	1.3455	0.9545	1.6636	1.2364	0.0000
Ag Recovery Factor				20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	

Ore - Mixed

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Beginning (kt)	9,076,000	9,076,000	9,076,000	9,076,000	9,022,000	8,540,000	7,498,000	6,723,000	5,140,000	4,050,000	3,227,000	2,458,000	1,425,000	540,000	69,000	0
Mined (kt)	9,076,000	0	0	54,000	482,000	1,042,000	775,000	1,583,000	1,090,000	823,000	769,000	1,033,000	885,000	471,000	69,000	0
End (kt)	0	9,076,000	9,076,000	9,022,000	8,540,000	7,498,000	6,723,000	5,140,000	4,050,000	3,227,000	2,458,000	1,425,000	540,000	69,000	0	0
Contained Gold - grams	14,883,115	0	0	148,284	1,057,990	2,286,148	1,457,775	2,562,877	1,597,940	920,114	1,171,956	1,512,312	1,436,355	640,560	90,804	0
Contained Gold - ounces	478,492	0	0	4,767	34,014	73,500	46,867	82,396	51,374	29,582	37,678	48,621	46,179	20,594	2,919	0
Au Grade - g/t	1.6398	0.0000	0.0000	2.7460	2.1950	2.1940	1.8810	1.6190	1.4660	1.1180	1.5240	1.4640	1.6230	1.3600	1.3160	
Au Grade -oz/s.ton	0.0479	0.0000	0.0000	0.0803	0.0642	0.0641	0.0550	0.0473	0.0428	0.0327	0.0445	0.0428	0.0474	0.0397	0.0385	0.0000
Au Recovery Factor				86.12%	84.94%	84.93%	83.94%	82.82%	81.95%	79.14%	82.30%	81.97%	82.82%	81.32%	80.93%	
Contained Silver - grams	27,053,860	0	0	187,920	1,648,440	1,990,220	1,581,000	4,052,480	2,932,100	2,024,580	1,684,110	1,311,910	3,380,700	5,063,250	1,197,150	0
Contained Silver - ounces	869,782	0	0	6,042	52,997	63,986	50,829	130,287	94,267	65,090	54,144	42,178	108,690	162,783	38,488	0
Ag Grade - g/t	2.9808	0.0000	0.0000	3.4800	3.4200	1.9100	2.0400	2.5600	2.6900	2.4600	2.1900	1.2700	3.8200	10.7500	17.3500	
Ag Grade -oz/s.ton	0.0871	0.0000	0.0000	3.1636	3.1091	1.7364	1.8545	2.3273	2.4455	2.2364	1.9909	1.1545	3.4727	9.7727	15.7727	0.0000
Ag Recovery Factor				20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	

Ore - Sulfide

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Beginning (kt)	24,505,000	24,505,000	24,505,000	24,505,000	24,403,000	22,466,000	20,371,000	17,783,000	16,121,000	14,345,000	11,868,000	9,184,000	6,847,000	4,314,000	1,297,000	0
Mined (kt)	24,505,000	0	0	102,000	1,937,000	2,095,000	2,588,000	1,662,000	1,776,000	2,477,000	2,684,000	2,337,000	2,533,000	3,017,000	1,297,000	0
End (kt)	0	24,505,000	24,505,000	24,403,000	22,466,000	20,371,000	17,783,000	16,121,000	14,345,000	11,868,000	9,184,000	6,847,000	4,314,000	1,297,000	0	0
Contained Gold - grams	41,537,479	0	0	192,780	4,028,960	3,961,645	4,490,180	3,254,196	3,228,768	4,260,440	4,001,844	3,725,178	3,604,459	4,999,169	1,789,860	0
Contained Gold - ounces	1,335,430	0	0	6,198	129,531	127,367	144,359	104,622	103,805	136,973	128,659	119,764	115,883	160,723	57,544	0
Au Grade - g/t	1.6951	0.0000	0.0000	1.8900	2.0800	1.8910	1.7350	1.9580	1.8180	1.7200	1.4910	1.5940	1.4230	1.6570	1.3800	
Au Grade -oz/s.ton	0.0495	0.0000	0.0000	0.0552	0.0608	0.0553	0.0507	0.0572	0.0531	0.0503	0.0436	0.0466	0.0416	0.0484	0.0403	0.0000
Au Recovery Factor				68.22%	68.67%	68.23%	67.76%	68.40%	68.01%	67.70%	66.80%	67.21%	66.52%	67.51%	66.26%	
Contained Silver - grams	84,754,920	0	0	228,480	5,501,080	4,692,800	6,030,040	4,254,720	7,068,480	10,230,010	9,984,480	6,333,270	7,193,720	18,192,510	5,045,330	0
Contained Silver - ounces	2,724,871	0	0	7,346	176,860	150,874	193,866	136,789	227,252	328,895	321,001	203,615	231,278	584,889	162,207	0
Ag Grade - g/t	3.4587	0.0000	0.0000	2.2400	2.8400	2.2400	2.3300	2.5600	3.9800	4.1300	3.7200	2.7100	2.8400	6.0300	3.8900	
Ag Grade -oz/s.ton	0.1011	0.0000	0.0000	2.0364	2.5818	2.0364	2.1182	2.3273	3.6182	3.7545	3.3818	2.4636	2.5818	5.4818	3.5364	0.0000
Ag Recovery Factor				20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	20.00%	

Ore - Combined

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Beginning (kt)	36,368,000	36,368,000	36,368,000	36,368,000	35,849,000	32,868,000	29,368,000	25,868,000	22,368,000	18,868,000	15,368,000	11,868,000	8,368,000	4,868,000	1,368,000	0
Mined (kt)	36,368,000	0	0	519,000	2,981,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	1,368,000	0
End (kt)	0	36,368,000	36,368,000	35,849,000	32,868,000	29,368,000	25,868,000	22,368,000	18,868,000	15,368,000	11,868,000	8,368,000	4,868,000	1,368,000	0	0
Contained Gold - grams	59,530,982	0	0	868,140	5,972,100	6,673,955	6,065,227	6,038,158	5,415,694	5,332,354	5,208,157	5,348,900	5,080,830	5,645,669	1,881,798	0
Contained Gold - ounces	1,913,921	0	0	27,911	192,003	214,568	194,997	194,127	174,115	171,435	167,442	171,967	163,349	181,508	60,500	0
Au Grade - g/t	1.6369	0.0000	0.0000	1.6727	2.0034	1.9068	1.7329	1.7252	1.5473	1.5235	1.4880	1.5283	1.4517	1.6130	1.3756	0.0000
Au Grade -oz/s.ton	0.0478	0.0000	0.0000	0.0489	0.0586	0.0557	0.0506	0.0504	0.0452	0.0445	0.0435	0.0447	0.0424	0.0471	0.0402	0.0000
Recoverable Au - grams	43,424,052	0	0	770,322	4,525,178	5,055,945	4,378,408	4,560,089	4,070,293	3,757,101	3,670,418	3,849,809	3,624,586	3,901,386	1,260,515	0
Recoverable Au - ounces	1,396,083	0	0	24,766	145,484	162,549	140,766	146,607	130,860	120,791	118,004	123,771	116,530	125,430	40,526	0
Au Recovery Factor	72.94%			88.73%	75.77%	75.76%	72.19%	75.52%	75.16%	70.46%	70.47%	71.97%	71.34%	69.10%	66.98%	
Contained Silver - grams	116,281,920	0	0	852,000	7,992,520	7,332,790	8,046,700	8,727,950	11,072,040	12,532,590	11,704,310	7,837,580	10,660,520	23,277,720	6,245,200	0
Contained Silver - ounces	3,738,464	0	0	27,392	256,960	235,749	258,701	280,604	355,966	402,923	376,294	251,978	342,736	748,379	200,783	0
Ag Grade - g/t	3.1974	0.0000	0.0000	1.6416	2.6812	2.0951	2.2991	2.4937	3.1634	3.5807	3.3441	2.2393	3.0459	6.6508	4.5652	0.0000
Ag Grade -oz/s.ton	0.0935	0.0000	0.0000	0.0480	0.0784	0.0612	0.0672	0.0729	0.0925	0.1047	0.0977	0.0654	0.0890	0.1944	0.1334	0.0000
Recoverable Ag - grams	23,256,384	0	0	170,400	1,598,504	1,466,558	1,609,340	1,745,590	2,214,408	2,506,518	2,340,862	1,567,516	2,132,104	4,655,544	1,249,040	
Recoverable Ag - ounces	747,693	0	0	5,478	51,392	47,150	51,740	56,121	71,193	80,585	75,259	50,396	68,547	149,676	40,157	
Ag Recovery Factor	20%			20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	20%	

Waste

Waste 1

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Beginning (kt)	50,793,000	50,793,000	50,793,000	50,793,000	48,311,000	43,292,000	37,769,000	32,197,000	26,464,000	20,786,000	15,870,000	11,084,000	7,891,000	4,723,000	1,077,000	(769,000)
Mined (kt)	51,562,000	0	0	2,482,000	5,019,000	5,523,000	5,572,000	5,733,000	5,678,000	4,916,000	4,786,000	3,193,000	3,168,000	3,646,000	1,846,000	0
End (kt)	(769,000)	50,793,000	50,793,000	48,311,000	43,292,000	37,769,000	32,197,000	26,464,000	20,786,000	15,870,000	11,084,000	7,891,000	4,723,000	1,077,000	(769,000)	(769,000)
Total Waste Mined (kt)	51,562,000	0	0	2,482,000	5,019,000	5,523,000	5,572,000	5,733,000	5,678,000	4,916,000	4,786,000	3,193,000	3,168,000	3,646,000	1,846,000	0

Stripping Ratios:

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Waste:Ore	1.40	0.00	0.00	4.78	1.68	1.58	1.59	1.64	1.62	1.40	1.37	0.91	0.91	1.04	1.35	0.00
Total Mine Waste:Ore	1.40	0.00	0.00	4.78	1.68	1.58	1.59	1.64	1.62	1.40	1.37	0.91	0.91	1.04	1.35	0.00

Total Material Mined (kt) — **Material Mined**

Item	Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12
Total Material Mined (kt)	87,930,000	0	0	3,001,000	8,000,000	9,023,000	9,072,000	9,233,000	9,178,000	8,416,000	8,286,000	6,693,000	6,668,000	7,146,000	3,214,000	0
Ore - TPD		0	0	1,483	8,517	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	10,000	9,985	0
Waste		0	0	7,091	14,340	15,780	15,920	16,380	16,223	14,046	13,674	9,123	9,051	10,417	13,474	0
Total - TPD		0	0	8,574	22,857	25,780	25,920	26,380	26,223	24,046	23,674	19,123	19,051	20,417	23,460	0

Production Statistics and Financial Analysis

		Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
						Gold Price =	$350		Silver Price =	$6										

Crushing Plant Production Statistics

Ore Stockpile:	Beginning Ore (kt)	0	0	0	0	519,000	0	0	0	0	0	0	0	0	0	0	0			
	Mined (kt)	36,368,000	0	0	519,000	2,981,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	1,368,000	0	0	
	Mined to Stockpile (kt)	519,000	0	0	519,000	0	0	0	0	0	0	0	0	0	0	0	0		
	Crushed - from Mine (kt)	35,849,000	0	0	0	2,981,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	1,368,000	0		
	Crushed - from Stockpile (kt)	519,000	0	0	0	519,000	0	0	0	0	0	0	0	0	0	0	0		
	Ending Balance (kt)	0	0	0	519,000	0	0	0	0	0	0	0	0	0	0	0	0		
Total Crushed	(kt)	36,368,000	0	0	0	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	3,500,000	1,368,000	0		
Grade Mined to Crushing Plant	g/t	1.6369	0.0000	0.0000	1.6727	2.0034	1.9068	1.7329	1.7252	1.5473	1.5235	1.4880	1.5283	1.4517	1.6130	1.3756	0.0000		
	oz/st	0.0478	0.0000	0.0000	0.0489	0.0586	0.0557	0.0506	0.0504	0.0452	0.0445	0.0435	0.0447	0.0424	0.0471	0.0402	0.0000		
Grade Mined to S/P	g/t	1.6369	0.0000	0.0000	1.6727	2.0034	1.9068	1.7329	1.7252	1.5473	1.5235	1.4880	1.5283	1.4517	1.6130	1.3756	0.0000		
	oz/st	0.0478	0.0000	0.0000	0.0489	0.0586	0.0557	0.0506	0.0504	0.0452	0.0445	0.0435	0.0447	0.0424	0.0471	0.0402	0.0000		
Overall Grade Mined	g/t	1.6369	0.0000	0.0000	1.6727	2.0034	1.9068	1.7329	1.7252	1.5473	1.5235	1.4880	1.5283	1.4517	1.6130	1.3756	0.0000		
	oz/st	0.0478	0.0000	0.0000	0.0489	0.0586	0.0557	0.0506	0.0504	0.0452	0.0445	0.0435	0.0447	0.0424	0.0471	0.0402	0.0000		
Stockpile Grade	Beginning Grade (g/t)		0.0000	0.0000	0.0000	1.6727	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000		
	Grade to Stockpile (g/t)	1.6727	0.0000	0.0000	1.6727	2.0034	1.9068	1.7329	1.7252	1.5473	1.5235	1.4880	1.5283	1.4517	1.6130	1.3756	0.0000		
	Grade Crushed from SP (g/t)	1.6727	0.0000	0.0000	1.6727	1.6727	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000		
	Ending Grade (g/t)		0.0000	0.0000	1.6727	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000	0.0000		
Overall Grade Crushed	g/t		0.0000	0.0000	0.0000	1.9544	1.9068	1.7329	1.7252	1.5473	1.5235	1.4880	1.5283	1.4517	1.6130	1.3756	0.0000		
	oz/st		0.000	0.000	0.000	0.057	0.056	0.051	0.050	0.045	0.045	0.043	0.045	0.042	0.047	0.040	0.000		
Contained Metal- Mined Ore	Gold, oz's	1,913,921	0	0	27,911	192,003	214,568	194,997	194,127	174,115	171,435	167,442	171,967	163,349	181,508	60,500	0		
Contained Metal - Crushed Ore	Gold, oz's	1,913,921				219,914	214,568	194,997	194,127	174,115	171,435	167,442	171,967	163,349	181,508	60,500	0		
Gold Recovery Factor		72.94%																	
Silver Production, ounces WIP		747,693				56,870	47,150	51,740	56,121	71,193	80,585	75,259	50,396	68,547	149,676	40,157	0		
Silver Credit -Equiv Gold Produced, oz's		12,818				975	808	887	962	1,220	1,381	1,290	864	1,175	2,566	688	0		
Gold Production, ounces WIP		1,396,083				170,250.3	162,548.6	140,765.8	146,606.9	130,859.9	120,790.8	118,003.9	123,771.4	116,530.4	125,429.6	40,525.6			
Equiv (Gold+Silver) Production, ounces WIP		1,408,901				171,225.3	163,356.9	141,652.8	147,568.9	132,080.4	122,172.2	119,294.1	124,635.3	117,705.5	127,995.4	41,214.0			
Gold Production, ounces sold	Gold, oz's	1,408,901				114,150.2	165,979.7	148,887.5	145,596.9	137,243.2	125,475.0	120,253.5	122,854.9	120,015.5	124,565.5	83,879.1			

Cash Flow and Economic Indicators

		Total	PPN-3	PPN-2	PPN-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Capital Cost - $'s																				
Initial Capital - Equity		$ 36,674,000			$ 36,674,000															
Initial Capital - Loan					$ 36,674,000															
Sustaining Capital																				
Mobile Equipment		$ 7,884,000				$ 4,251,000	$ -	$ 100,000	$ 200,000	$ -	$ 975,000	$ 174,000	$ 2,184,000	$ -	$ -					
Other		$ 2,400,000				$ 450,000	$ 475,000	$ 480,000	$ 500,000	$ 95,000	$ 50,000	$ 75,000	$ 80,000	$ 100,000	$ 95,000					
Total Capital (Initial + Sustaining)		$ 82,486,000		$ -	$ 72,202,000	$ 4,701,000	$ 475,000	$ 580,000	$ 700,000	$ 95,000	$ 1,025,000	$ 249,000	$ 2,264,000	$ 100,000	$ 95,000	$ -	$ -	$ -	$ -	$ -
Salvage Value		$ (6,000,000)															$ (6,000,000)			
Working Capital																				
Metal WIP and Finished Goods (1 /2- month)		$ -	$ -	$ -	$ -	$ 1,665,000	$ 756,000	$ (250,000)	$ (48,000)	$ (122,000)	$ (171,000)	$ (76,000)	$ 38,000	$ (42,000)	$ 67,000	$ (594,000)	$ (1,223,000)	$ -	$ -	$ -
Inventory - Parts, Supplies and Comodities		$ -			$ 1,146,800											$ (1,146,800)				
Total Working Capital		$ -		$ -	$ 1,146,800	$ 1,665,000	$ 756,000	$ (250,000)	$ (48,000)	$ (122,000)	$ (171,000)	$ (76,000)	$ 38,000	$ (42,000)	$ 67,000	$ (1,740,800)	$ (1,223,000)	$ -	$ -	$ -
Revenue - $'s	Gold Sales $'s	$ 493,116,000	$ -	$ -	$ -	$ 39,953,000	$ 58,093,000	$ 52,111,000	$ 50,959,000	$ 48,035,000	$ 43,916,000	$ 42,089,000	$ 42,999,000	$ 42,005,000	$ 43,598,000	$ 29,358,000	$ -	$ -	$ -	$ -
	Average price realized	$ 350				$ 350	$ 350	$ 350	$ 350	$ 350	$ 350	$ 350	$ 350	$ 350	$ 350	$ 350				
Cash Operating Costs - $'s																				
$ 2.40 Mine	Operations	$ 87,355,700	$ -	$ -	$ 3,323,000	$ 7,886,700	$ 8,534,700	$ 8,159,700	$ 7,967,700	$ 8,174,700	$ 7,865,700	$ 8,110,700	$ 6,951,700	$ 7,311,700	$ 8,160,700	$ 4,908,700	$ -	$ -	$ -	$ -
$ 3.29 Processing	Operations	$ 119,548,826			$ 165,900	$ 9,822,300	$ 9,852,300	$ 9,954,300	$ 15,381,500	$ 9,782,300	$ 20,225,700	$ 9,976,300	$ 9,906,300	$ 9,945,300	$ 10,040,300	$ 4,496,326				
$ 0.99 Other	G & A	$ 36,117,200			$ 252,100	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 3,395,400	$ 1,911,100				
$ 2.00 Shipping & Refining		$ 2,818,000	$ -	$ -	$ -	$ 228,000	$ 332,000	$ 298,000	$ 291,000	$ 274,000	$ 251,000	$ 241,000	$ 246,000	$ 240,000	$ 249,000	$ 168,000				
Total Cash Operating Costs		$ 245,839,726	$ -	$ -	$ 3,741,000	$ 21,332,400	$ 22,114,400	$ 21,807,400	$ 27,035,600	$ 21,626,400	$ 31,737,800	$ 21,723,400	$ 20,499,400	$ 20,892,400	$ 21,845,400	$ 11,484,126	$ -	$ -	$ -	$ -
Total Cash Costs - $'s																				
Total Cash Operating Costs		$ 245,839,726	$ -	$ -	$ 3,741,000	$ 21,332,400	$ 22,114,400	$ 21,807,400	$ 27,035,600	$ 21,626,400	$ 31,737,800	$ 21,723,400	$ 20,499,400	$ 20,892,400	$ 21,845,400	$ 11,484,126	$ -	$ -	$ -	$ -
Employee Profit Sharing	0	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -				
Payment to Ejido		$ 2,800,000	$ -	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ 200,000	$ -		
Property Tax		$ 650,000		$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000	$ 50,000				
Total Cash Costs		$ 249,289,726	$ -	$ 250,000	$ 3,991,000	$ 21,582,400	$ 22,364,400	$ 22,057,400	$ 27,285,600	$ 21,876,400	$ 31,987,800	$ 21,973,400	$ 20,749,400	$ 21,142,400	$ 22,095,400	$ 11,734,126	$ 200,000	$ -	$ -	$ -
Total Production Costs - $'s																				
Total Cash Costs		$ 249,289,726	$ -	$ 250,000	$ 3,991,000	$ 21,582,400	$ 22,364,400	$ 22,057,400	$ 27,285,600	$ 21,876,400	$ 31,987,800	$ 21,973,400	$ 20,749,400	$ 21,142,400	$ 22,095,400	$ 11,734,126	$ 200,000			
Reclamation & closure bond fee		$ 2,750,000				$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000	$ 250,000				
Reclamation & closure		$ 2,480,500	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,860,400	$ 620,100	$ -	$ -
Depreciation		$ 82,489,000	$ -	$ -	$ 18,051,000	$ 19,226,000	$ 19,345,000	$ 19,490,000	$ 1,614,000	$ 463,000	$ 600,000	$ 517,000	$ 908,000	$ 910,000	$ 677,000	$ 615,000	$ 49,000	$ 24,000	$ -	$ -
Total Production Costs		$ 337,009,226	$ -	$ 250,000	$ 22,042,000	$ 41,058,400	$ 41,959,400	$ 41,797,400	$ 29,149,600	$ 22,589,400	$ 32,837,800	$ 22,740,400	$ 21,907,400	$ 22,302,400	$ 23,022,400	$ 12,599,126	$ 2,109,400	$ 644,100	$ -	$ -
Loan Interest Payments		$ 7,099,200				$ 2,746,900	$ 2,128,200	$ 1,466,200	$ 757,900											
Income from Operations - $'s		$ 149,007,574	$ -	$ (250,000)	$ (22,042,000)	$ (3,852,300)	$ 14,005,400	$ 8,847,400	$ 21,051,500	$ 25,445,600	$ 11,078,200	$ 19,348,600	$ 21,091,600	$ 19,702,600	$ 20,575,600	$ 16,758,874	$ (2,109,400)	$ (644,100)	$ -	$ -
Tax loss carry forward applied		$ -	$ -	$ (250,000)	$ (22,292,000)	$ (26,144,300)	$ (12,138,900)	$ (3,291,500)	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	
Taxes at 32% with loss carry forward applied		$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 5,683,200	$ 8,142,592	$ 3,545,024	$ 6,191,552	$ 6,749,312	$ 6,304,832	$ 6,584,192	$ 5,362,840	$ -	$ -	$ -	
Net Income After Taxes - $'s		$ -	$ -	$ (250,000)	$ (22,042,000)	$ (3,852,300)	$ 14,005,400	$ 8,847,400	$ 15,368,300	$ 17,303,008	$ 7,533,176	$ 13,157,048	$ 14,342,288	$ 13,397,768	$ 13,991,408	$ 11,396,034	$ (2,109,400)	$ (644,100)	$ -	$ -
Royalty per RTE Agreement		$ 11,975,480				$ 970,590	$ 1,410,790	$ 1,265,330	$ 1,237,770	$ 1,167,050	$ 1,066,480	$ 1,021,670	$ 1,043,970	$ 1,020,150	$ 1,058,940	$ 712,740	$ -	$ -	$ -	$ -
Loan Payment - Principal						$ 8,838,200	$ 9,456,900	$ 10,118,900	$ 10,827,200											
Cash Flow from Property after Tax after Royalty		$ 90,758,350	$ -	$ (250,000)	$ (41,811,800)	$ (801,090)	$ 21,251,710	$ 16,623,170	$ 4,265,330	$ 16,625,958	$ 6,212,696	$ 12,479,378	$ 11,904,318	$ 13,229,618	$ 13,447,468	$ 13,039,094	$ 5,162,600	$ (620,100)	$ -	$ -
	Cumulative		$ -	$ (250,000)	$ (42,061,800)	$ (42,862,890)	$ (21,611,180)	$ (4,988,010)	$ (722,680)	$ 15,903,278	$ 22,115,974	$ 34,595,352	$ 46,499,670	$ 59,729,288	$ 73,176,756	$ 86,215,850	$ 91,378,450	$ 90,758,350	$ 90,758,350	$ 90,758,350
Payback - Years from Startup		4.0				1	1	1	1.0	0	0	0	0	0	0	0	0	0	0	0

Property Economics:

Economic Statistics

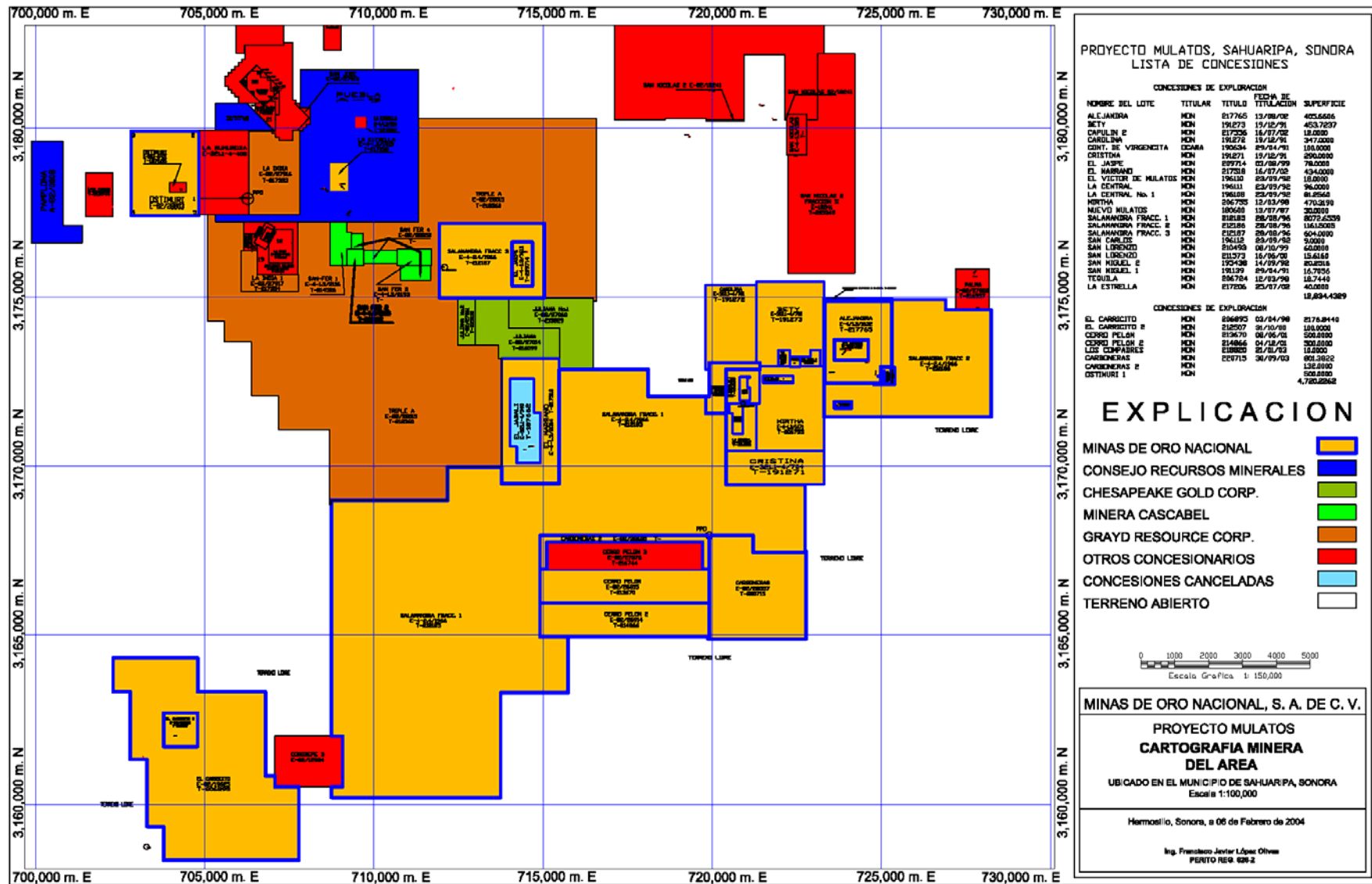| NPV Discount Rate	NPV $'s (000's) After Tax
0%	$90,758,350
7%	$45,530,616
10%	$32,794,204
IRR	23.3%

Figure 1.1
Mulatos Deposit Identification

Figure 1.2
Mulatos Project Location Map
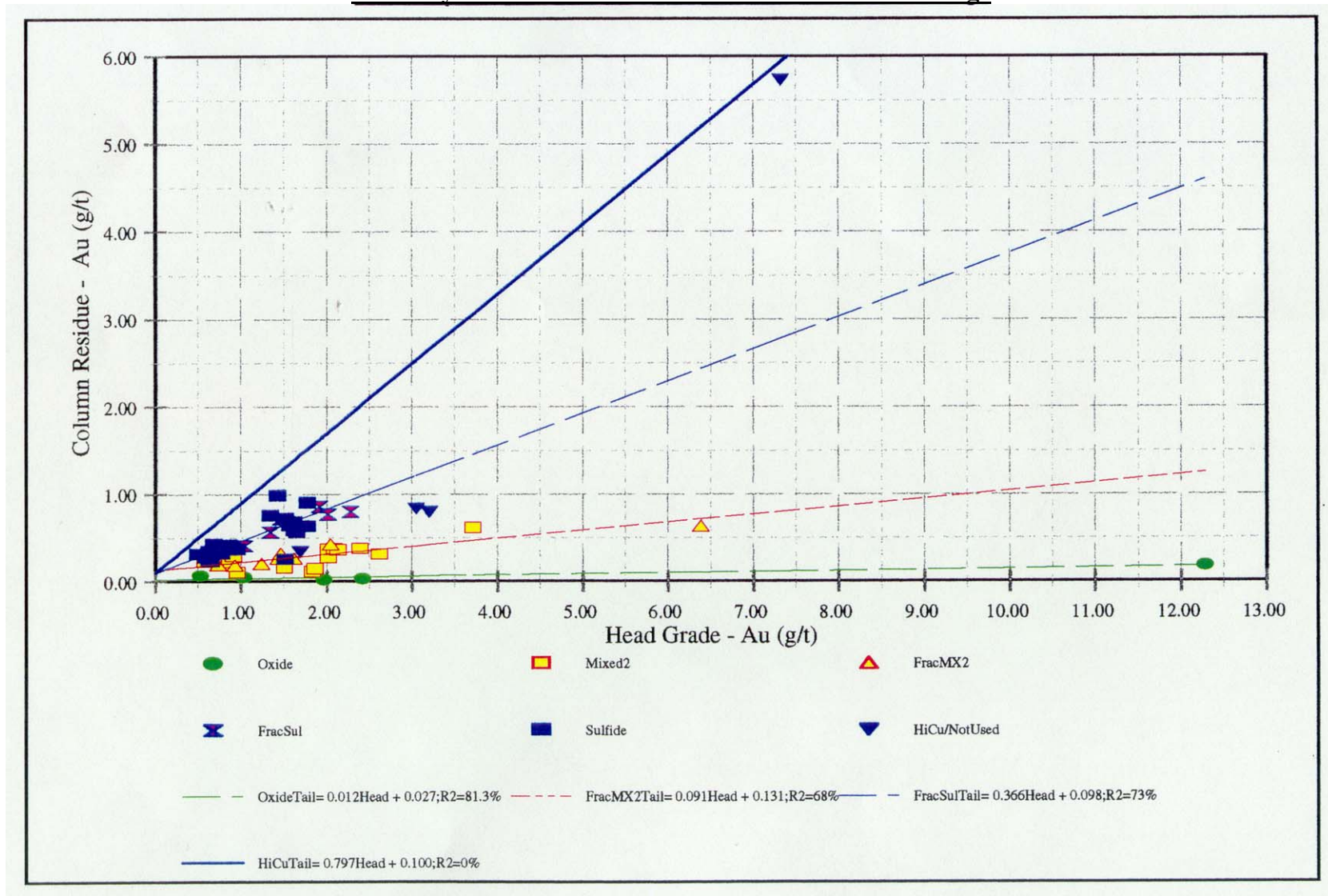
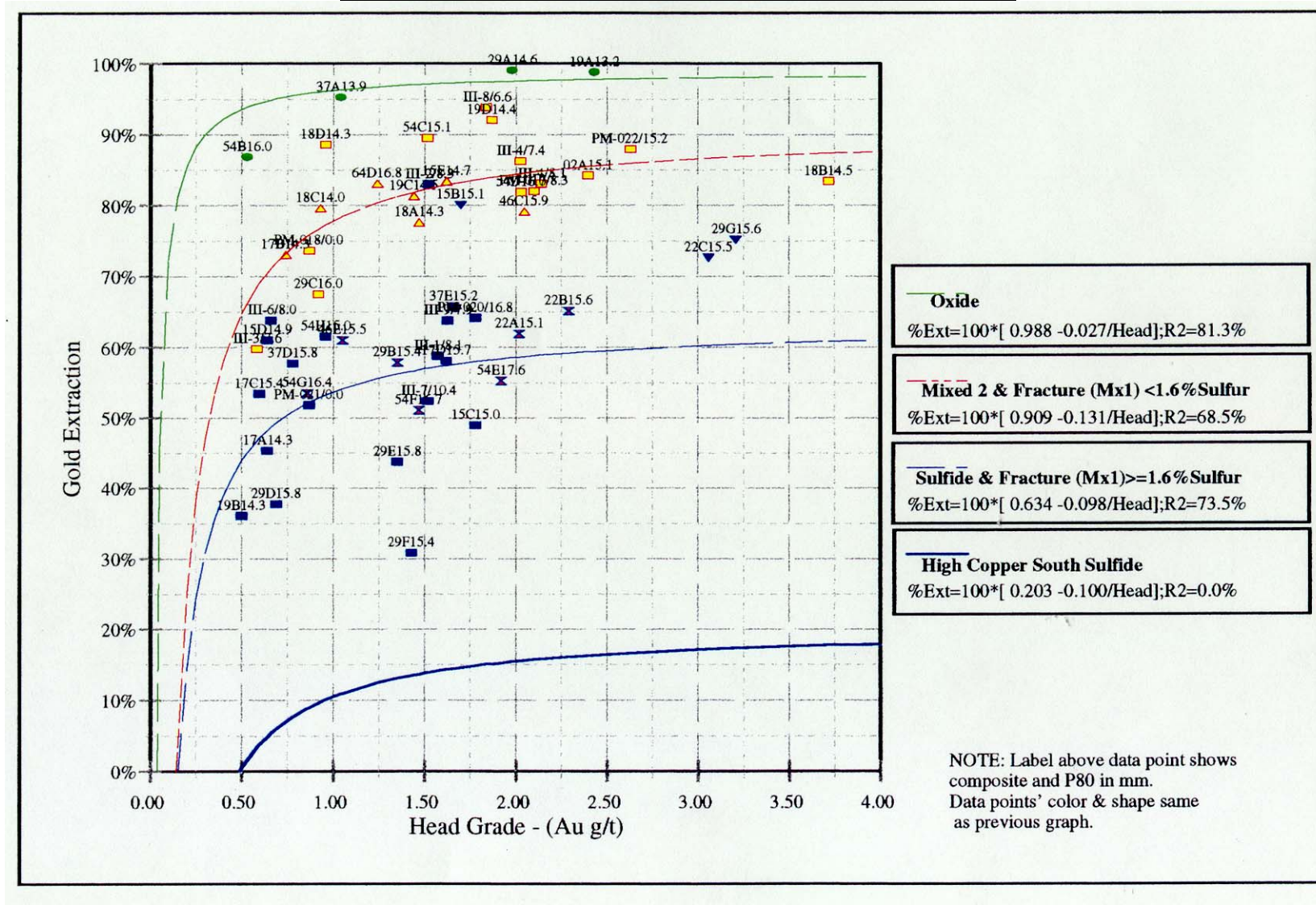# Figure 1.6
Claims Map

Figure 1.7
District Claim Map

Figure 3.1
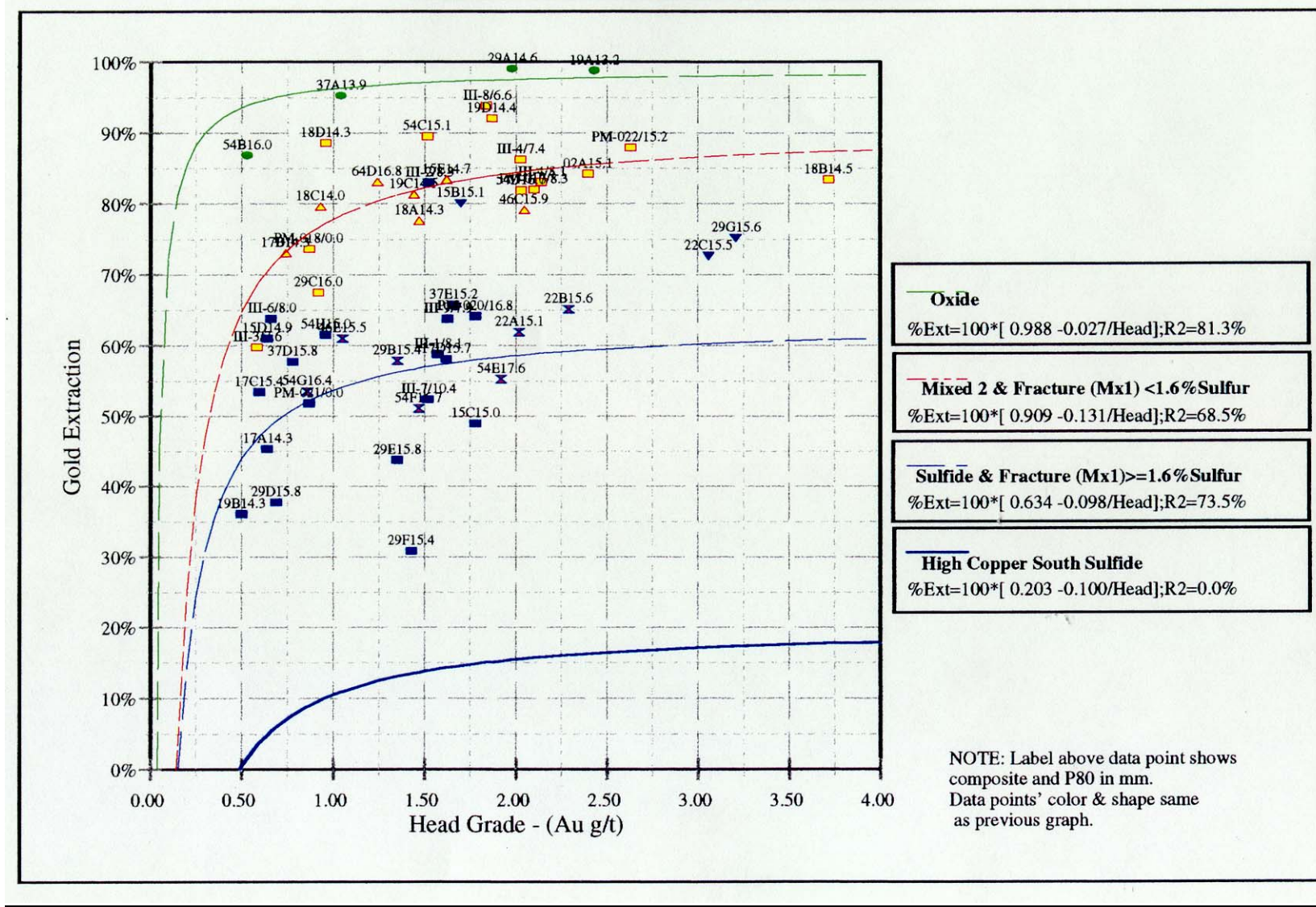**Recovery Model - Residue Versus Head Gold – Full Range**

Figure 3.2
## Recovery Model - Residue Versus Head Gold – Partial Range

Oxide
%Ext=100*[0.988 -0.027/Head];R2=81.3%

Mixed 2 & Fracture (Mx1) <1.6%Sulfur
%Ext=100*[0.909 -0.131/Head];R2=68.5%

Sulfide & Fracture (Mx1)>=1.6%Sulfur
%Ext=100*[0.634 -0.098/Head];R2=73.5%

High Copper South Sulfide
%Ext=100*[0.203 -0.100/Head];R2=0.0%

NOTE: Label above data point shows
composite and P80 in mm.
Data points' color & shape same
as previous graph.

Figure 3.3
Recovery Model – Extraction Versus Head Gold

Figure 3.4
Location Map; Metallurgical Drill Holes

Figure 1.17
LOCATION MAP
METALLURGICAL DRILL HOLES
MULATOS PROJECT
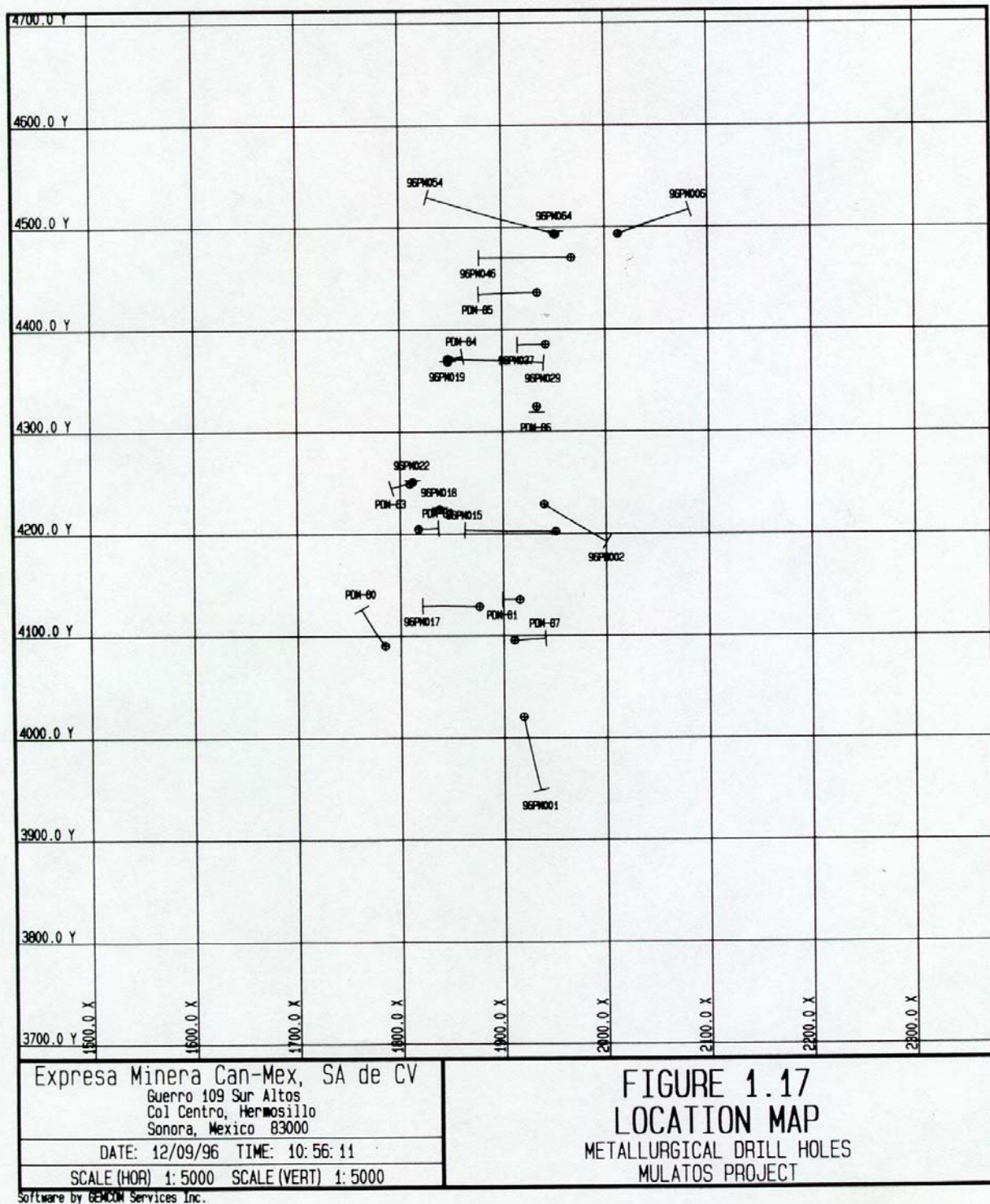
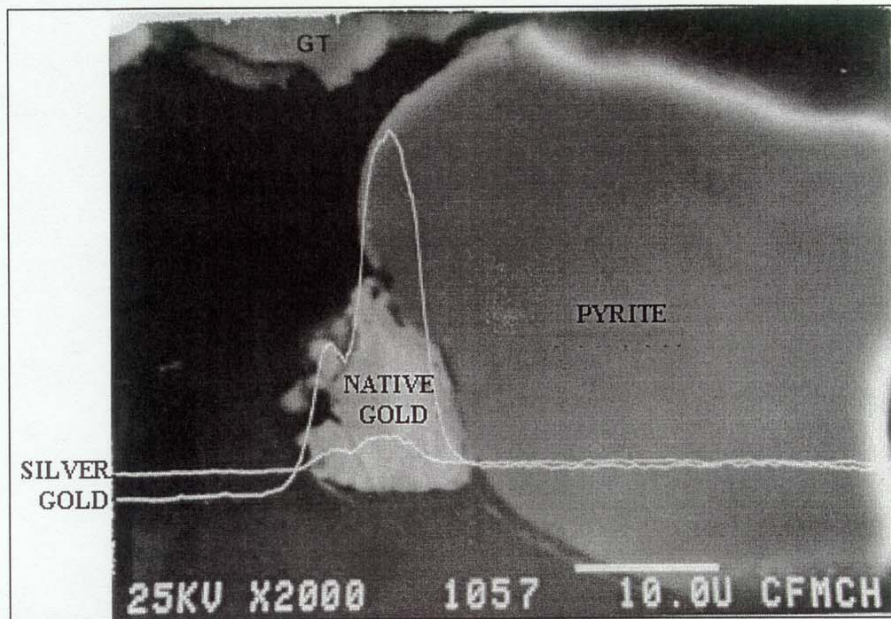# Figure 3.5
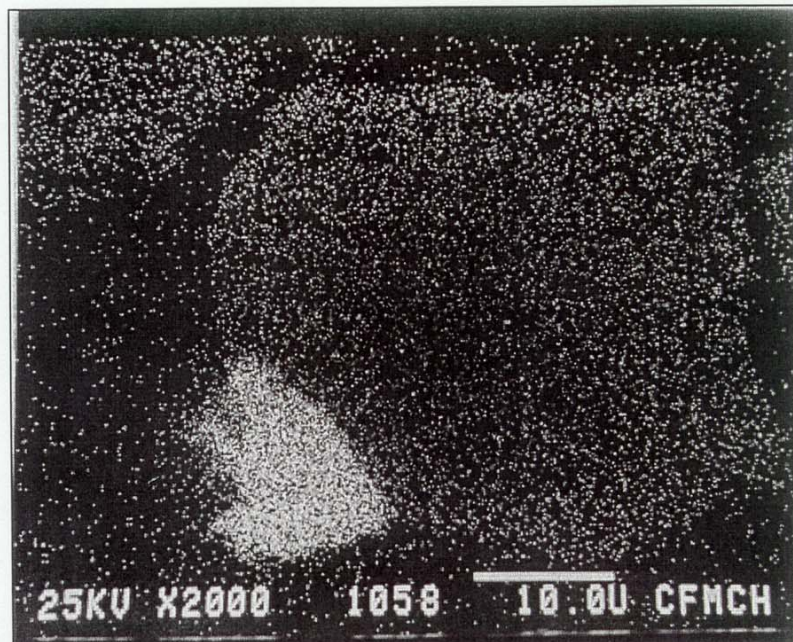## Photomicrographs of Gold and Pyrite Particles
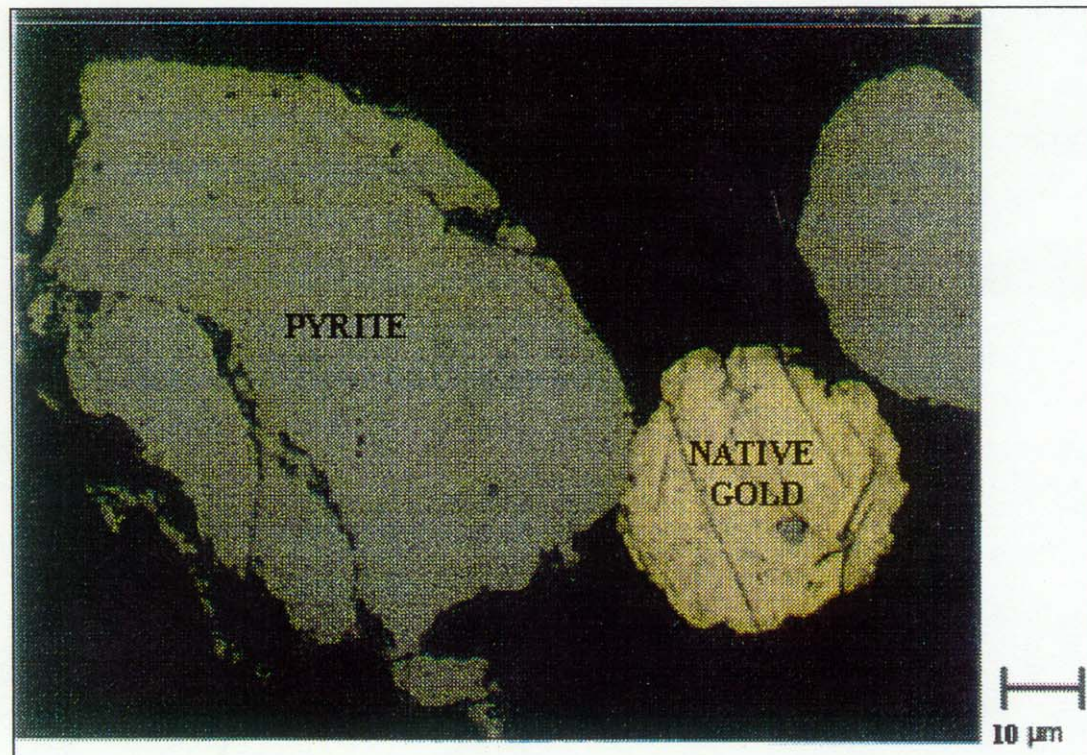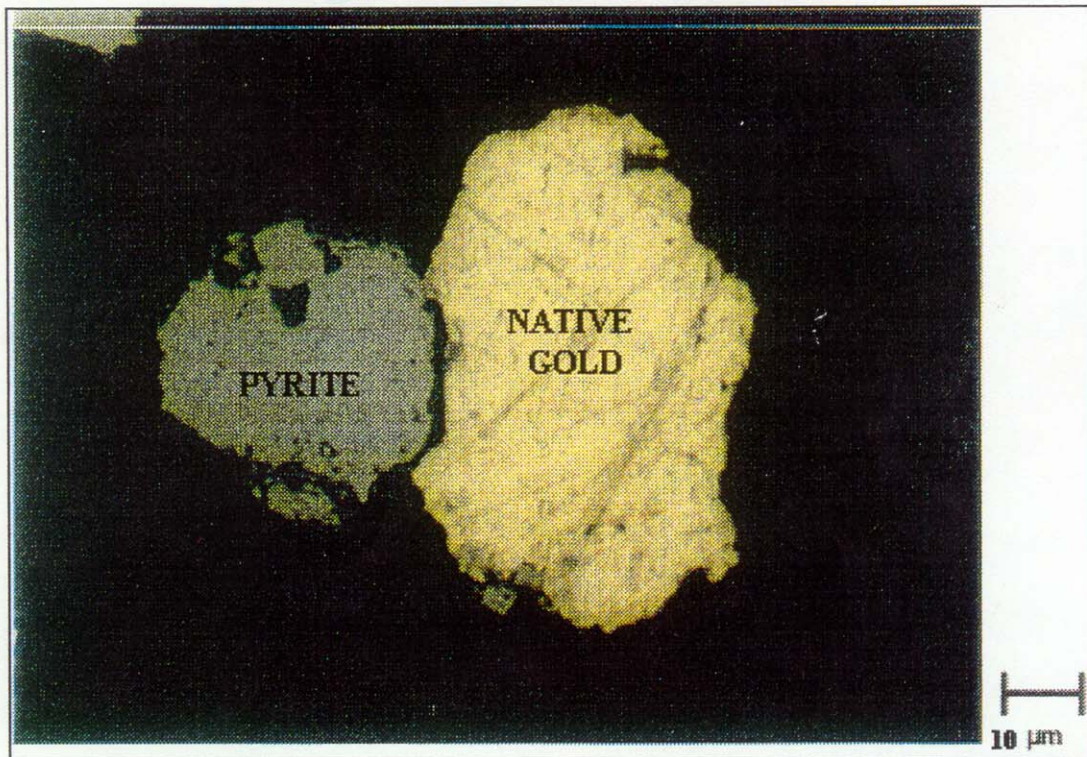
Figure 3.6
Photomicrographs of Morphological Pyrite Types
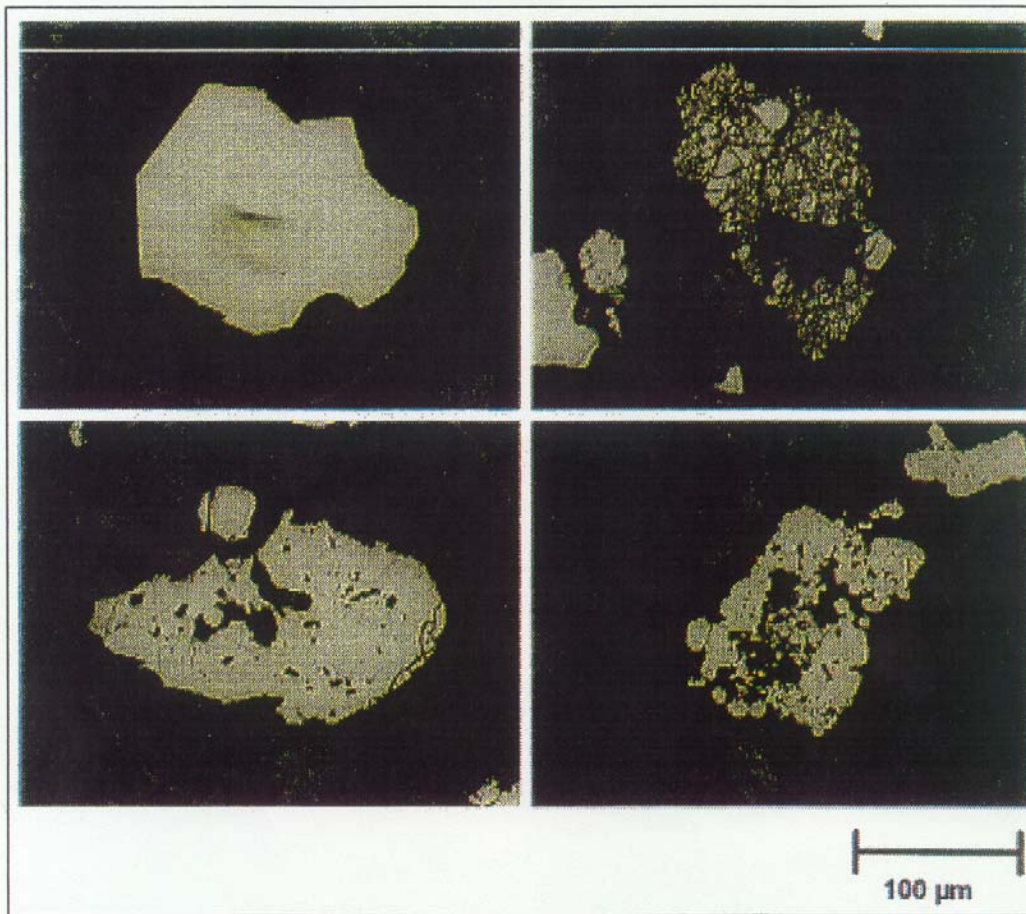
Relative concentrations of gold and silver in native gold particle. Scale shown is 10 μm.

X-ray image of top photo showing distribution of gold (white dots) in pyrite and native gold particles. Scale shown is 10 μm.

Figure 3.7
Photomicrographs of Gold and Silver Distributions

100 µm

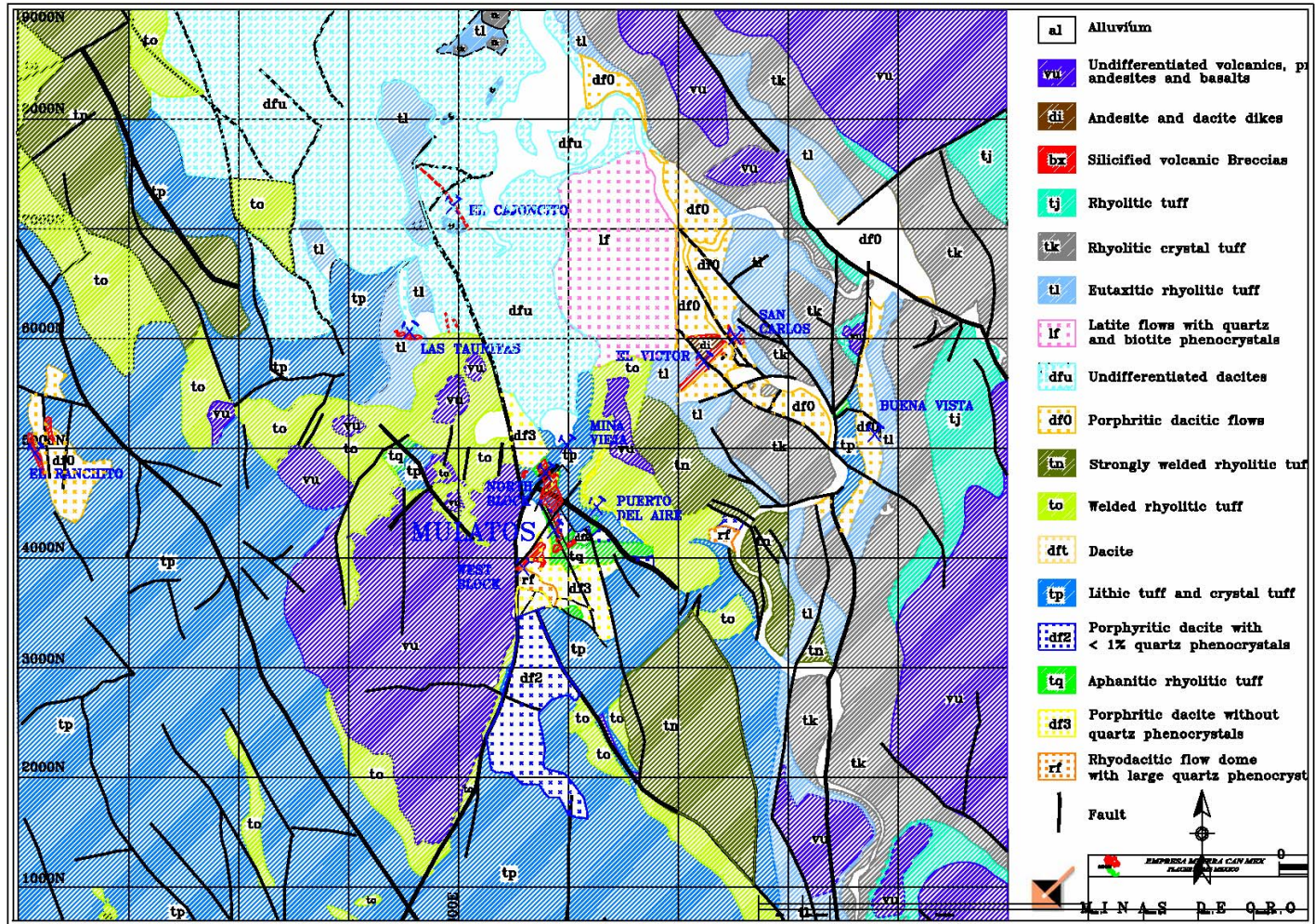Top Left: Coarse-grained
Bottom Left: With dissolution features

Top Right: Mylonitic or fractured
Bottom Right: Fine-grained

Gold concentration (ppm) in pyrite crystal structure.

Sample	Coarse-grained	Framboidal, fracture, and with dissolution features	Fine-grained
III-1 NZS Silicified Sulfide	0.34 ± 0.14	4.1 ± 2.0	5.5 ± 2.3
III-5 SZS High Cu High Au Silicified Sulfide	0.32 ± 0.16	23.2 ± 12.8	43.0 ± 4.1
III-7 Argillized Sulfide	0.24 ± 0.10	5.8 ± 2.5	10.1 ± 2.5
III-9 Moderate Copper Sulfide	0.21 ± 0.10	0.7 ± 0.3	8.6 ± 5.0

Figure 9.1
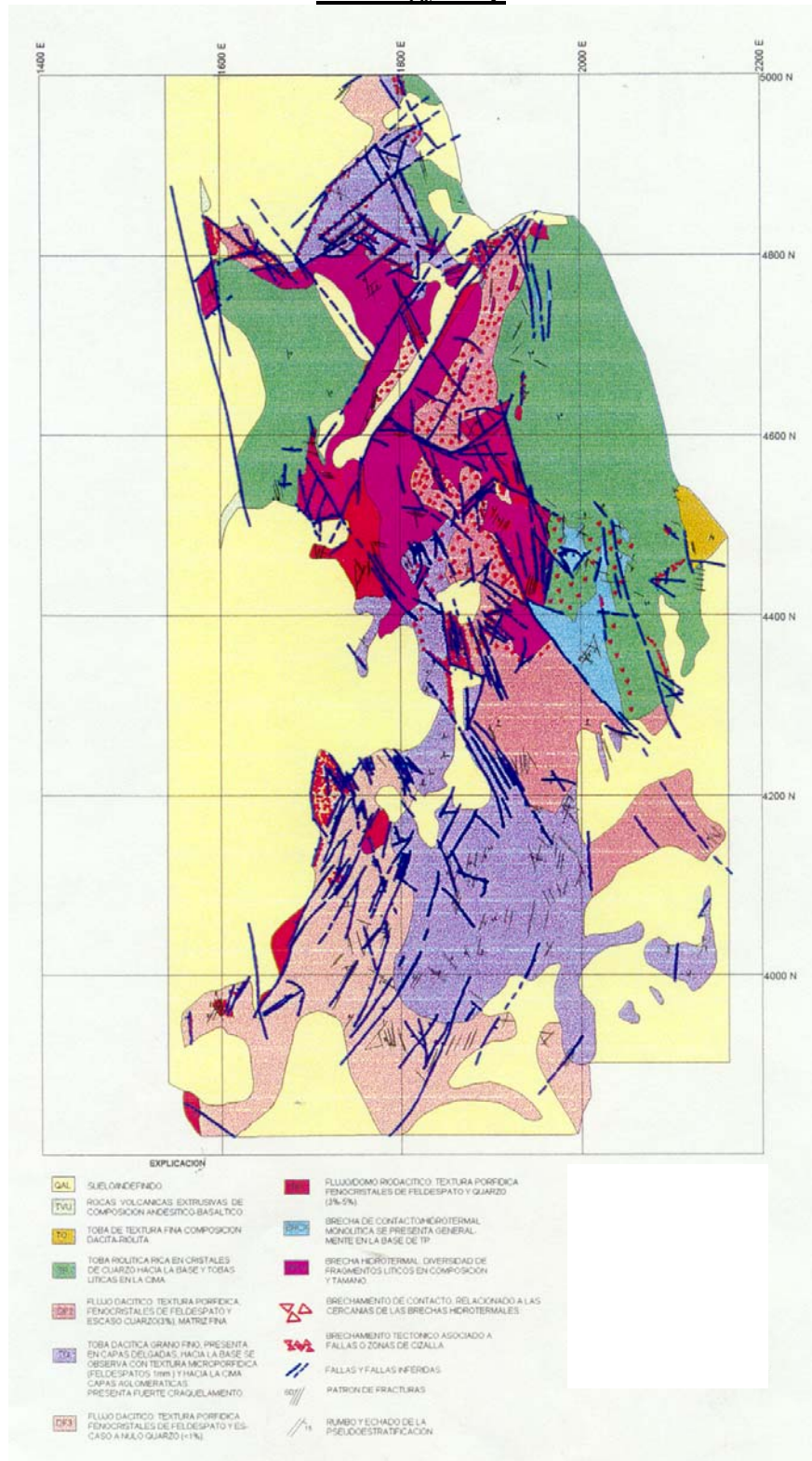# Regional Geology Map

Legend:

- **al** — Alluvium
- **vu** — Undifferentiated volcanics, andesites and basalts
- **di** — Andesite and dacite dikes
- **bx** — Silicified volcanic Breccias
- **tj** — Rhyolitic tuff
- **tk** — Rhyolitic crystal tuff
- **tl** — Eutaxitic rhyolitic tuff
- **lf** — Latite flows with quartz and biotite phenocrystals
- **dfu** — Undifferentiated dacites
- **df0** — Porphritic dacitic flows
- **tn** — Strongly welded rhyolitic tuff
- **to** — Welded rhyolitic tuff
- **dft** — Dacite
- **tp** — Lithic tuff and crystal tuff
- **df2** — Porphyritic dacite with < 1% quartz phenocrystals
- **tq** — Aphanitic rhyolitic tuff
- **df3** — Porphritic dacite without quartz phenocrystals
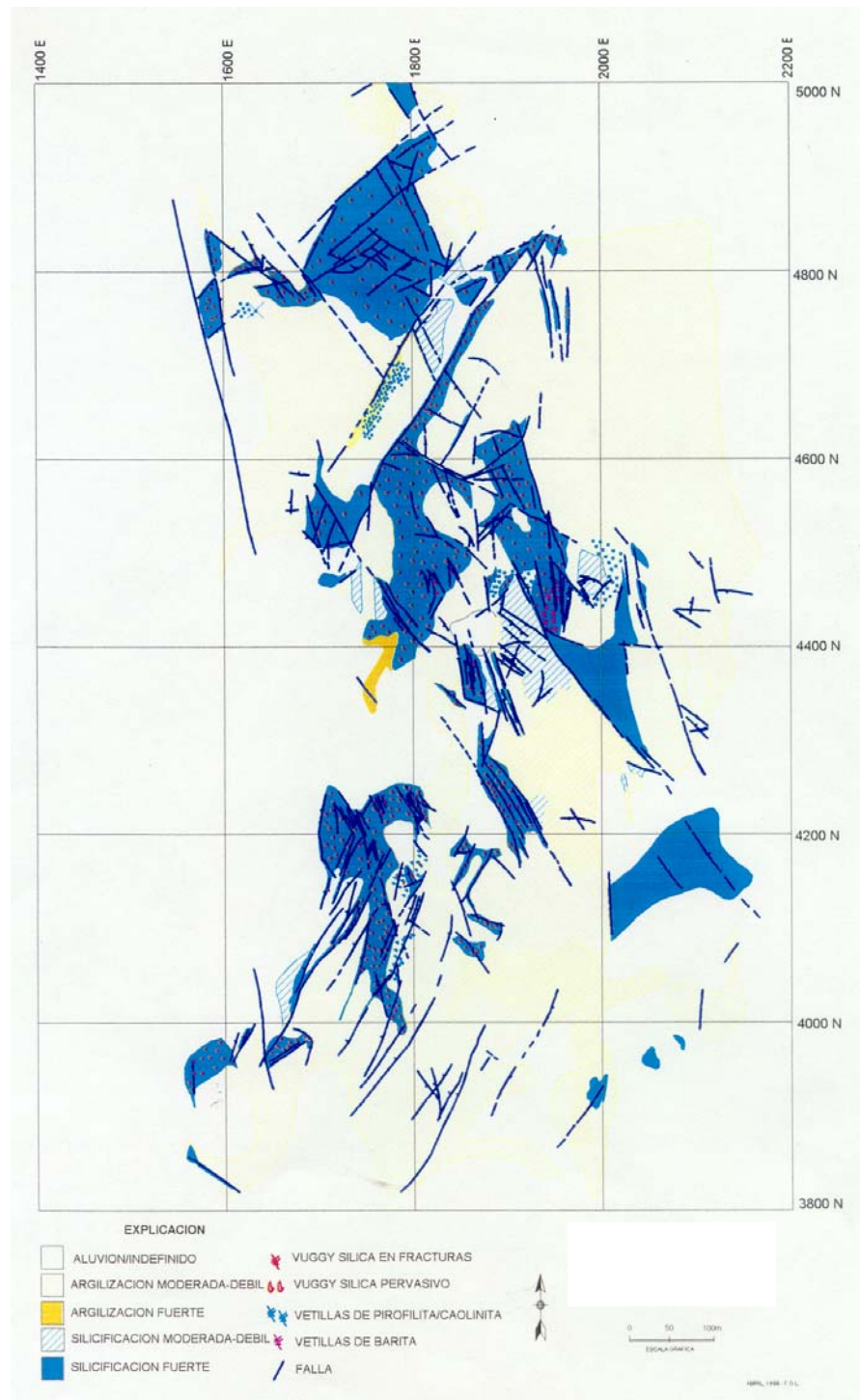- **rf** — Rhyodacitic flow dome with large quartz phenocryst
- Fault

Figure 9.2
Lithology Map

Figure 9.3
Alteration Map

Figure 9.4
Structural Interpretation
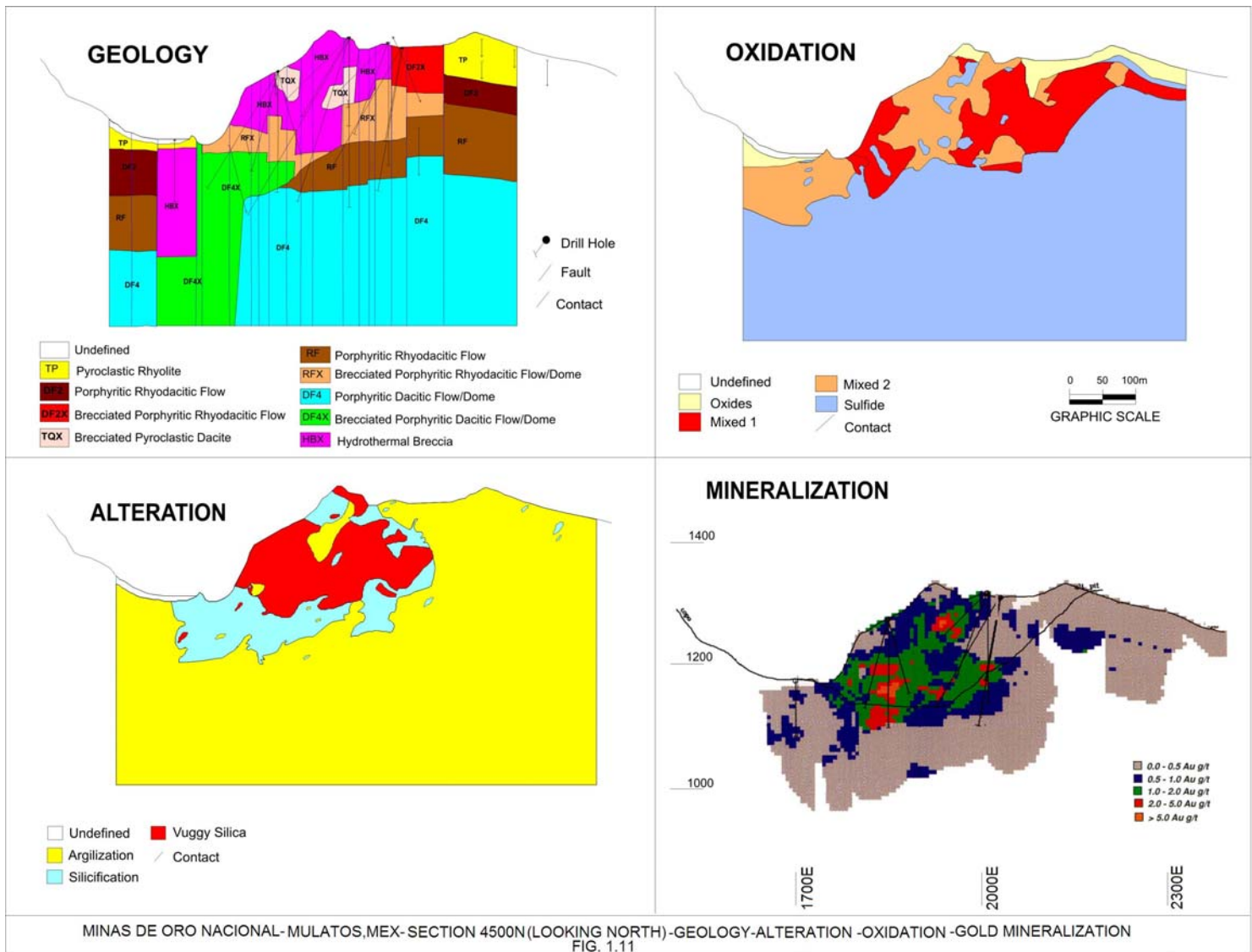

MULATOS PROJECT
STRUCTURAL INTERPRETATION
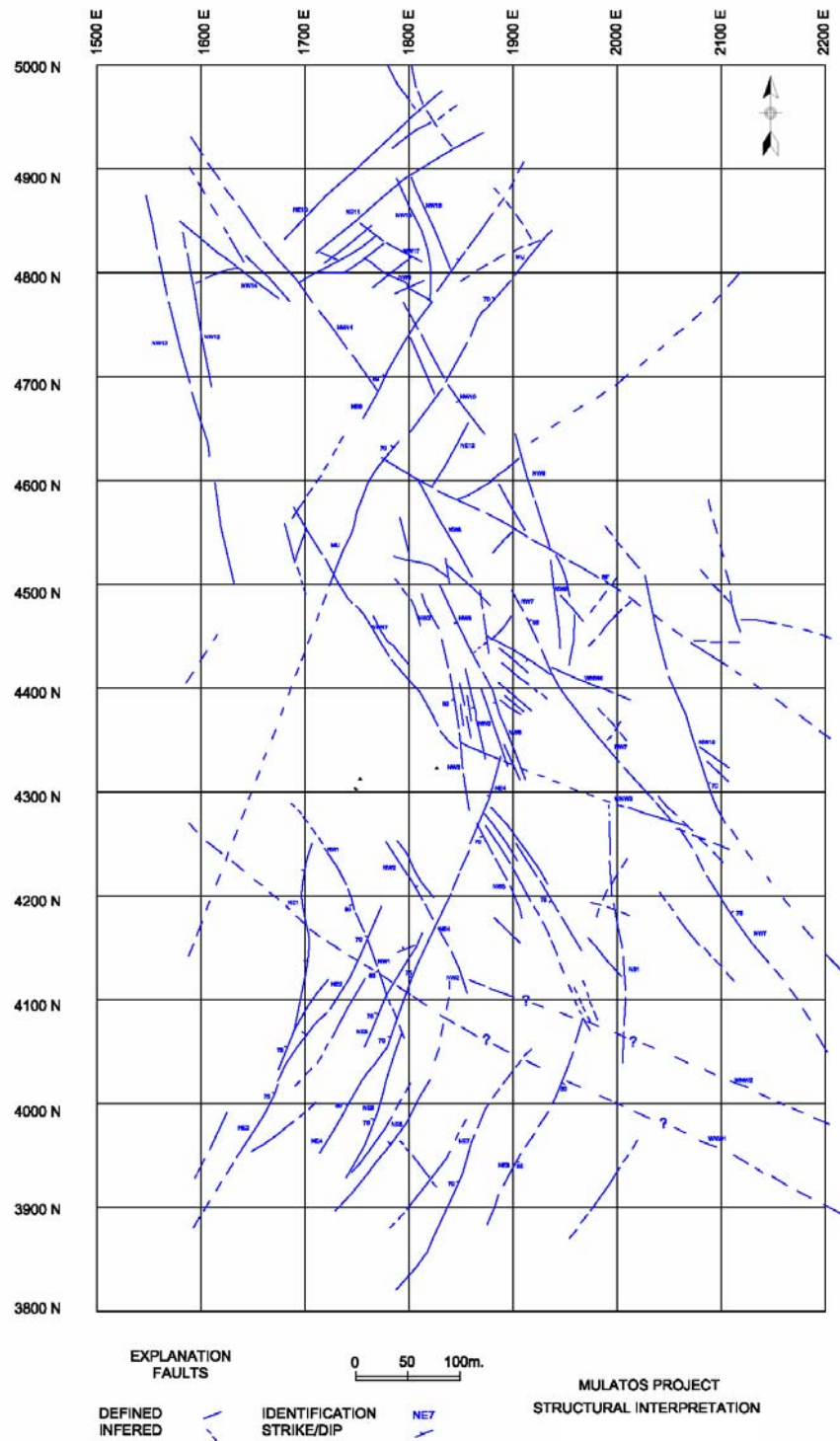
Figure 9.5
Section 4200 N
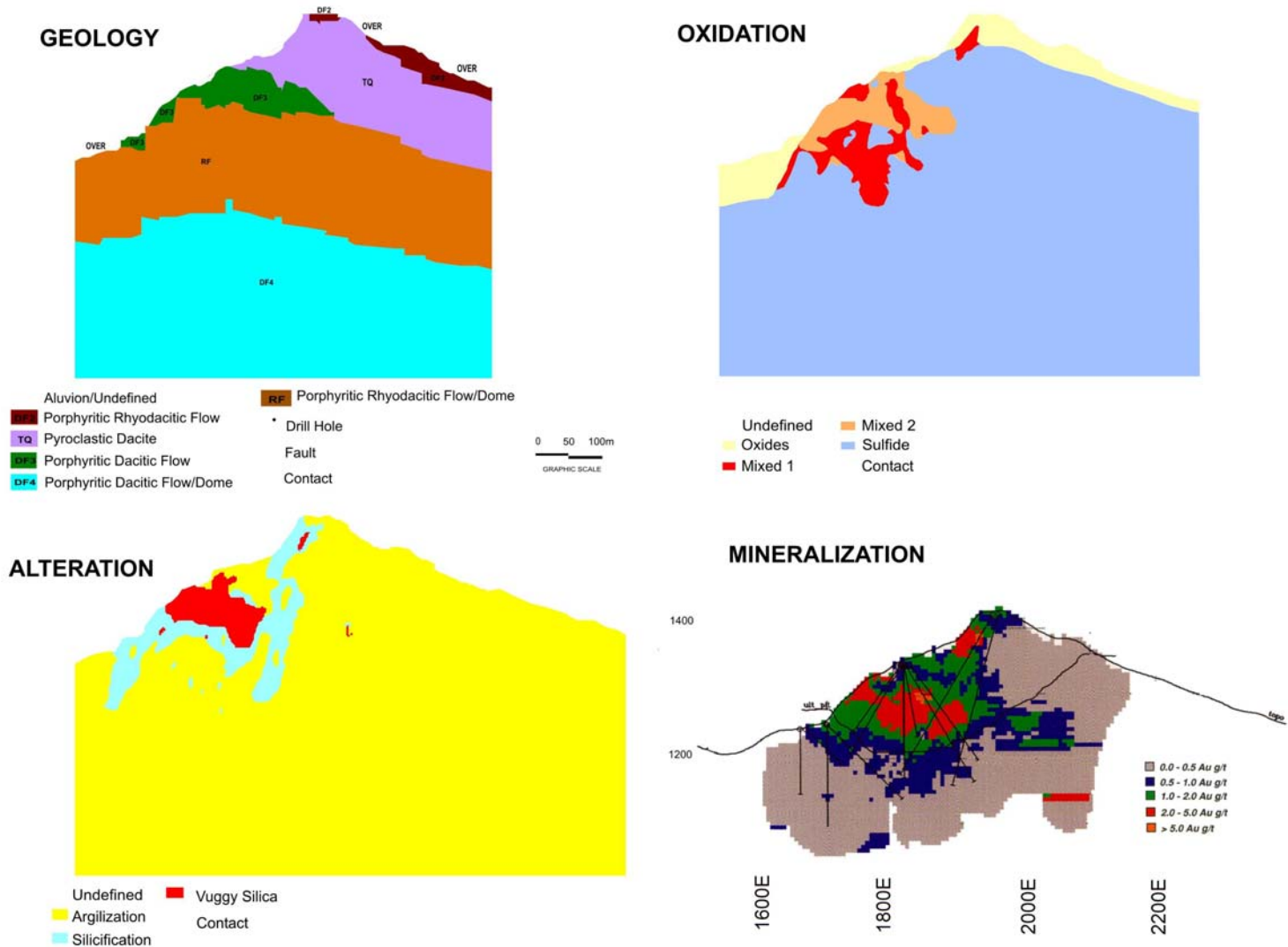
MINAS DE ORO NACIONAL-MULATOS, SECTION 4200N (LOOKING NORTH)-GEOLOGY-ALTERATION-OXIDATION-GOLD MINERALIZATION
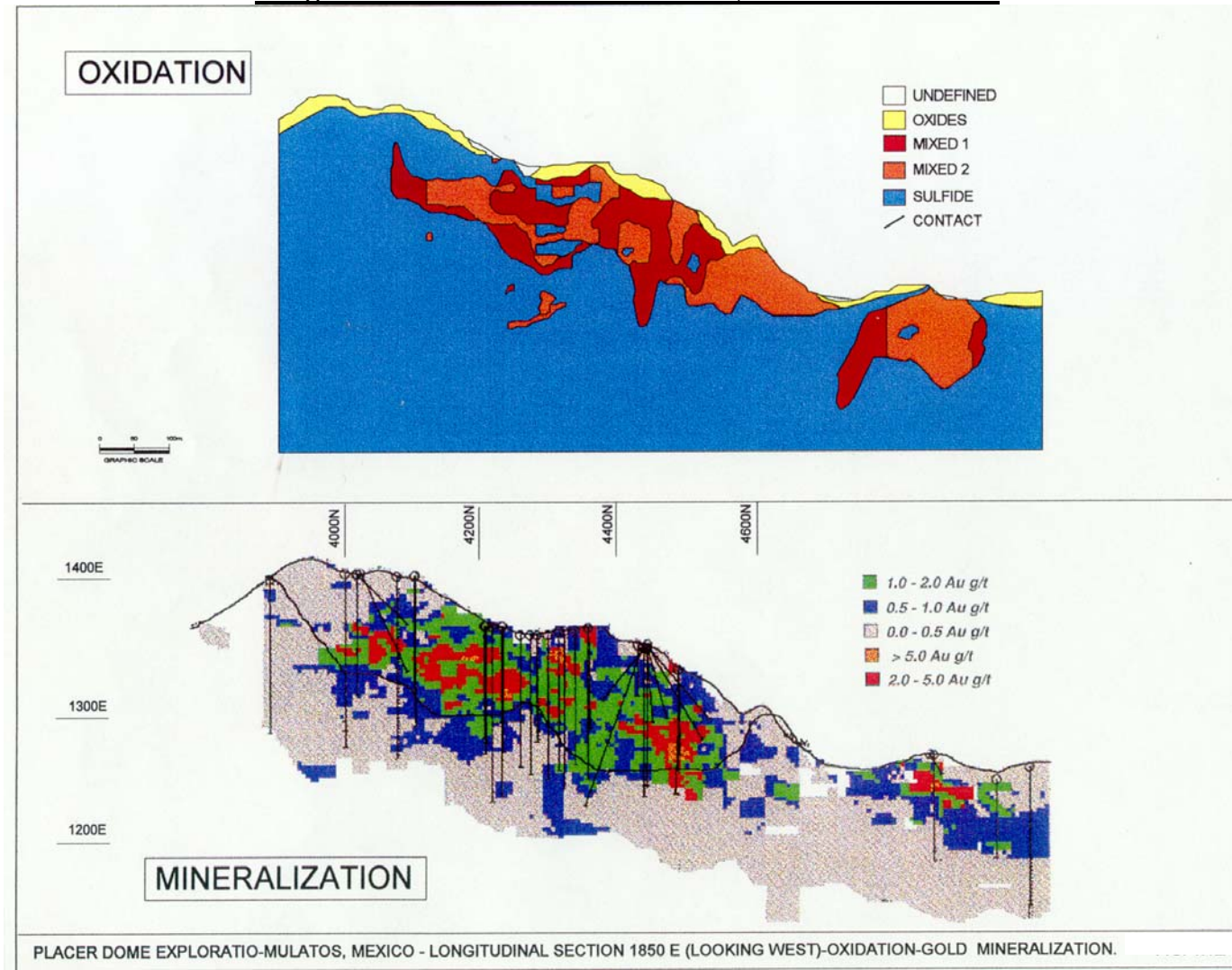
Figure 9.6
Section 4500 N

MINAS DE ORO NACIONAL - MULATOS, MEX - SECTION 4500N (LOOKING NORTH) - GEOLOGY - ALTERATION - OXIDATION - GOLD MINERALIZATION
FIG. 1.11

Figure 9.7
Longitudinal Sectin 1850 E. Geology, Alteration

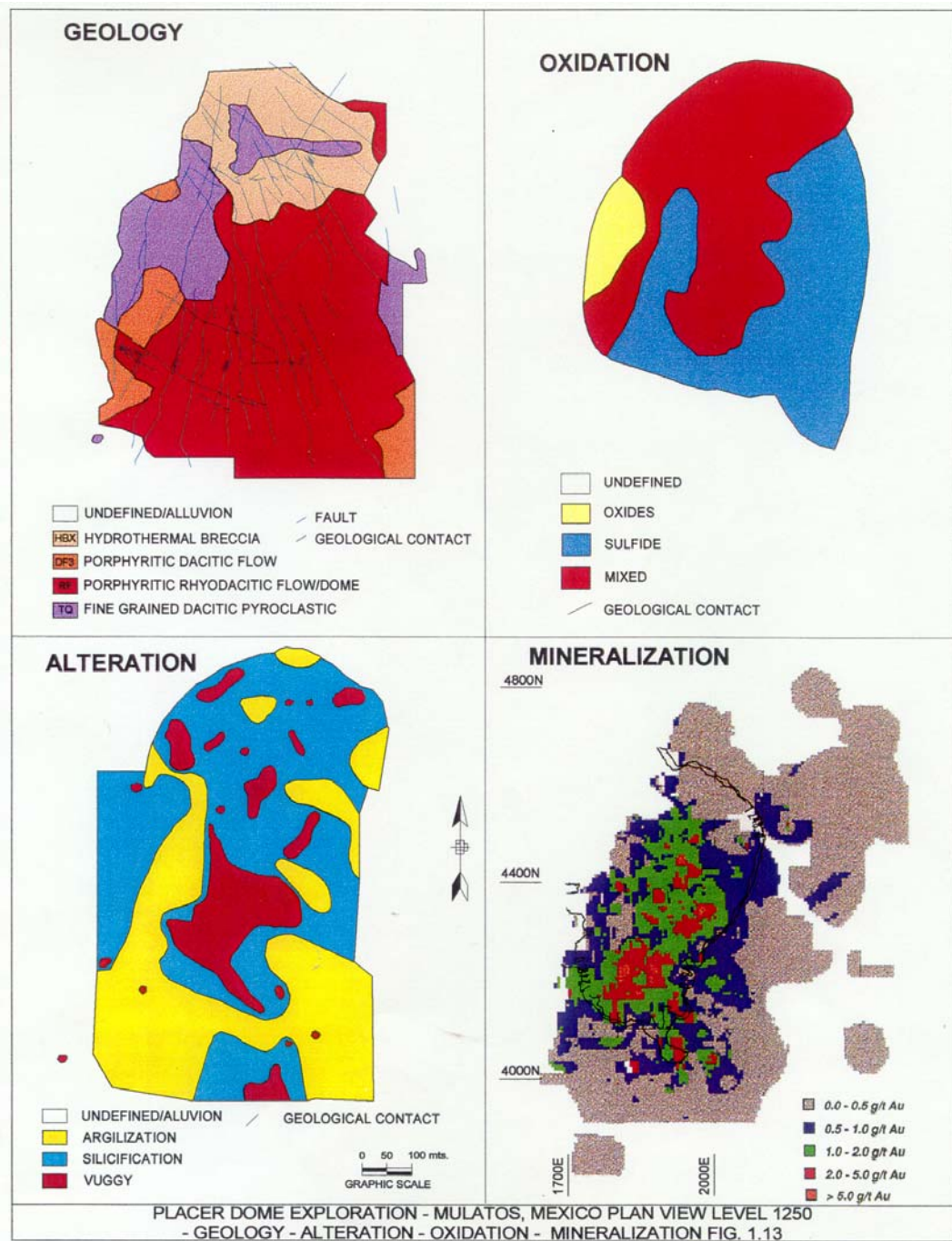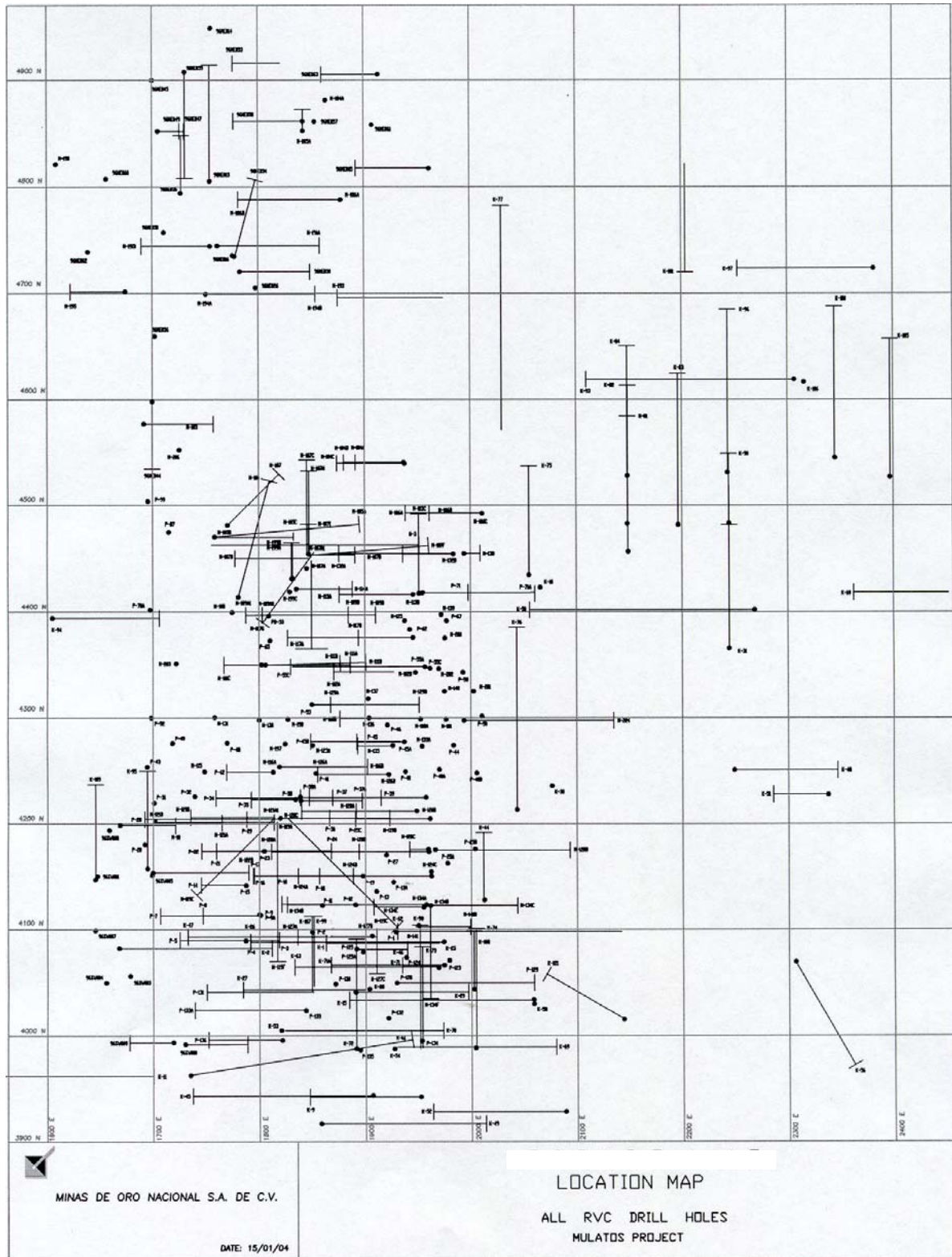MINAS DE ORO NACIONAL- MULATOS,MEX- SECTION 4500N (LOOKING NORTH) -GEOLOGY-ALTERATION -OXIDATION -GOLD MINERALIZATION
FIG. 1.11
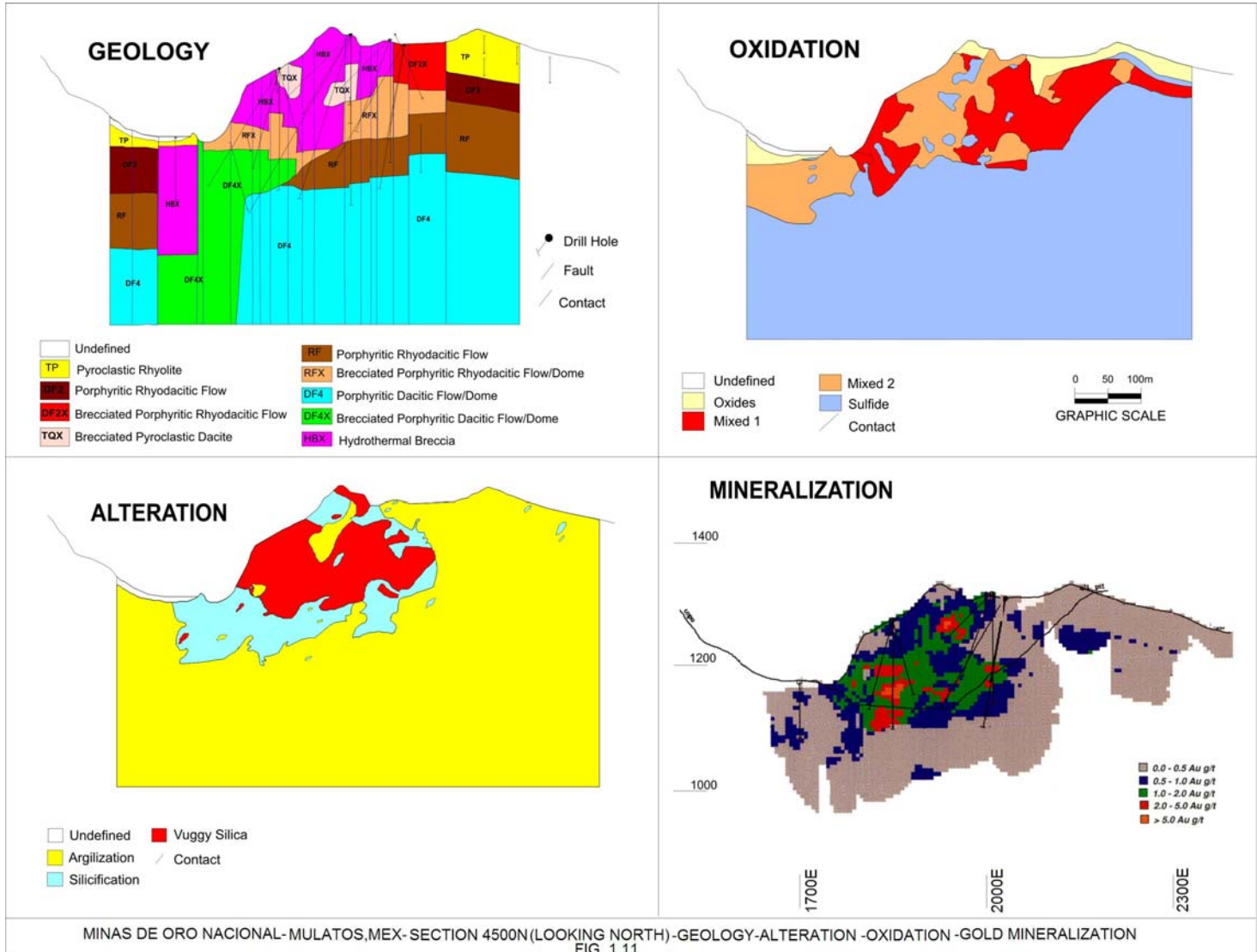
Figure 9.8
Longitudinal Section 1850 E Oxidation, Gold Mineralization

PLACER DOME EXPLORATIO-MULATOS, MEXICO - LONGITUDINAL SECTION 1850 E (LOOKING WEST)-OXIDATION-GOLD MINERALIZATION.

Figure 9.9
Plan View 1250

PLACER DOME EXPLORATION - MULATOS, MEXICO PLAN VIEW LEVEL 1250
- GEOLOGY - ALTERATION - OXIDATION - MINERALIZATION FIG. 1.13

Figure 13.1
All Reverse Circulation Drill Holes

LOCATION MAP

ALL RVC DRILL HOLES
MULATOS PROJECT

MINAS DE ORO NACIONAL S.A. DE C.V.

DATE: 15/01/04

Figure 13.2
Core Drill Locations

F I G U R E 1 . 1 6
LOCATION MAP

ALL CORE DRILL HOLES
MULATOS PROJECT

MINAS DE ORO NACIONAL S.A. DE C.V.

DATE: 15/01/04
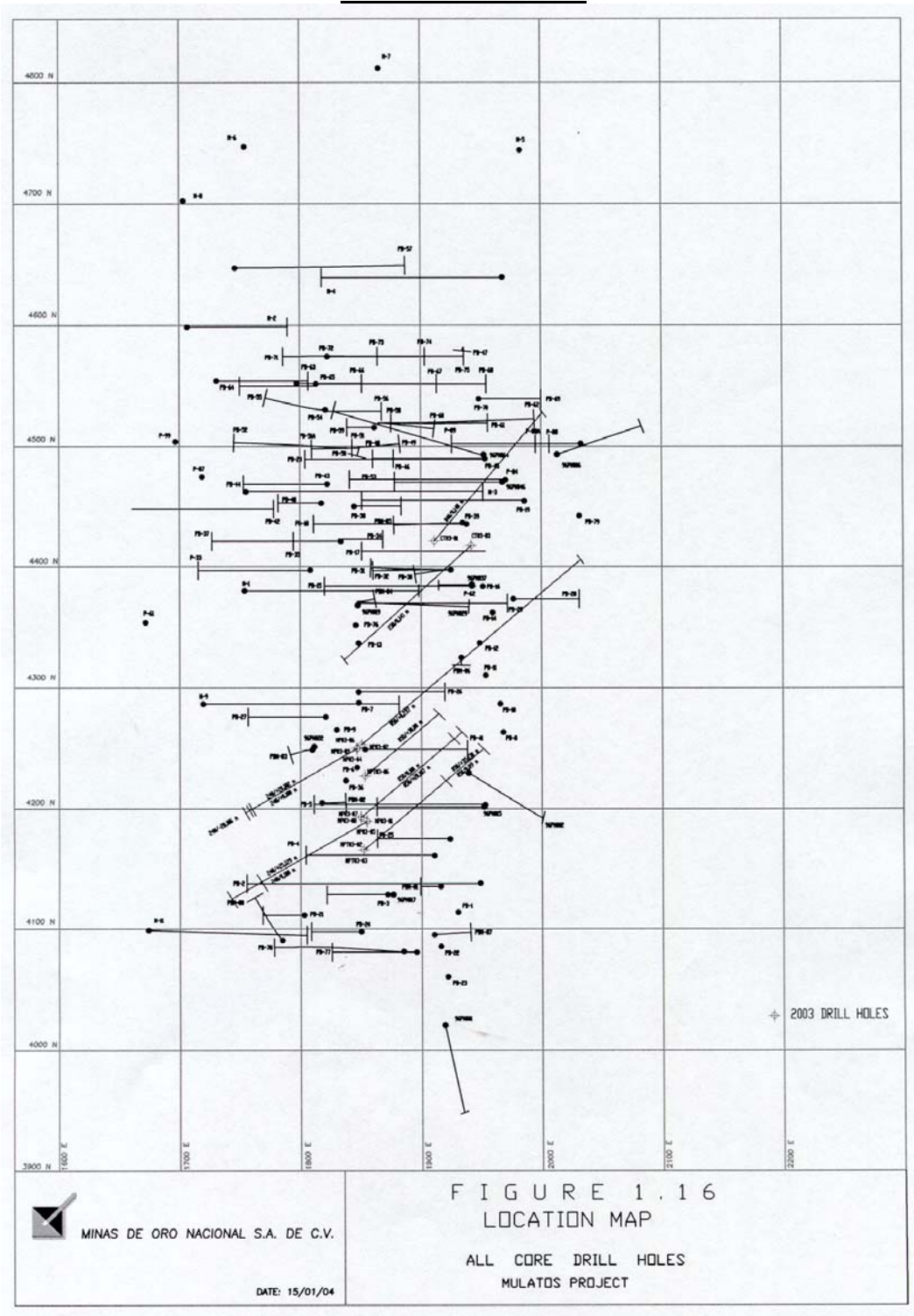
Figure 16.1
Comparison of Original MRA Assay to Placer Research Centre Assay
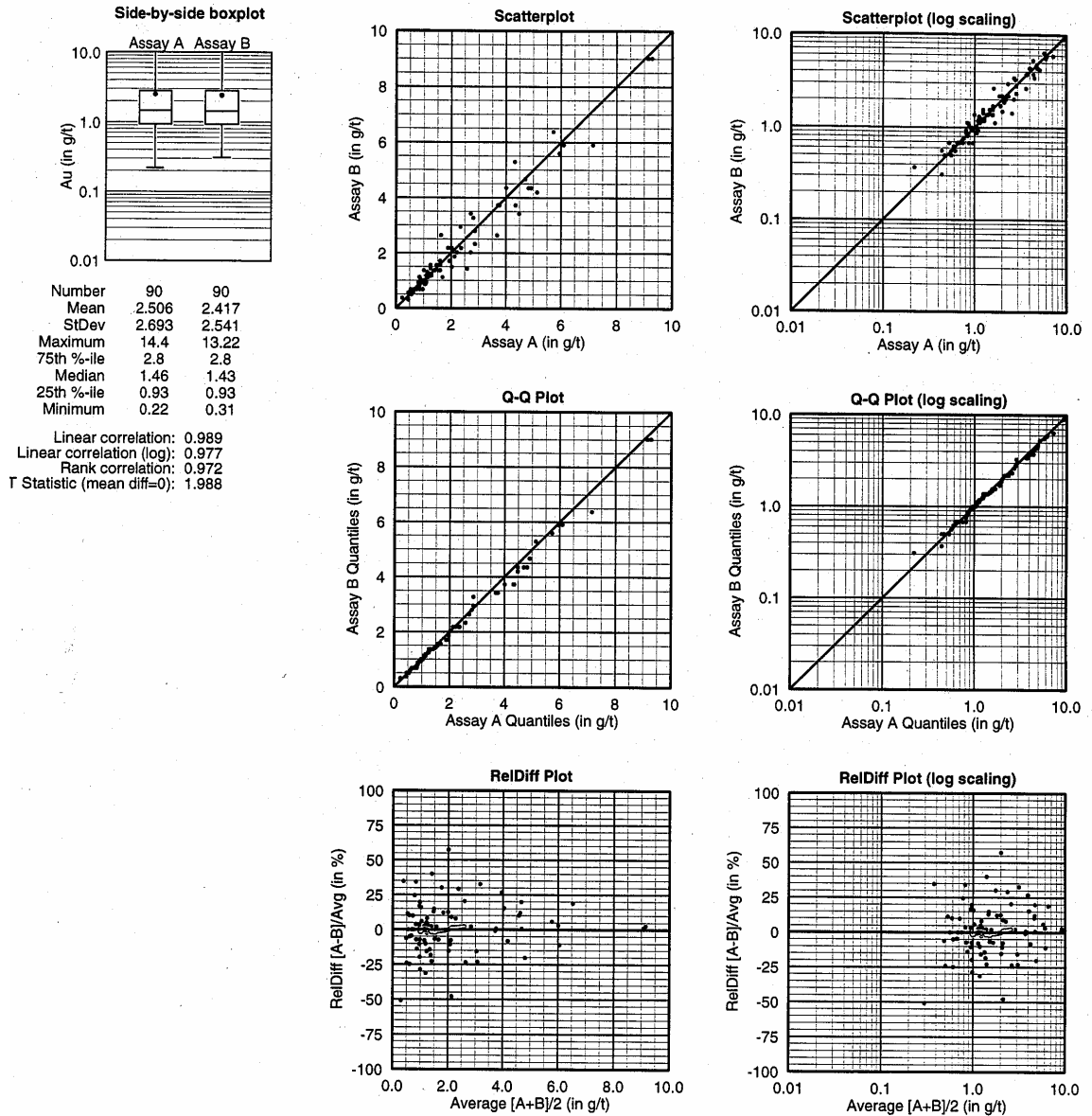
ASSAY A: Kennecott original
ASSAY B: Skyline check

	Assay A	Assay B
Number	90	90
Mean	2.506	2.417
StDev	2.693	2.541
Maximum	14.4	13.22
75th %-ile	2.8	2.8
Median	1.46	1.43
25th %-ile	0.93	0.93
Minimum	0.22	0.31

Linear correlation: 0.989
Linear correlation (log): 0.977
Rank correlation: 0.972
T Statistic (mean diff=0): 1.988

Figure 1.21: Comparison of original Kennecott assay (Rocky Mt.) to Skyline check assay: 1993

Figure 16.2
Comparison of Original Kennecott Assay to Skyline Check Assay

ASSAY A: Kennecott original
ASSAY B: Skyline check
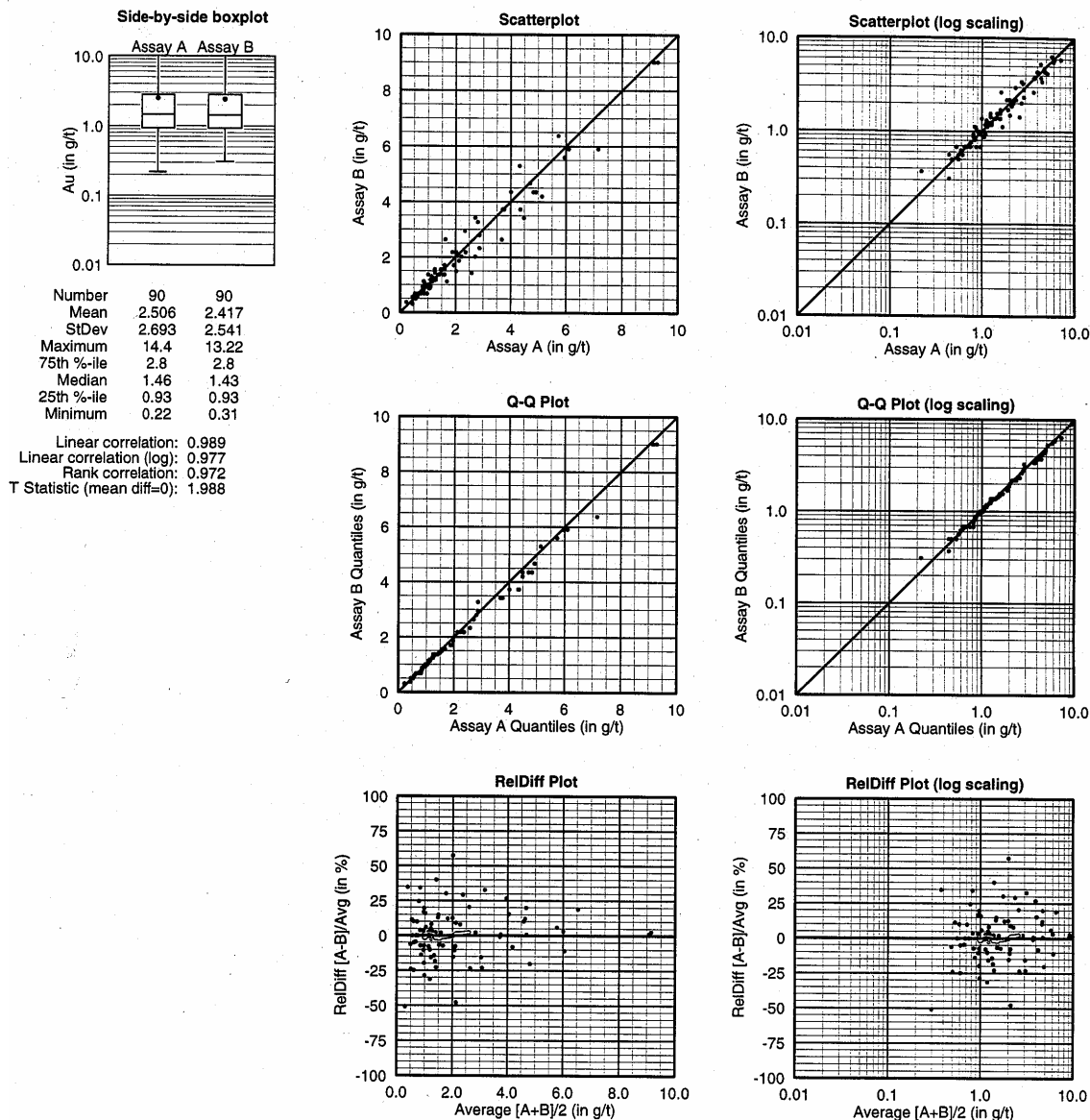
Number	90	90
Mean	2.506	2.417
StDev	2.693	2.541
Maximum	14.4	13.22
75th %-ile	2.8	2.8
Median	1.46	1.43
25th %-ile	0.93	0.93
Minimum	0.22	0.31

Linear correlation: 0.989
Linear correlation (log): 0.977
Rank correlation: 0.972
T Statistic (mean diff=0): 1.988

Figure 1.21: Comparison of original Kennecott assay (Rocky Mt.) to Skyline check assay: 1993

Figure 16.3
Comparison of Original Kennecott Assay to Duplicate Sample Assay

ASSAY A: Orignal Assay
ASSAY B: Duplicate Sample

	Assay A	Assay B
Number	401	401
Mean	0.802	0.817
StDev	2.231	2.437
Maximum	29.86	32.48
75th %-ile	0.62	0.62
Median	0.25	0.26
25th %-ile	0.06	0.1
Minimum	0.03	0.01

Linear correlation: 0.922
Linear correlation (log): 0.885
Rank correlation: 0.882
Statistic (mean diff=0): -0.324
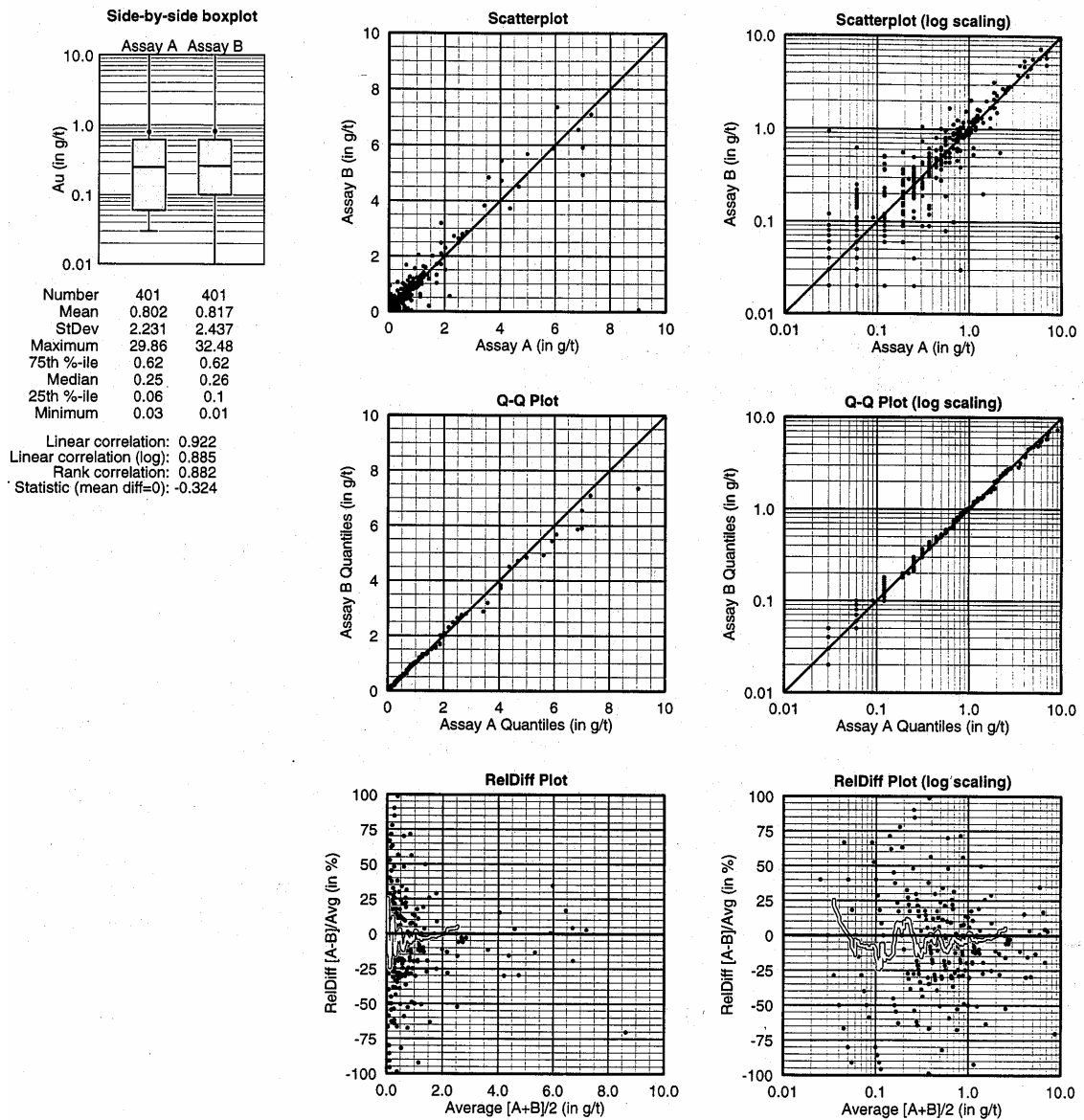
Figure 1.22: Comparison of orignal Kennecott as say to duplicate sample assay (Rocky Mt: 1993)

Figure 16.4
Comparison of Original Barringer Assays to Placer Check Assays

ASSAY A: Barringer original
ASSAY B: Placer check

	Assay A	Assay B
Number	213	213
Mean	1.419	1.45
StDev	4.934	4.865
Maximum	65.93	63.0
75th %-ile	1.16	1.21
Median	0.51	0.52
25th %-ile	0.16	0.16
Minimum	0.01	0.01

Linear correlation: 0.995
Linear correlation (log): 0.987
Rank correlation: 0.980
Statistic (mean diff=0): -0.879
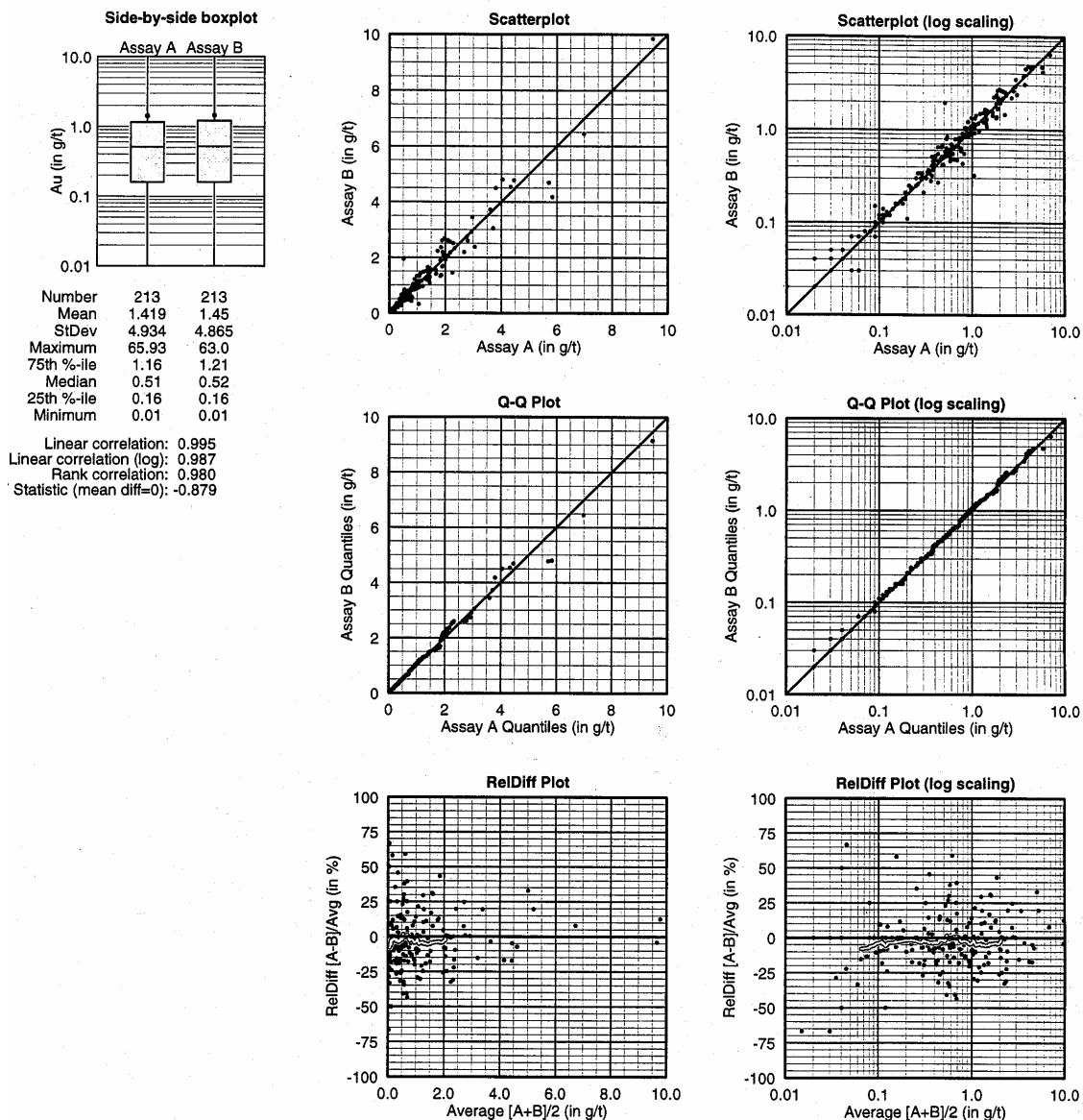
Figure 1.23: Comparison of original Barringer assays to Placer check assays: 1996

Figure 19.1
Resource Area and Pit Reserve Location

See Illustration 1.1

Figure 19.2
Sample Location Map
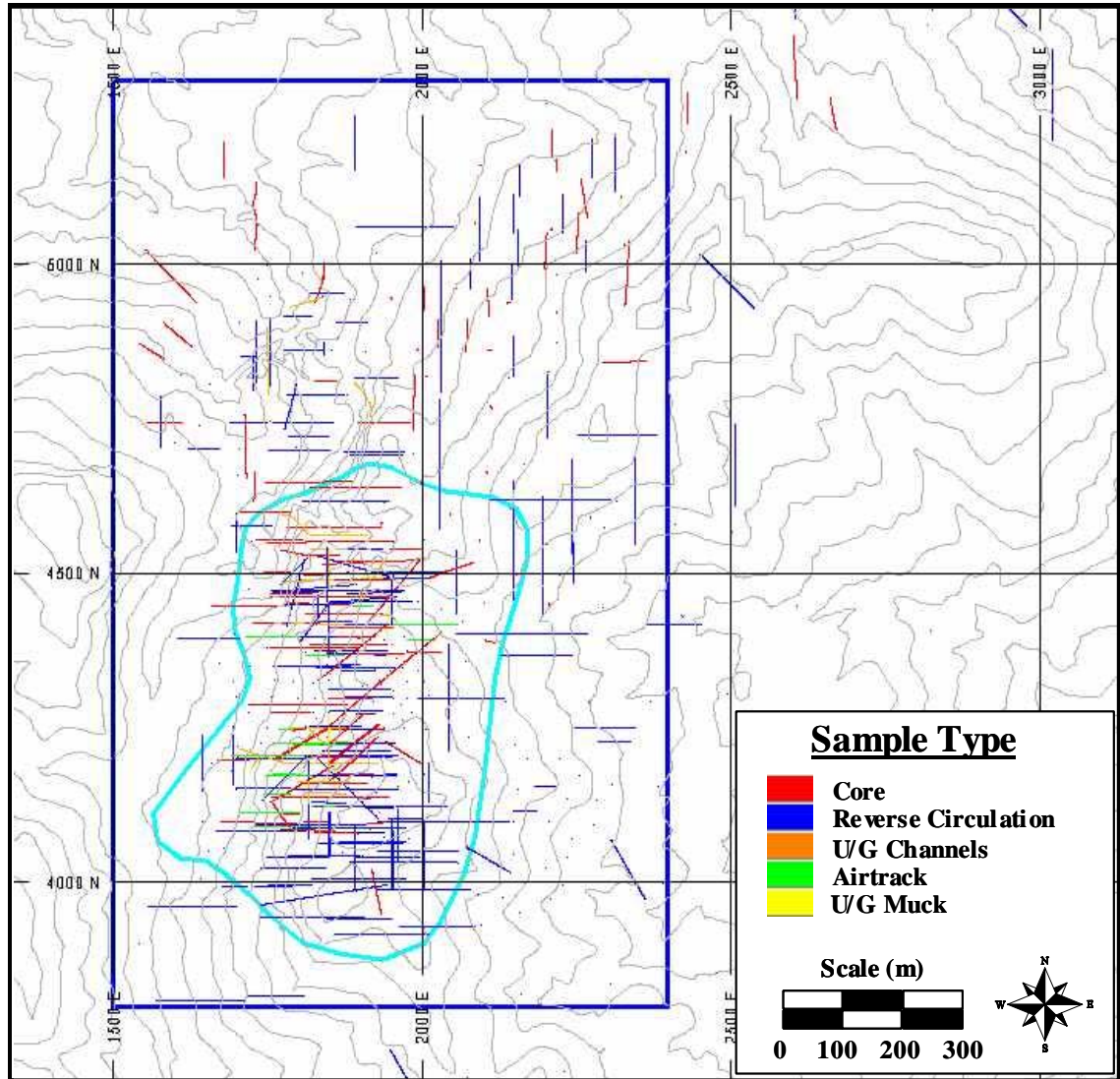
Figure 19.3
MRA vs. PDI Research Center

ASSAY A: MRA
ASSAY B: Placer Research

Side-by-side boxplot

Number	263	263
Mean	2.065	1.943
StDev	3.99	3.79
Maximum	39.0	35.4
75th %-ile	1.8	1.69
Median	0.900	0.830
25th %-ile	0.41	0.35
Minimum	0.06	0.01

Linear correlation: 0.968
Linear correlation (log): 0.944
Rank correlation: 0.961
T Statistic (mean diff=0): 1.974
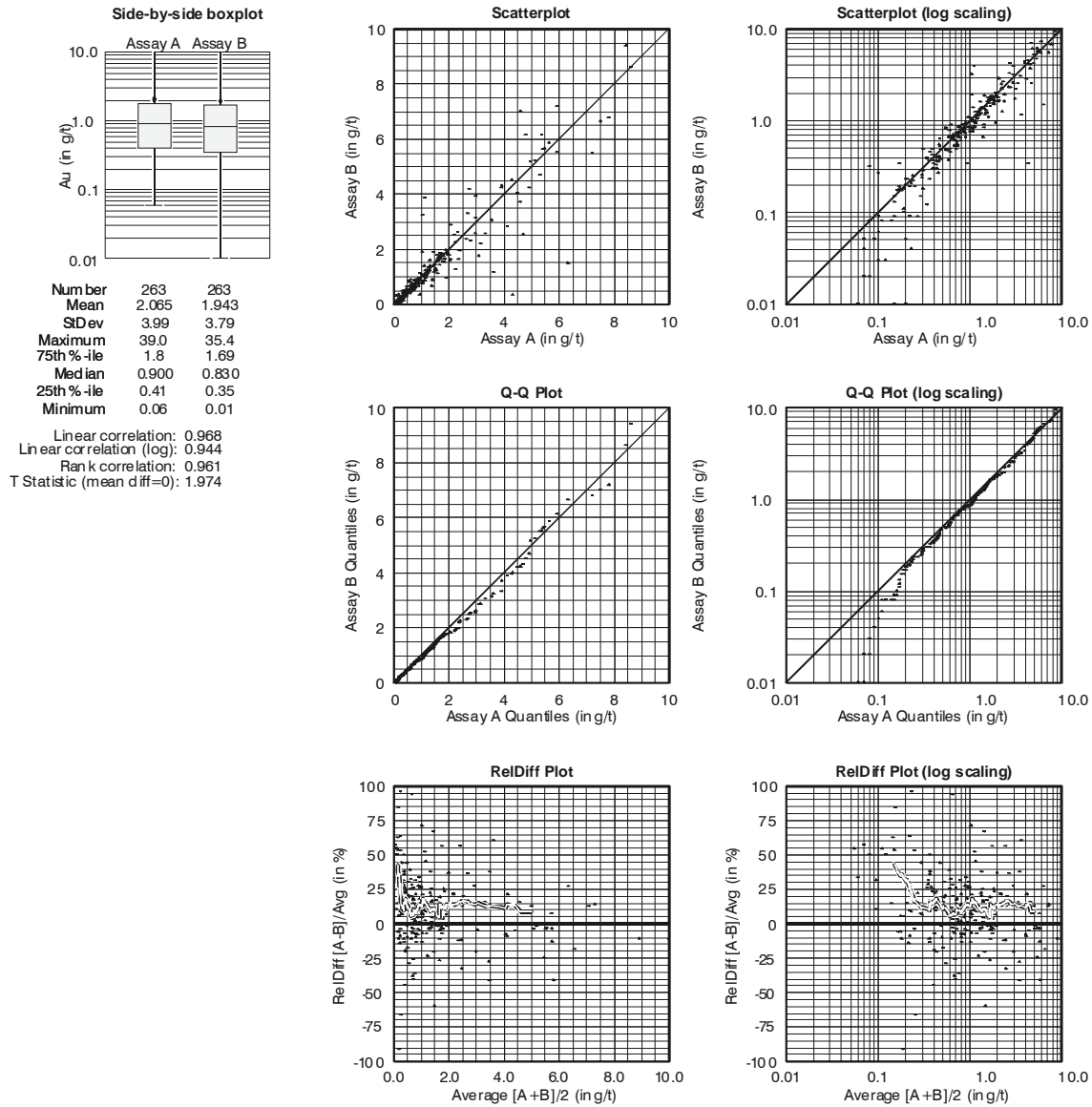
Figure 19.6
Kennecott Original vs. Skyline

ASSAY A: Kennecott original
ASSAY B: Skyline check

Side-by-side boxplot

	Assay A	Assay B
Number	90	90
Mean	2.506	2.417
StDev	2.693	2.541
Maximum	14.4	13.22
75th %-ile	2.8	2.8
Median	1.46	1.43
25th %-ile	0.93	0.93
Minimum	0.22	0.31

Linear correlation: 0.989
Linear correlation (log): 0.977
Rank correlation: 0.972
T Statistic (mean diff=0): 1.988
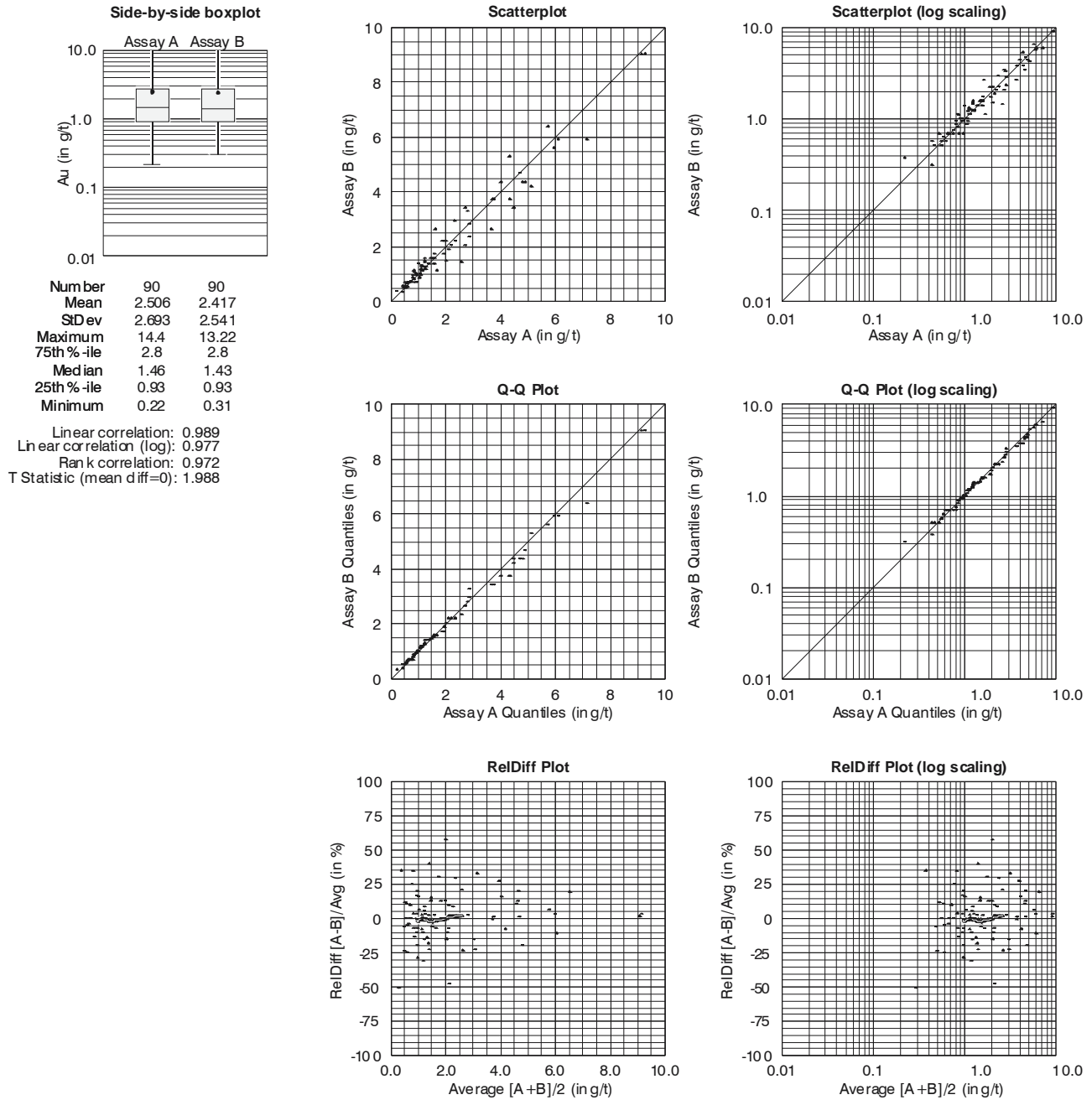
Figure 19.7
Kennecott Duplicate Pulp Assays

ASSAY A: Orignal Assay
ASSAY B: Duplicate Sample

Side-by-side boxplot

Number	401	401
Mean	0.802	0.817
StDev	2.231	2.437
Maximum	29.86	32.48
75th %-ile	0.62	0.62
Median	0.25	0.26
25th %-ile	0.06	0.1
Minimum	0.03	0.01

Linear correlation: 0.922
Linear correlation (log): 0.885
Rank correlation: 0.882
T Statistic (mean diff=0): -0.324
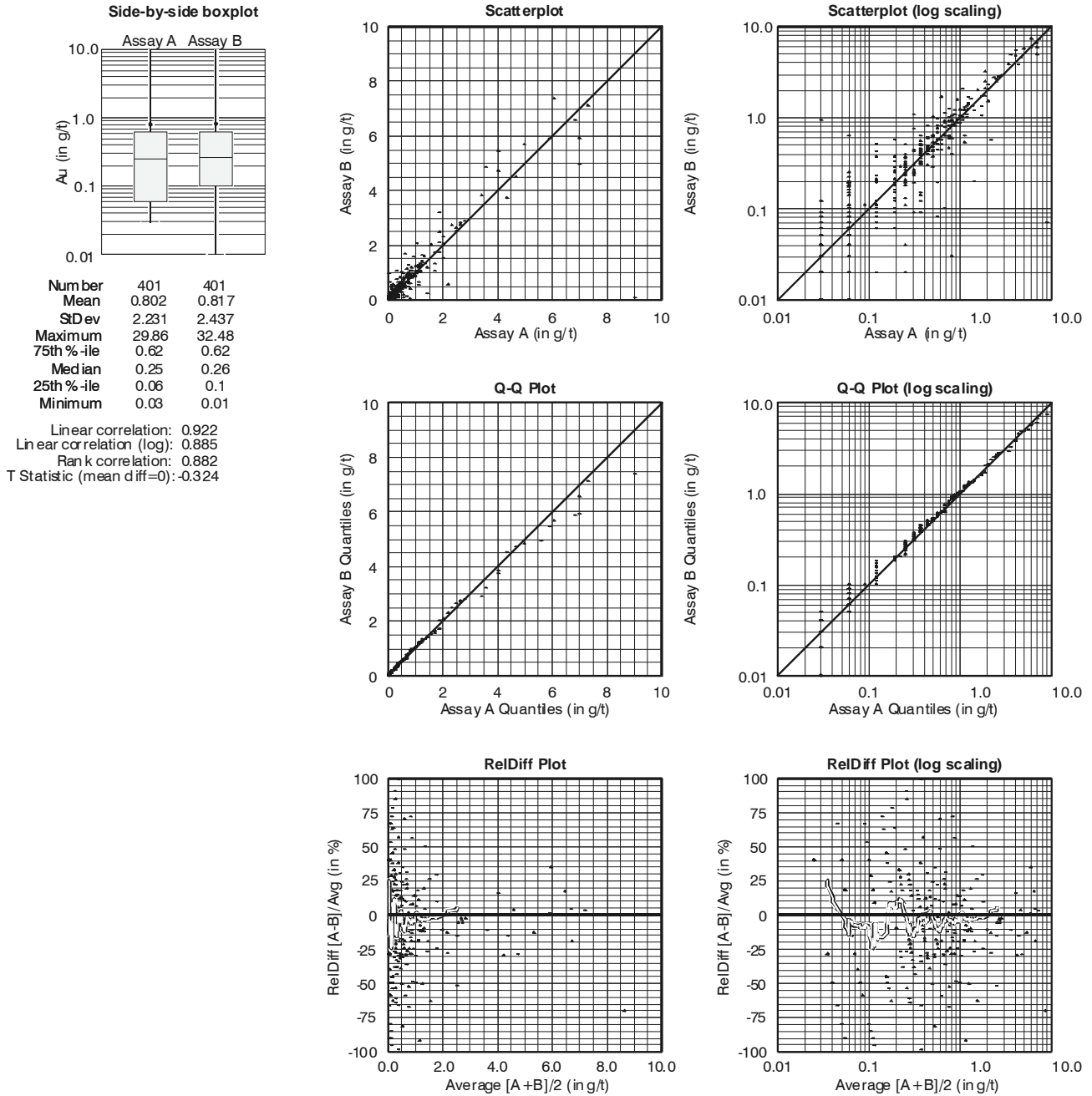
Figure 19.8
1996 PDI Check Assays

ASSAY A: Barringer original
ASSAY B: Placer check

Side-by-side boxplot

	Assay A	Assay B
Number	213	213
Mean	1.419	1.45
StDev	4.934	4.865
Maximum	65.93	63.0
75th %-ile	1.16	1.21
Median	0.510	0.520
25th %-ile	0.16	0.16
Minimum	0.01	0.01

Linear correlation: 0.995
Linear correlation (log): 0.987
Rank correlation: 0.980
T Statistic (mean diff=0): -0.879

Scatterplot

Scatterplot (log scaling)

Q-Q Plot

Q-Q Plot (log scaling)

RelDiff Plot

RelDiff Plot (log scaling)
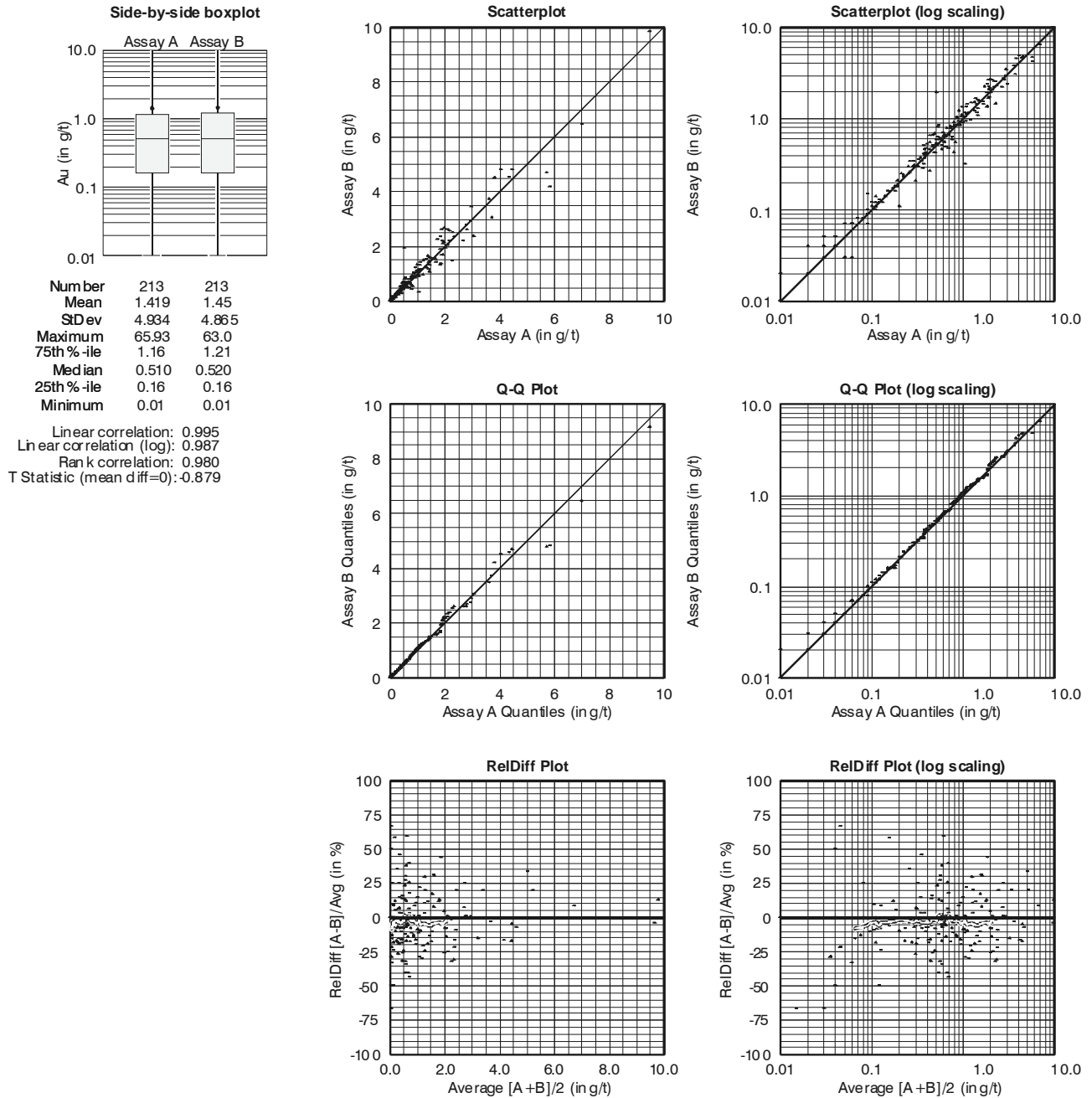
Figure 19.9
Raw Gold Assay Histogram
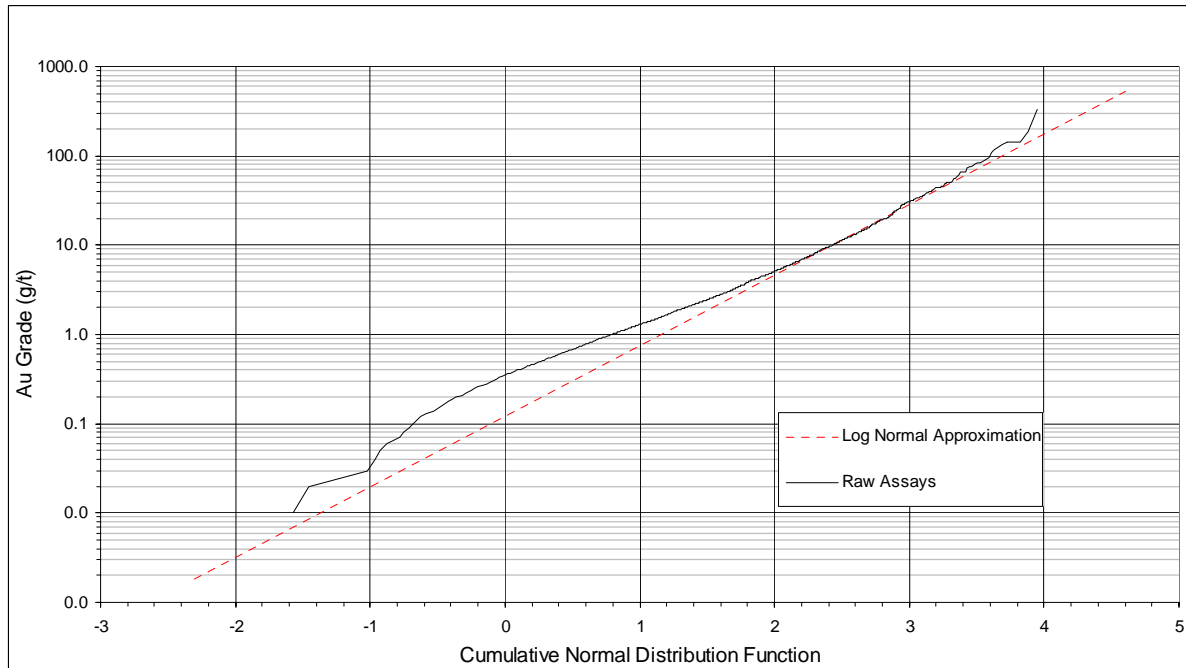
Figure 19.10
Gold Dilution
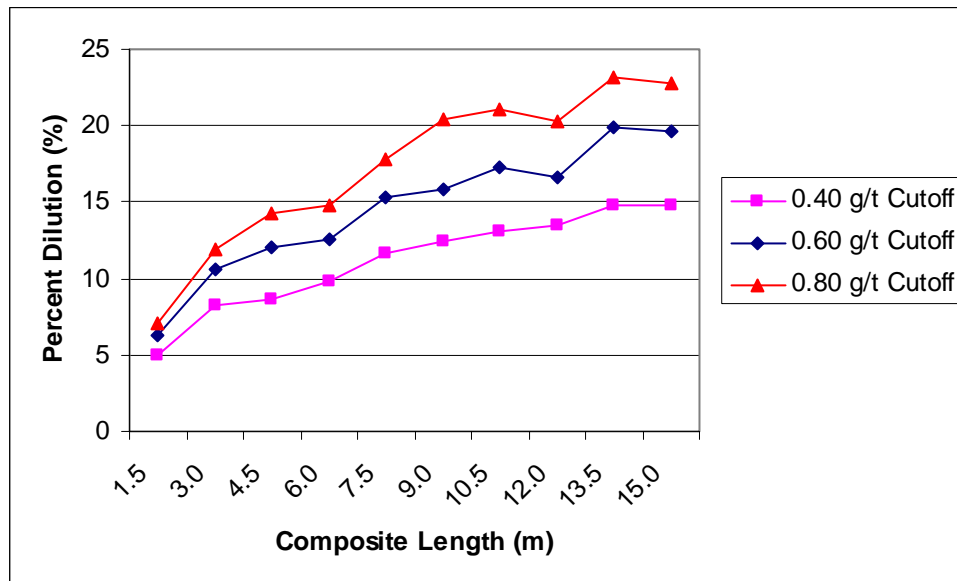
Figure 19.11
Gold Ore Loss
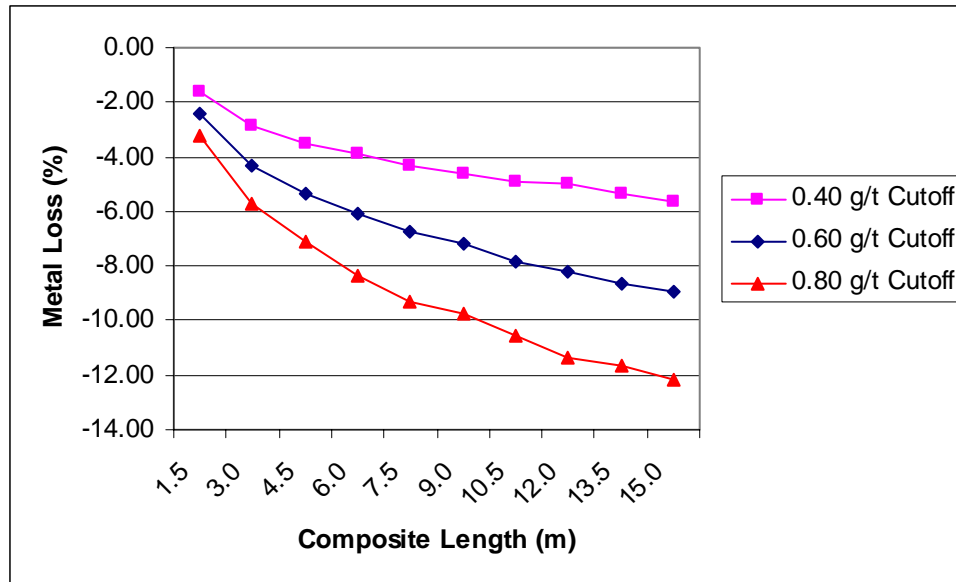
Figure 19.12
Gold Correlogram

Gold Grade Correlogram
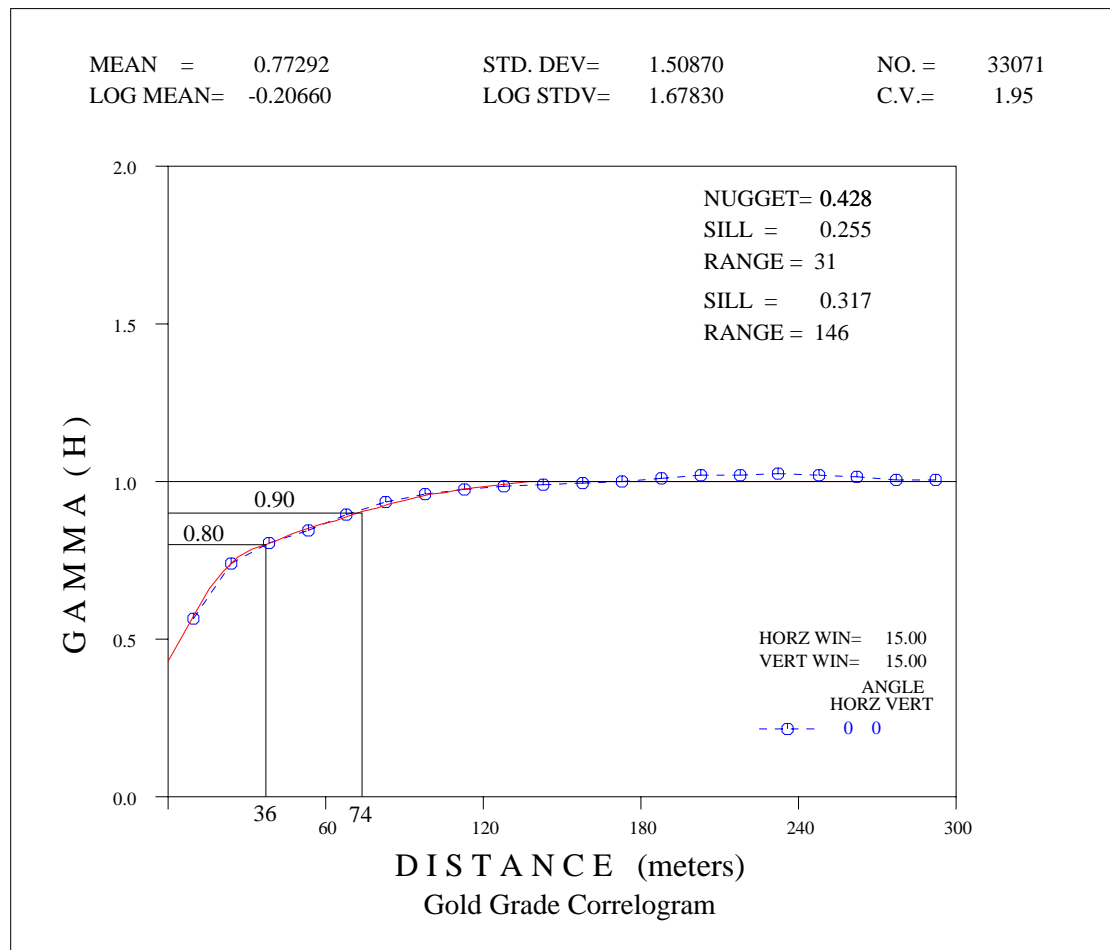
Figure 19.13
Gold and Silver Search Ellipses
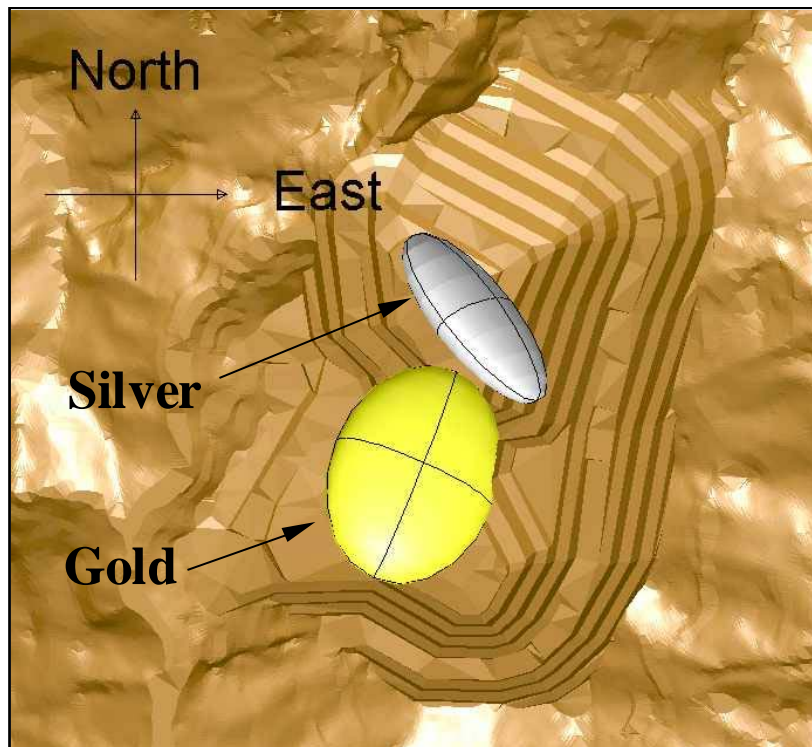
Plan View
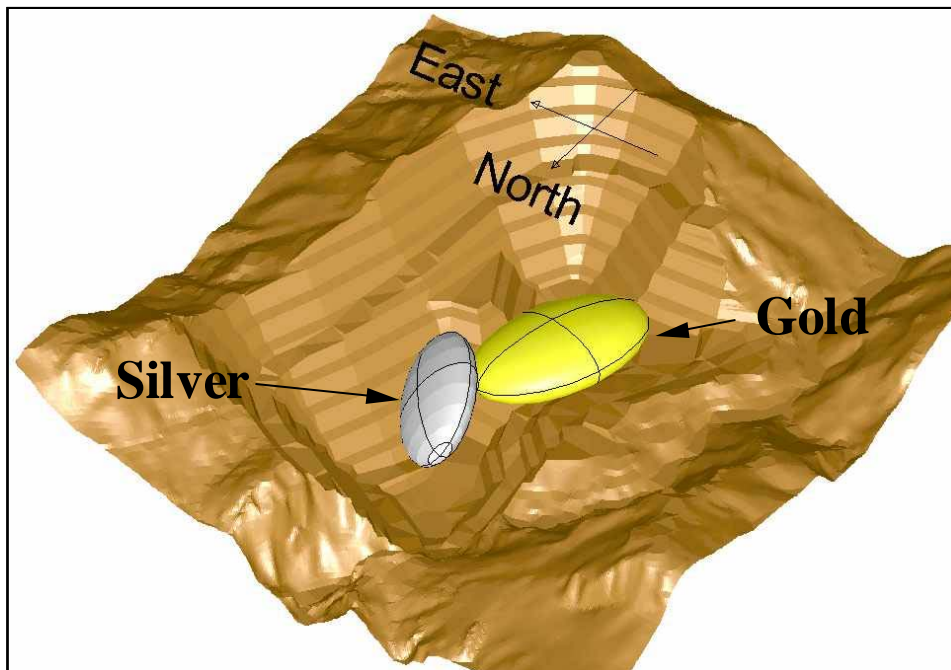
Perspective View

Figure 19.14
Cross Section Locations
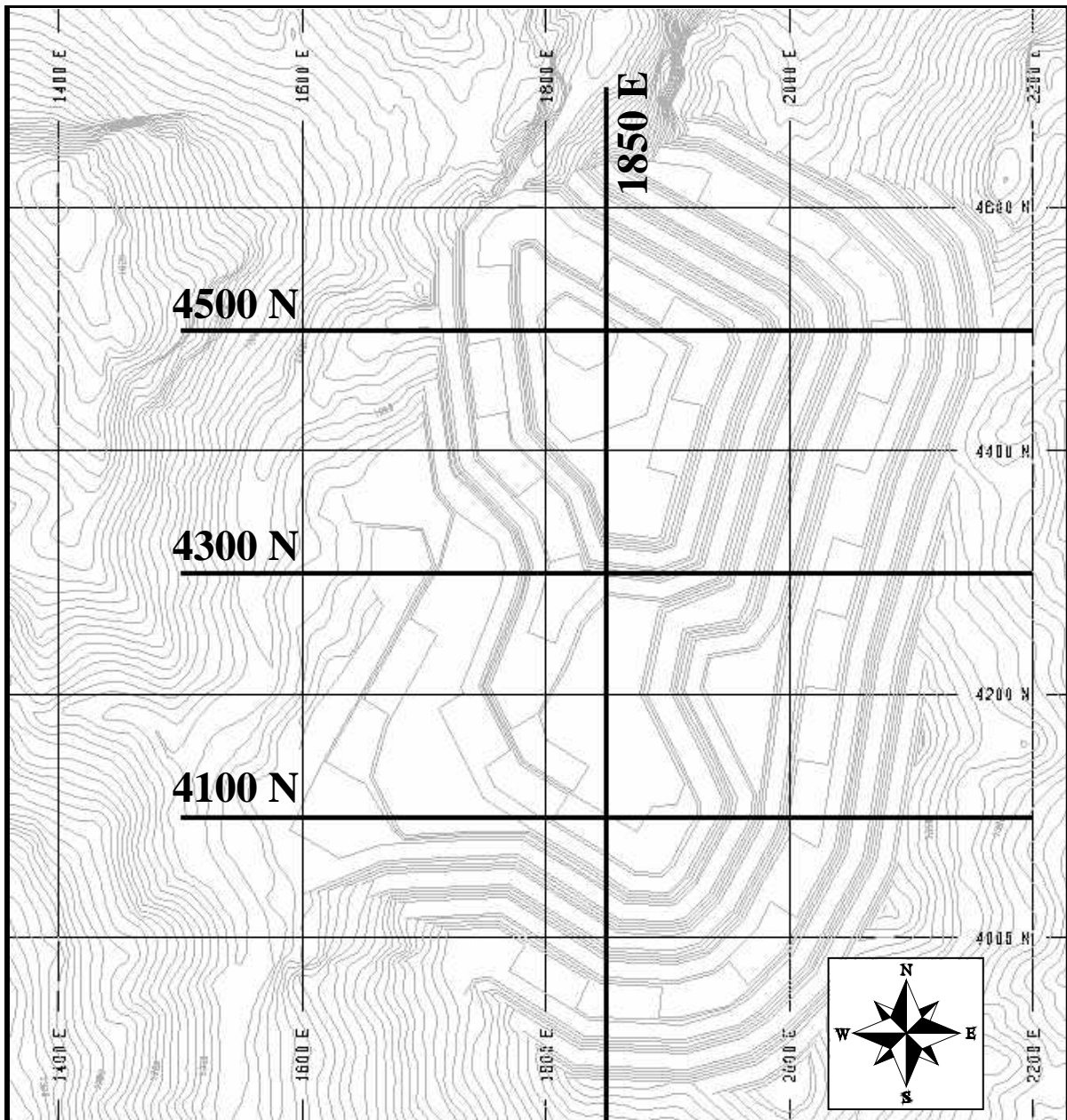
Figure 19.15
Block Model Cross Section 4100 North
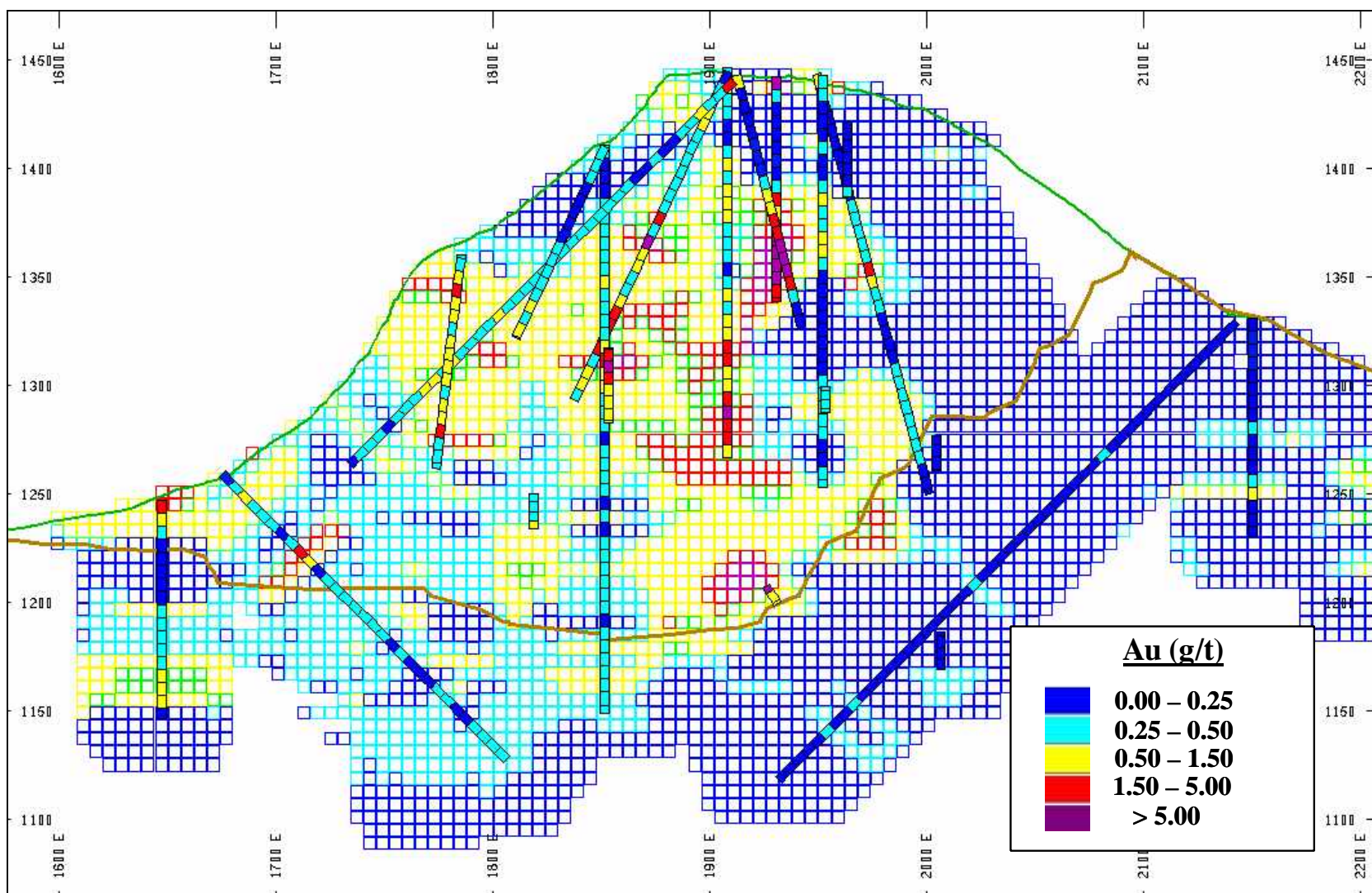
Figure 19.16
Block Model Cross Section 4300 North

Au (g/t)

- 0.00 – 0.25
- 0.25 – 0.50
- 0.50 – 1.50
- 1.50 – 5.00
- > 5.00
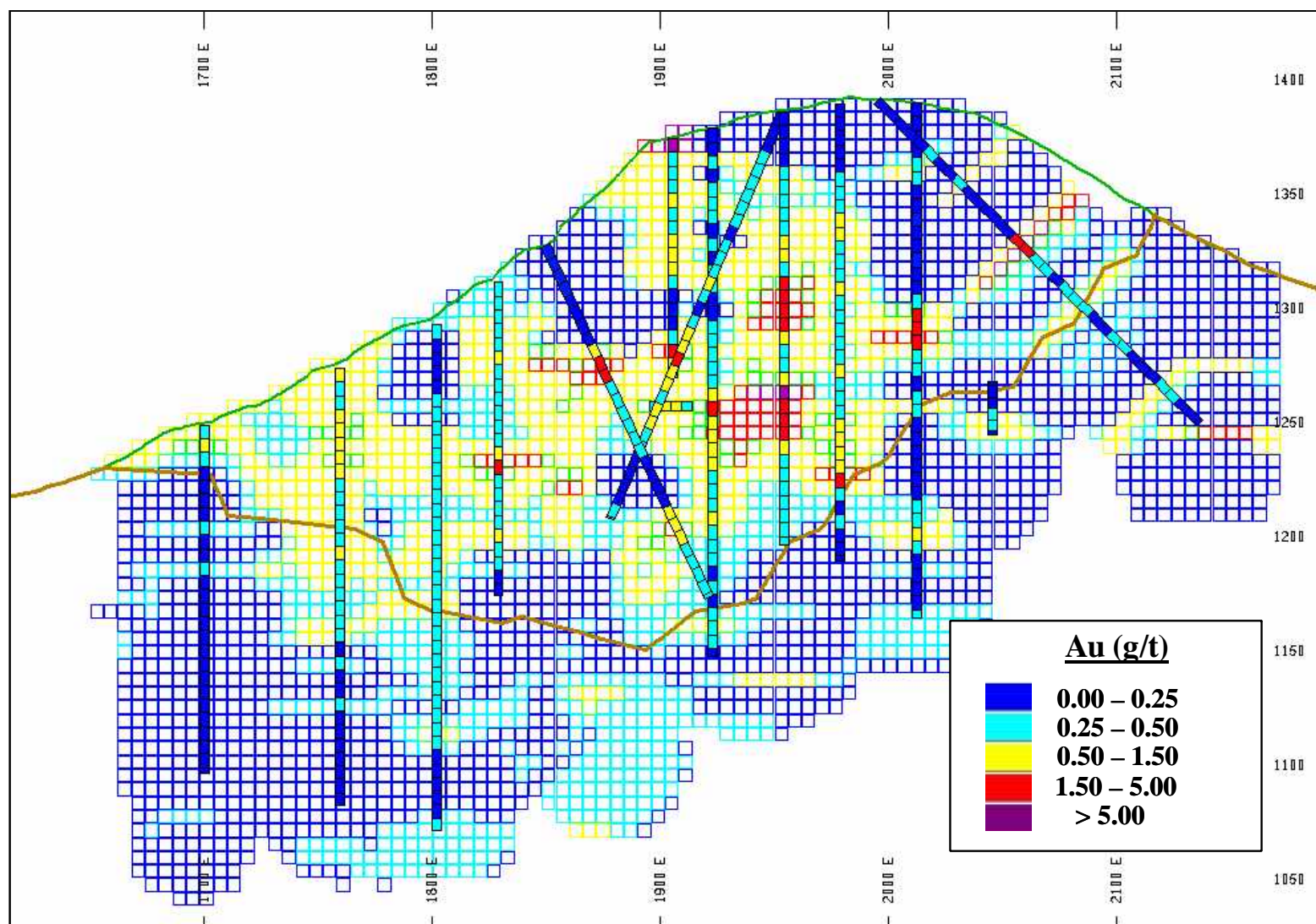
Figure 19.17
Block Model Cross Section 4500 North
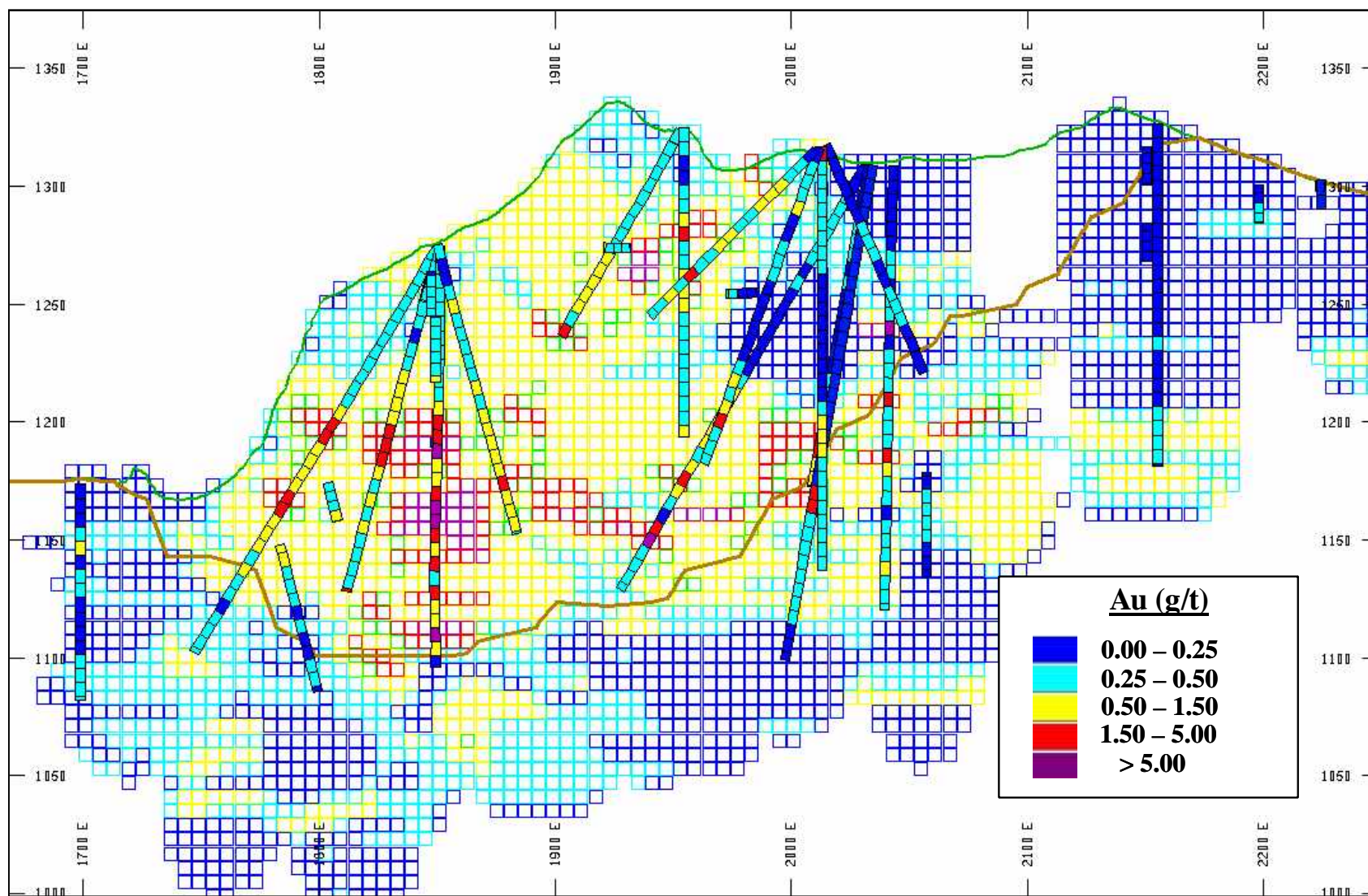
Figure 19.18
Block Model Cross Section 1850 East
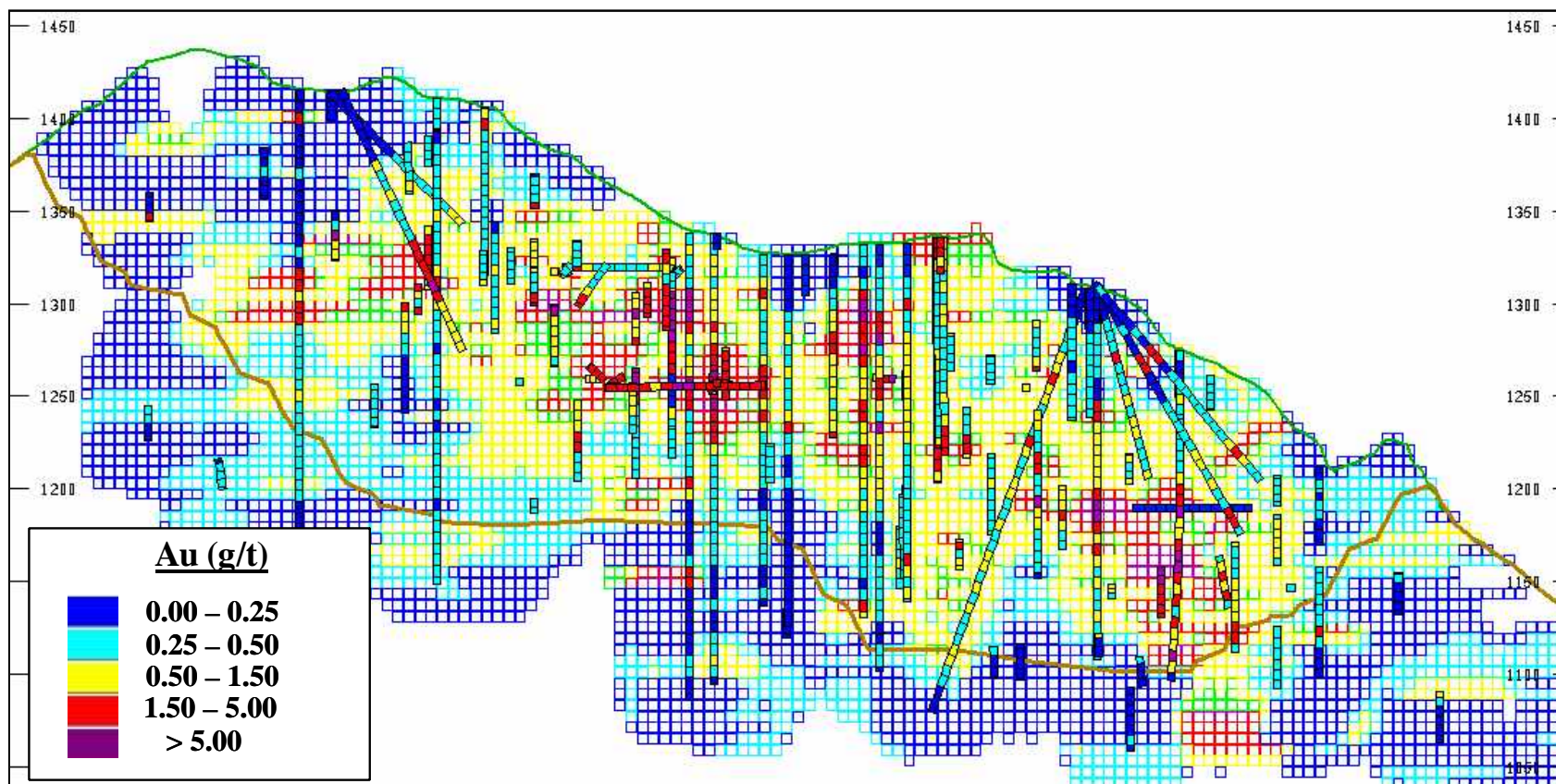
Figure 19.19
Block Model 1350 Elevation Plan
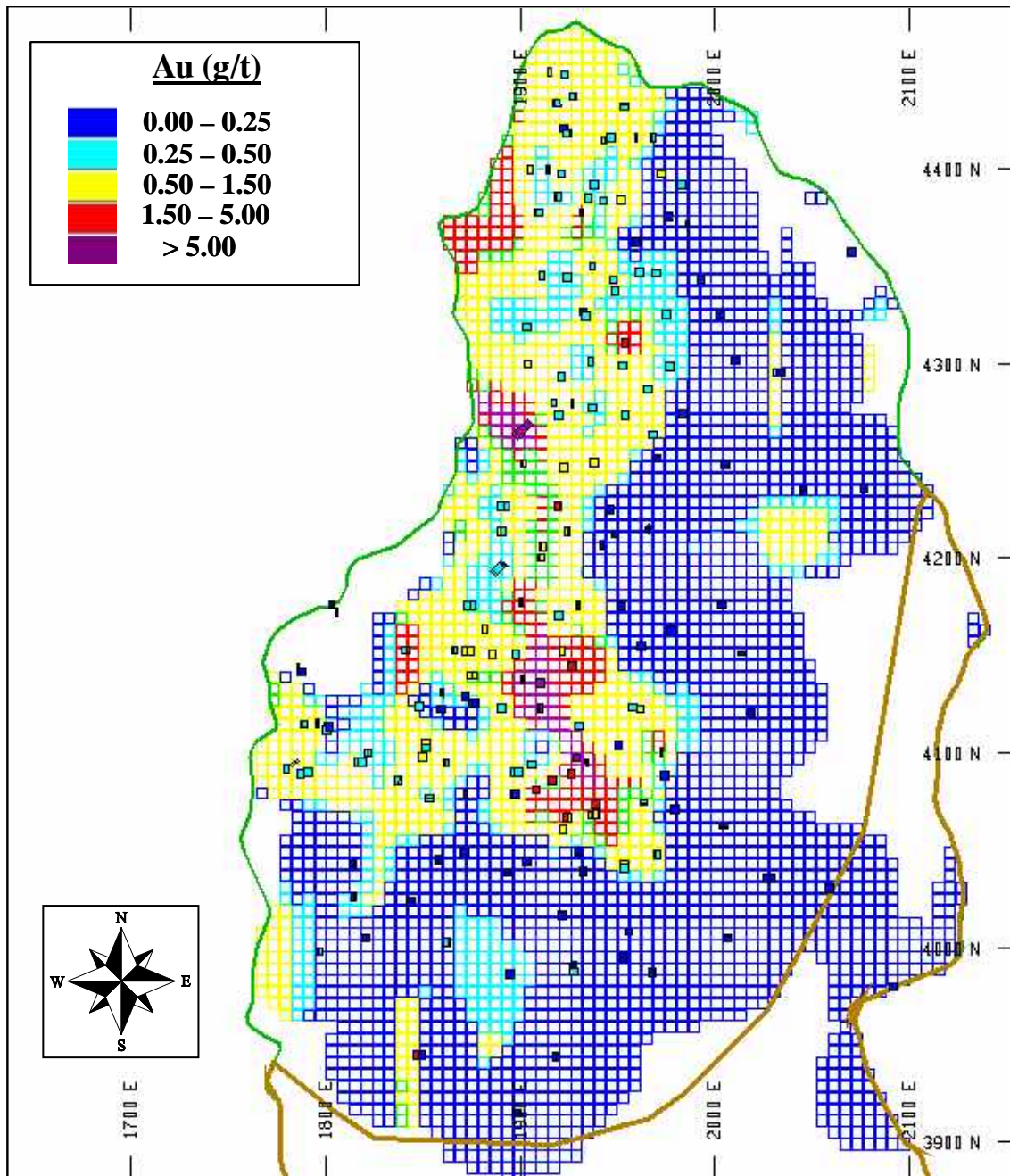
Figure 19.20
Block Model 1254 Elevation Plan
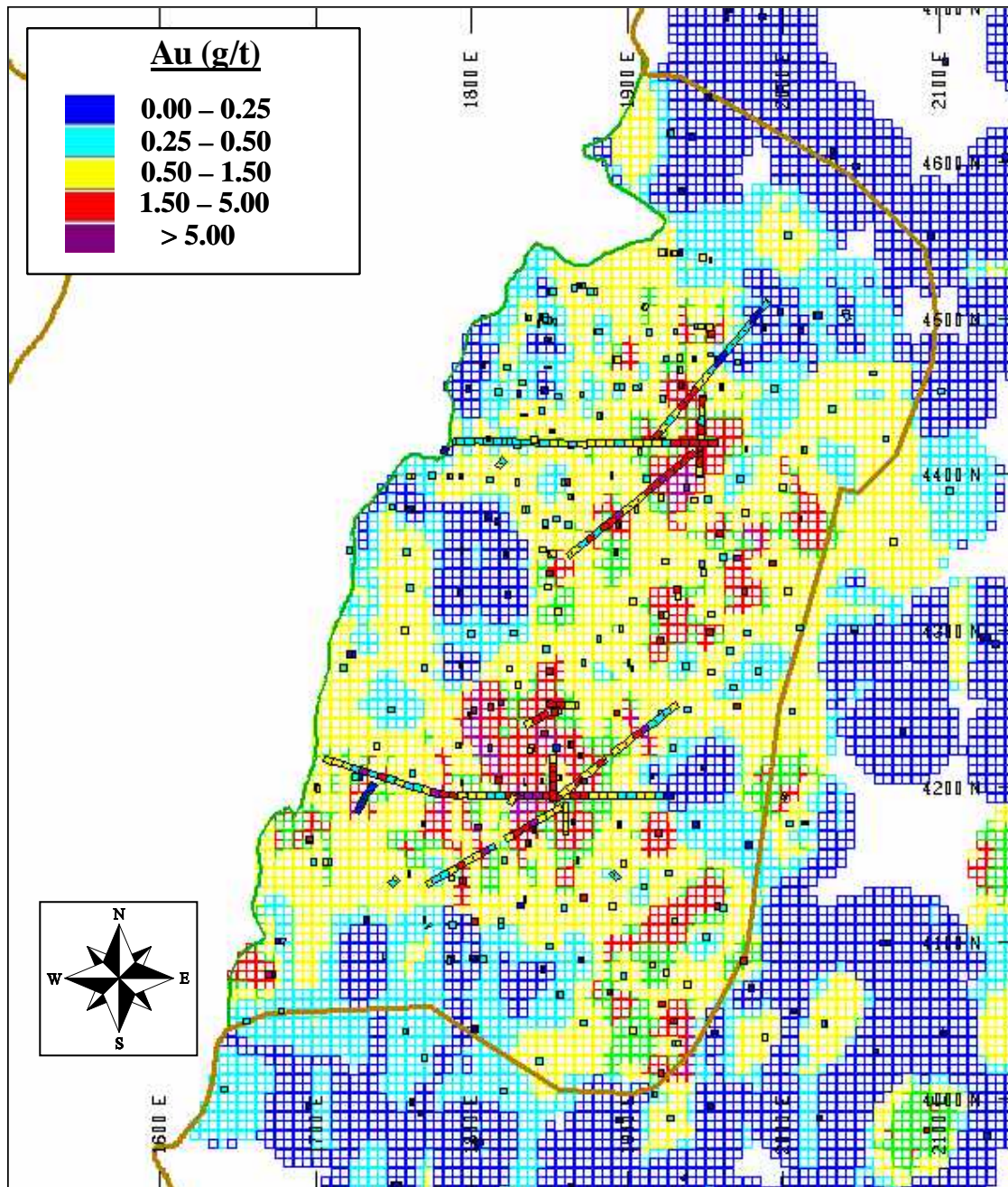
Figure 19.21
Block Model 1158 Elevation Plan
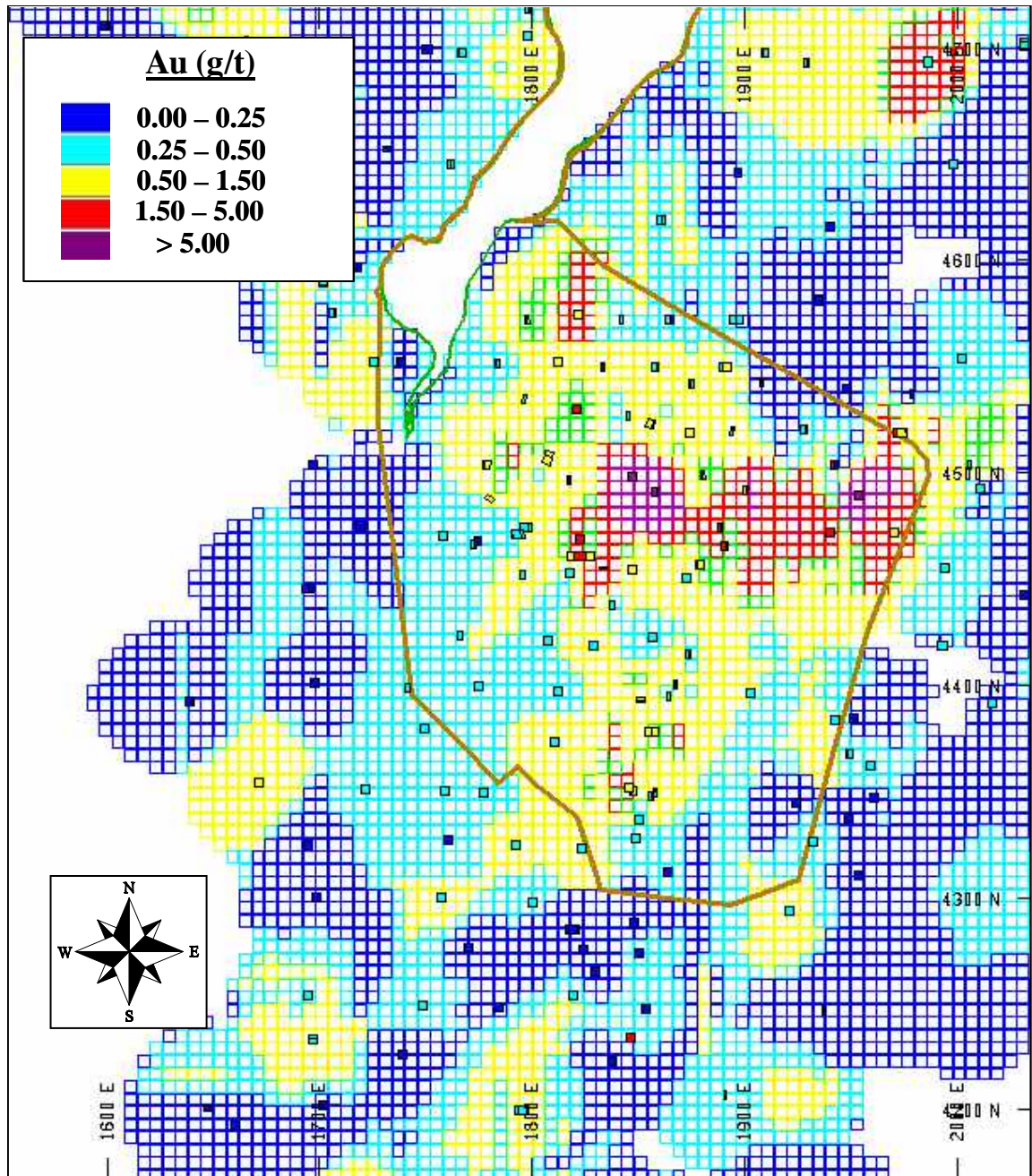
Figure 19.22
Data Above Cutoff
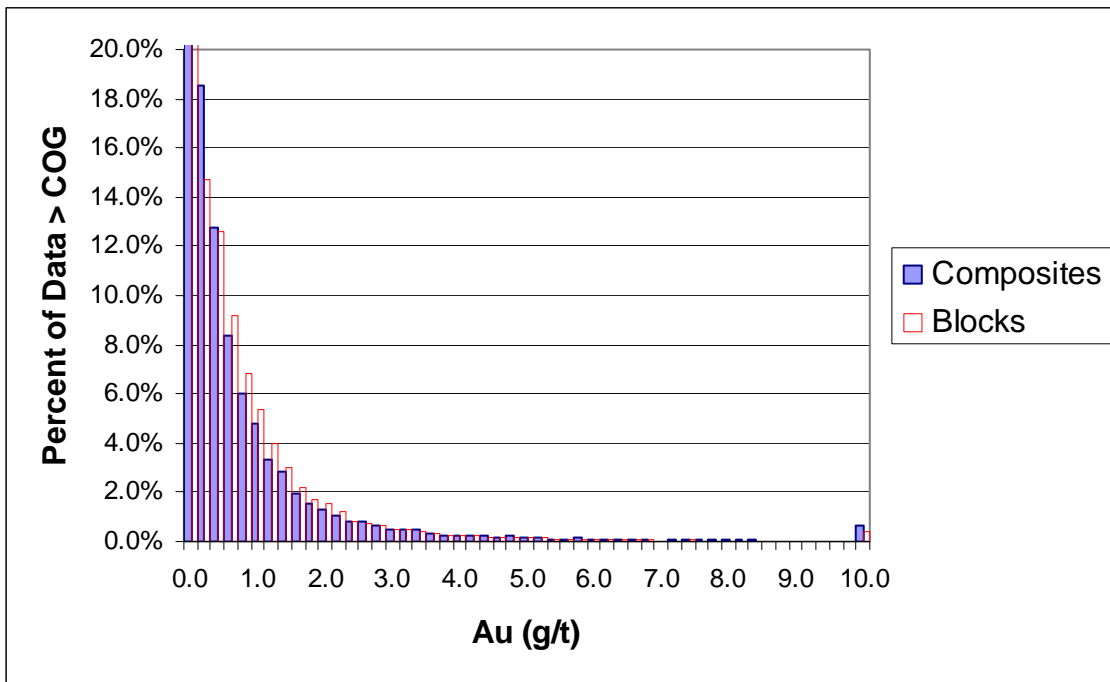
Figure 19.23
Mean Grade Above Cutoff
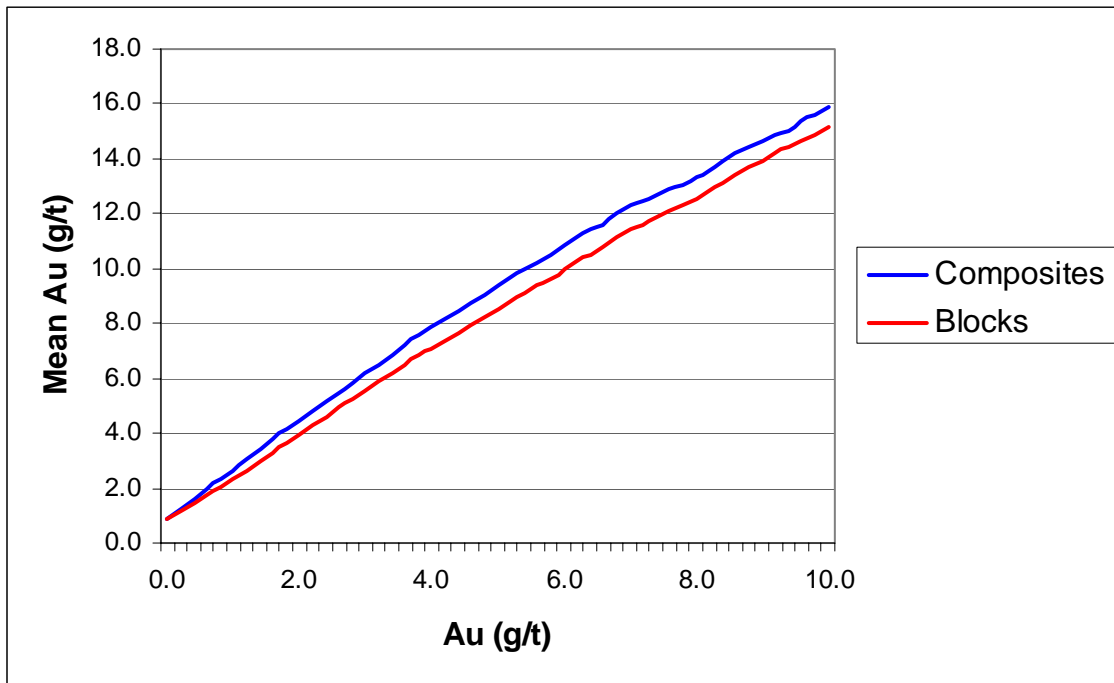
Figure 19.24
Block Regularization
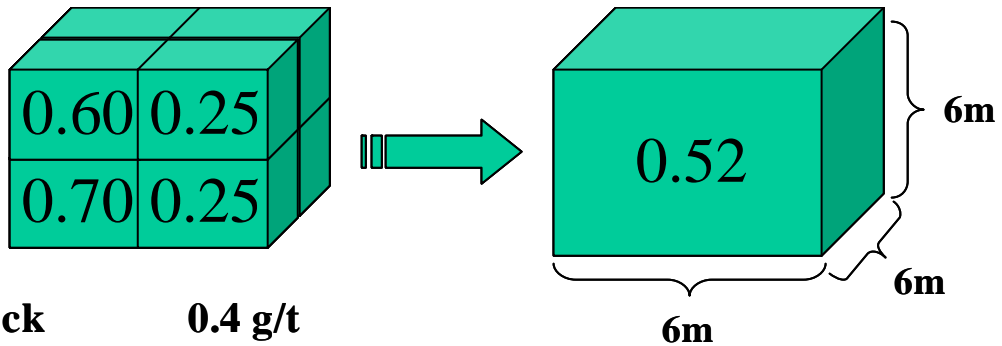
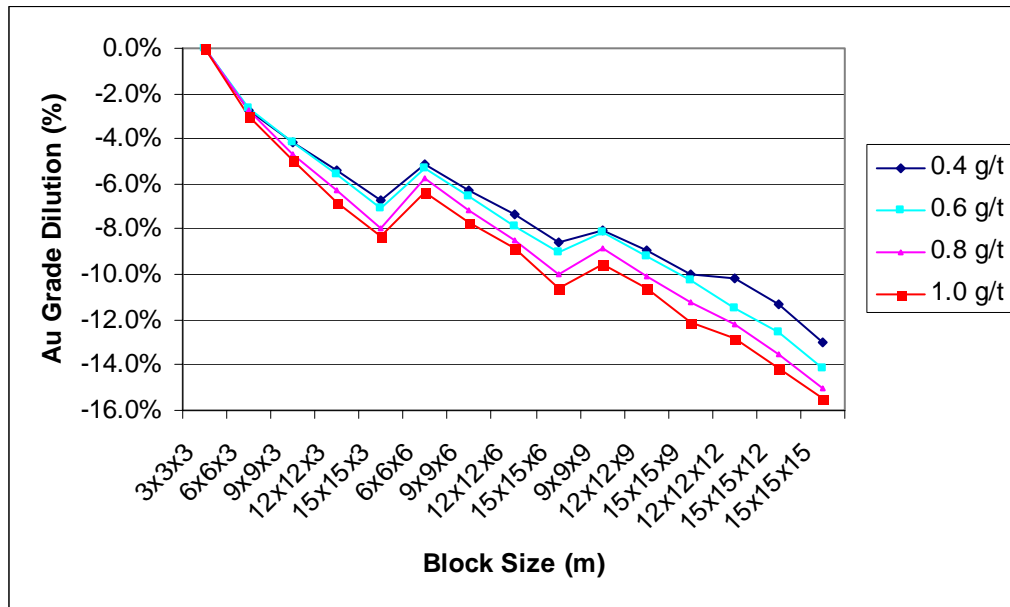| Block Grade	0.4 g/t Indicator
0.60	1.0
0.25	0.0
0.25	0.0
0.30	0.0
0.70	1.0
0.80	1.0
0.45	0.0
0.80	1.0
0.52	0.50

Regularized
Block Grade = 0.52
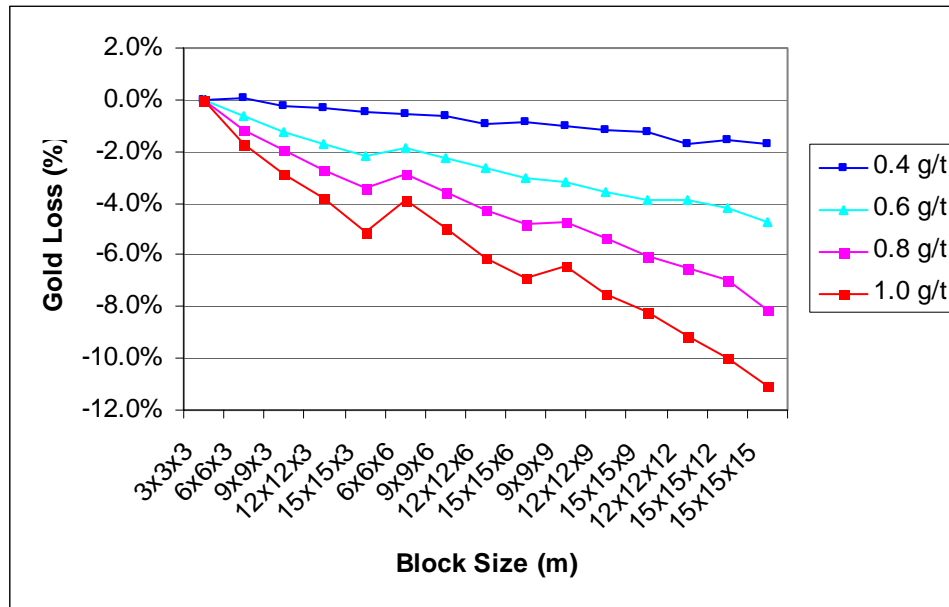
At a 0.50 g/t cutoff
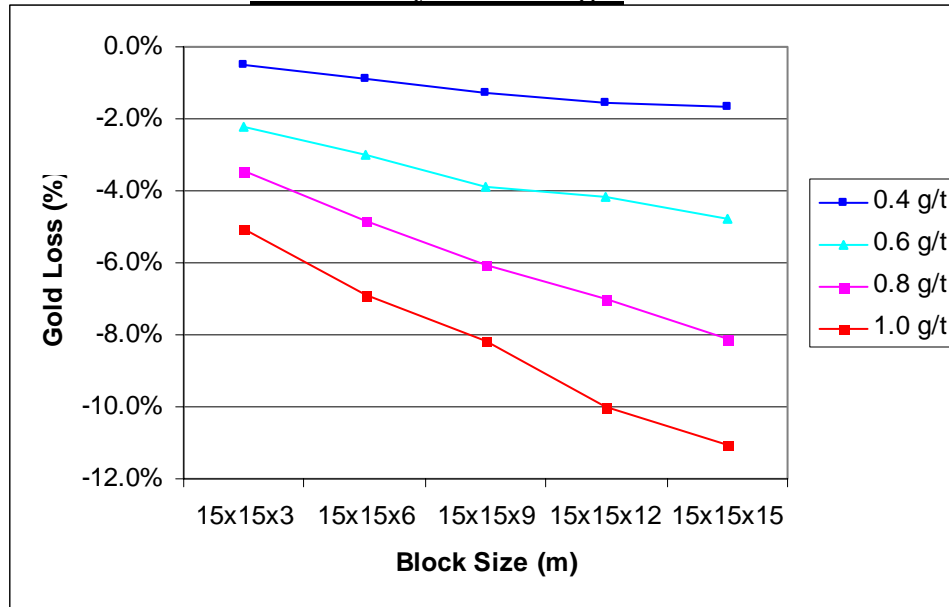**50% Dilution**

Figure 19.25
Tonnage Dilution by SMU
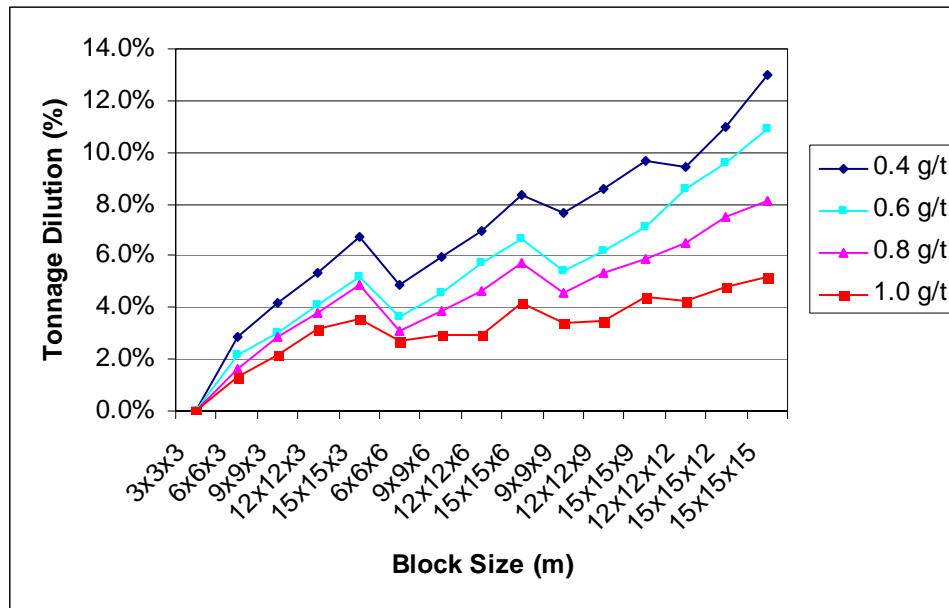
Figure 19.26
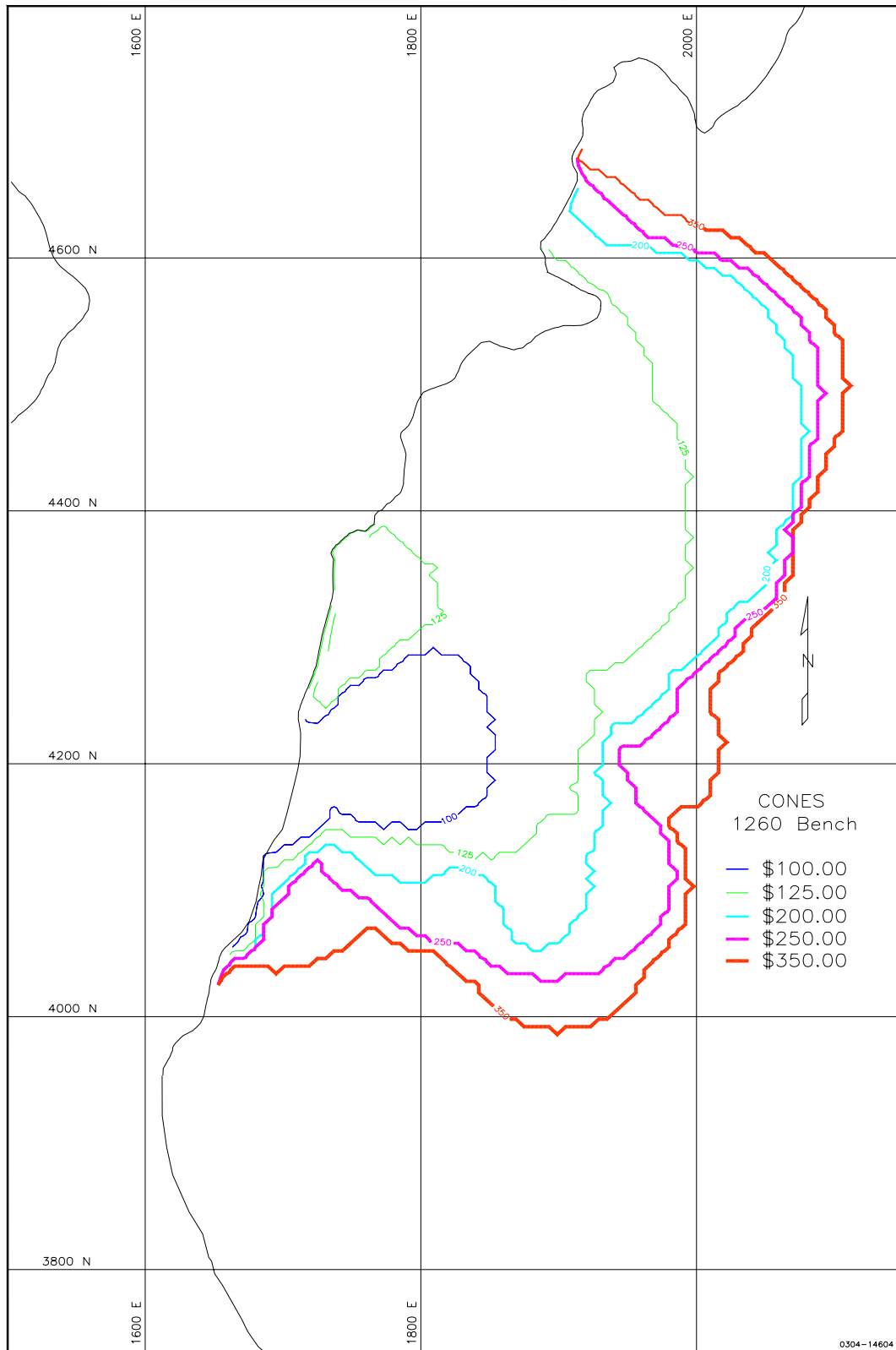**Grade Reduction for Various SMU's**

Figure 19.27
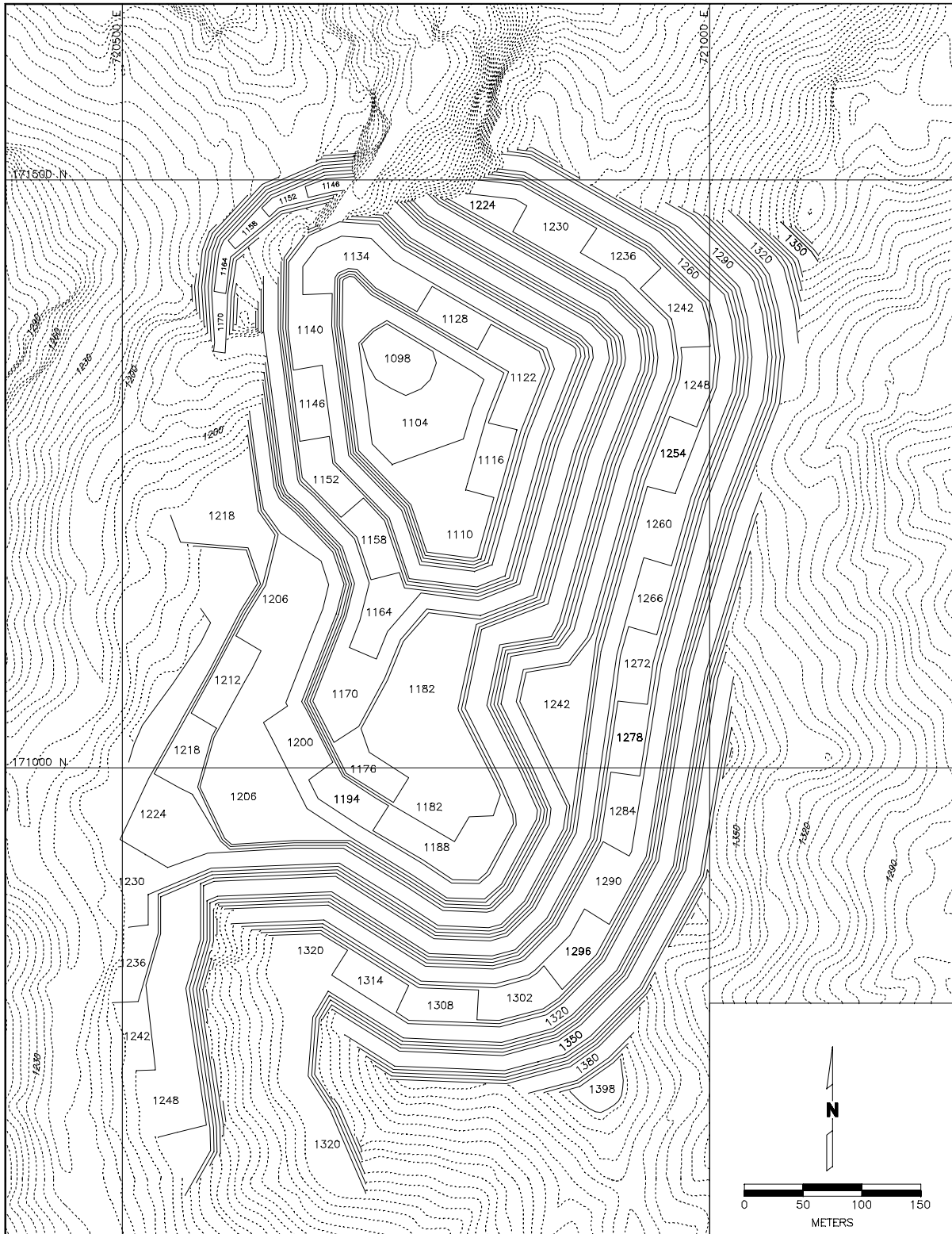# Gold Loss By SMU

Figure 19.28
Gold Loss by Bench Height
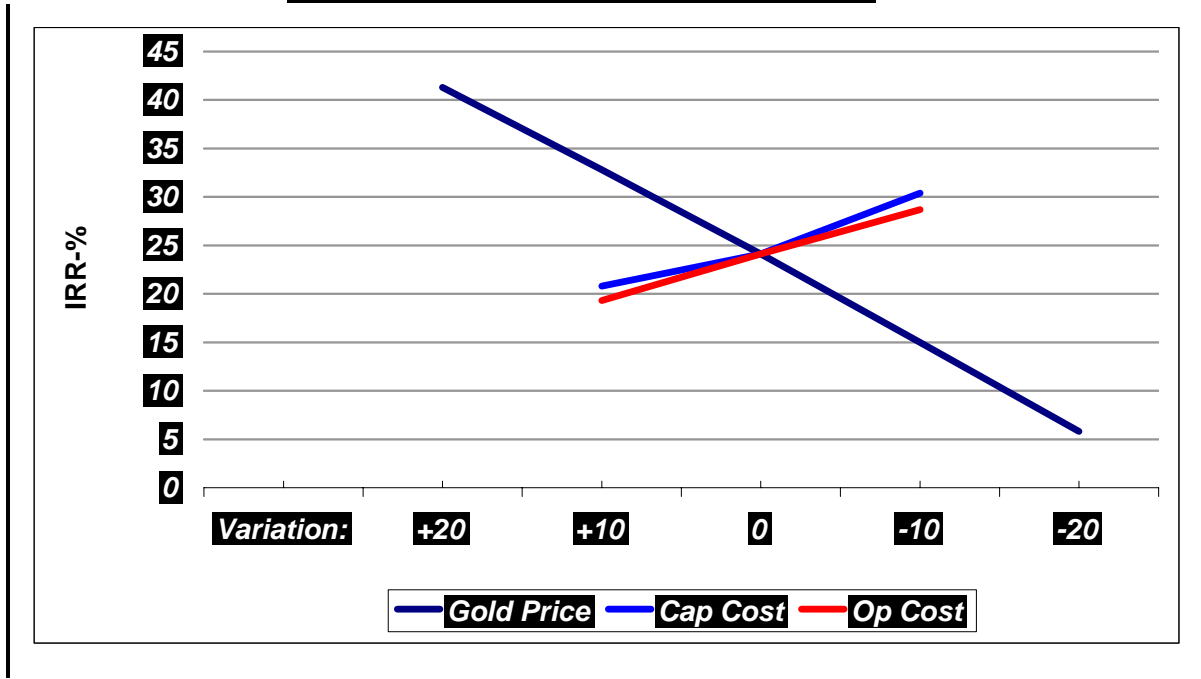
Figure 19.29
Cone Outlines on 1260 Bench

Figure 19.30
Mulatos Final Pit, Estrella

Figure 25.1
IRR - % Variation in Economic Parameter

Signatures

Pursuant to the requirements of the Securities Exchange Act of 1934, the registrant has duly caused this report to be signed on its behalf by the undersigned, thereunto duly authorized.

ALAMOS GOLD INC.
(Registrant)

July 26, 2004   By: /s/ Sharon L. Fleming
Date   Sharon L. Fleming
                            Corporate Secretary