PAN AMERICAN SILVER CORP
Form 6-K
January 31, 2008

UNITED STATES
SECURITIES AND EXCHANGE COMMISSION
Washington, D.C. 20549
FORM 6-K
REPORT OF FOREIGN PRIVATE ISSUER TO RULE 13A or 15D-16
UNDER THE SECURITIES EXCHANGE ACT OF 1934

For the Month of: January, 2008 **File No.: 000-13727**

PAN AMERICAN SILVER CORP.

(Translation of Registrant s Name into English)

Suite 1500, 625 Howe Street Vancouver British Columbia, Canada V6C 2T6

(Address of Principal Executive Office)

Indicate by check mark whether the registrant files or will file annual reports under cover of Form 20F or Form 40F:
Form 20F o Form 40F þ

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). Yes o No þ

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). Yes o No þ

Indicate by check mark whether the registrant by furnishing the information contained in this Form is also thereby furnishing the information to the Commission pursuant to Rule 12g3-2(b) under the Securities Exchange Act of 1934.
Yes o No þ

If Yes is marked, indicate below the file number assigned to the registrant in connection with rule 12g-3-2(b): 82 - __________.

Submitted herewith:

1. **Form 43-101 Technical Report for the La Colorada Property.**

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.

PAN AMERICAN SILVER CORP.

Date: January 30, 2008

Robert Pirooz

General Counsel

TECHNICAL REPORT
FOR THE
LA COLORADA PROPERTY
ZACATECAS, MÉXICO
Effective Date: September 30, 2007
PREPARED BY:
Andrew Sharp, AusIMM
Michael Steinmann, P.Geo
Martin Wafforn, P.Eng

TABLE OF CONTENTS

LIST OF FIGURES

LIST OF TABLES

1.0 TITLE PAGE

This Technical Report has been prepared in accordance with National Instrument 43-101 - *Standards of Disclosure for Mineral Projects* (NI 43-101) and the contents herein are organized and in compliance with Form 43-101F1 *Contents of the Technical Report* (Form 43-101 F1). The first two items are the Title Page and the Table of Contents presented previously in this report. They are mentioned here simply to maintain the specific report outline numbering required in Form 43-101F1.

2.0 TABLE OF CONTENTS

See discussion in Section 1.

3.0 SUMMARY

3.1. Background

Pan American Silver Corp. (PAS) prepared this Technical Report in support of its public disclosure of mineral reserve and mineral resource estimates as of September 30, 2007, as required by NI 43-101.

Mr. Andrew Sharp, AusIMM member, Planning Manager Mexico Operations of Minera Corner Bay S.A. de C.V. (Minera Corner Bay), a wholly-owned subsidiary of PAS, Dr. Michael Steinmann, P. Geo., Senior Vice President of Exploration and Geology of PAS, and Mr. Martin Wafforn, P.Eng., Vice President of Mine Engineering of PAS, are authors of this Technical Rreport. Each of Mr. Sharp, Dr. Steinmann and Mr. Wafforn is a Qualified Person (QP) as that term is defined in NI 43-101.

3.2. Property Ownership, Location and Description

The La Colorada property was acquired by PAS in April 1998 through its wholly owned subsidiary Plata Pan Americana S.A., de C.V. (Plata).

The La Colorada property is located in the Chalchihuites district, Zacatecas State, México, approximately 99 km south of the city of Durango and 156 km north-west of the city of Zacatecas. The district s general co-ordinates are longitude 23°23 N and latitude 103°46 W. The La Colorada mine-site is accessible by road approximately1/2 hours south-east of the city of Durango. The road consists of 120 km of a paved two-lane highway (Highway 45), and 23 km of public, all weather gravel road. The access from Zacatecas takes approximately the same time on similar types of roads. Durango and Zacatecas are serviced by daily flights from México City, other major centers in México and direct flights from some cities in the United States.

The La Colorada property is comprised of 37 exploitation claims totalling 2,864.1 ha (figures 2, 3 and 4). In addition, PAS also has control over approximately 571 ha of surface rights covering the main workings, namely the Candelaria, Campaña, Recompensa and Estrella mines.

3.3. Geology and Mineralization

The La Colorada property is located on the eastern flanks of the Sierra Madre Occidental at the contact between the Lower Volcanic Complex and the Upper Volcanic Supergroup. The La Colorada property lays 16km southeast of Chalchihuites and 30km south-southwest of Sombrerete, two mining camps with significant silver and base metal production from veins and associated skarn deposits (San Martin and Sabinas mines).

The oldest rocks exposed in the mine area are Cretaceous carbonates and calcareous clastic rocks of the Cuesta del Cura and Indidura formations. Overlying the calcareous rocks is a conglomerate unit containing clasts derived mostly from the subadjacent sedimentary rocks. In the Chalchihuites district this unit is called the Ahuichila Formation and is of Early Tertiary age.

Most of the outcrop in the mine area is represented by intermediate to felsic volcanic rocks of the regional Lower Volcanic Complex. This unit is identified as a trachyte in older mine data, although recent petrography indicates that it is actually an altered dacite. There are several subgroups within this unit, including plagioclase porphyry, crystal to crystal-lapilli tuffs, and volcanic breccias. Generally these sub-units do not form mappable units.

East to northeast striking faults form the dominant structures in the project area and play a strong role in localizing mineralization. Most of these faults dip moderately too steeply to the south and juxtapose younger hangingwall strata against older footwall rocks. Evidence suggests down-dip motion on these faults; however, most of the faults have been reactivated at some point so the movement direction during the initial formation is uncertain. Stratigraphic contacts are displaced from ten to over a hundred metres lower on down-dropped blocks.

The mineralized veins at La Colorada contain both oxide and sulphide material. The depth below surface and the permeability of the mineralized zone controls the level of oxidation in the veins. The most common sulphide minerals are galena, sphalerite, tetrahedrite, argentite, and pyrite.

3.4. Exploration and Development

The bulk of PAS exploration on the La Colorada property has been surface and underground diamond drilling and underground drifting on the veins and mineralized zones. Exploration work conducted by PAS as of September 30, 2007 includes 53,253 metres of surface and underground diamond drilling and approximately 22,600 metres of underground drifting. The drifting was completed along the vein for stope extraction, ramp and stoping access, drifting in mineralized structures has been mapped for geology and sampled. NQ and HQ sized core was obtained from surface drilling and underground diamond drilling is typically done with BQ sized core. In certain cases, HQ sized core was used in underground drilling in an attempt to improve drill core recovery. Prior to PAS involvement in the La Colorada project, previous operators had drilled 131 holes for a total of 8,665 metres. These holes were not used in PAS reserve or resource calculation, with the exception of four holes where the original core was found and assayed by PAS. Drill holes generally range in length from 100 to 300 metres with dips of plus 45º to minus 90º. Standard logging and sampling processes have been used to record information from the holes drilled by PAS.

There are on-going development and exploration programs in place in order to secure the future production from the mine. In 2008, PAS plans to complete 18,232 metrrs of surface and underground exploration and definition diamond drilling. PAS also plans approximately 6,300 metres of underground development (1,750 metres in ore sills) during 2008.

3.5. Mineral Resource and Reserve Estimates as at Sep. 30, 2007

The proven and probable mineral reserves at the La Colorada mine as at September 30, 2007 were estimated to be as shown in Table 1. This mineral reserve estimate was calculated using a price of $11.00 per ounce of silver, $600 per ounce of gold, $2,100 per tonne of zinc, $1,700 per tonne of lead and was prepared under the supervision of and reviewed by Andrew Sharp, AusIMM member, Planning Manager of Mexican Operations of Minera Corner Bay, and Dr. Michael Steinmann, P. Geo., Senior Vice President of Exploration and Geology of PAS. Each of Mr. Sharp and Dr. Steinmann is a QP as that term is defined in NI 43-101.

Table 1 La Colorada Mineral Reserves

	Tonnage		**Grade**		
Category	**kT**	**Ag g/t**	**Au g/t**	**Pb %**	**Zn %**
Proven	449.40	421.57	0.46	0.01	0.01
Probable	566.50	460.12	0.53	0.01	0.01
Total Reserve	**1,015.90**	**443.07**	**0.50**	**0.01**	**0.01**

Notes:

1. Total grades of silver and zinc are shown as contained metal before mill recoveries are applied.

2. La Colorada mineral reserves have been estimated at a cut off value per tonne of $66.53 in the Calendaria Mine and $58.31 in the Estrella Mine for oxide ore and $58.48 per tonne in sulphide ore.

3. The geological model employed

for La Colorada involves geological interpretations on sections and plans derived from core drill hole information and channel sampling.

4. Mineral reserves have been estimated using the O Hara dilution formula, which typically adds 20% to 50% dilution at zero grade depending on dip angle and vein width. As a result of reconciliation to actual production the mining dilution is increased by a further 13%.

5. Mineral reserves have been estimated using a mining recovery of 85-94% (pillars are left in some thicker zones leading to lower mining recovery). A further 7.5% subtracted from the grade with no change in tonnage to further account for other mining losses.

6. Mineral reserves were estimated based on the use of cut and fill mining methods. The mining rate is projected to be a maximum of 940 tpd ore for the full year of 2008. The processing plants have the capacity to process more than this and are assumed to process all of the ore produced by the mine in each year.

7. Mineral reserves are estimated using polygonal methods on longitudinal sections.

8. Mineral reserves were estimated using a price of $11.00 per ounce of silver, $600 per ounce of gold, $2,100 per tonne of zinc and $1,700 per tonne of lead.

9. Environmental, permitting, legal, title, taxation, socio economic, political, marketing or other issues are

not expected to
materially affect
the above
estimate of
mineral
reserves.

The measured, indicated and inferred mineral resources at the La Colorada Mine as of September 30, 2007 were estimated to be as shown in Table 2. This mineral resource estimate was calculated using a price of $11.00 per ounce of silver, $600 per ounce of gold, $2,100 per tonne of zinc, $1,700 per tonne of lead and was prepared under the supervision of and reviewed by Andrew Sharp, AusIMM member, Planning Manager of Mexican Operations of Minera Corner Bay, and Dr. Michael Steinmann, P. Geo., Senior Vice President of Exploration and Geology of PAS. The mineral resources shown in Table 2 are in addition to the mineral reserves shown in Table 1.

Table 2 La Colorada Mineral Resources

Category	Tonnage k tonne	Grade Ag g/t	Au g/t	Pb %	Zn %
Measured	186.79	329.66	0.51	0.01	0.01
Indicated	547.33	243.62	0.30	0.01	0.01
Total Resource	**734.11**	**265.51**	**0.36**	**0.01**	**0.01**
Inferred	1701.18	346.46	0.39	0.02	0.02

Notes:

1. PAS reports mineral resources and mineral reserves separately. Reported mineral resources do not include amounts identified as mineral reserves.

2. The geological model employed for La Colorada involves geological interpretations on sections and plans derived from core drill hole information and channel sampling.

3. Mineral resources have been estimated using the O Hara dilution formula, which typically adds 20% to 50% dilution at zero

grade depending on dip angle and vein width. .As a result of reconciliation to actual production the mining dilution is increased by a further 13%.

4. Mineral resources have been estimated using a mining recovery of 85-94% (pillars are left in some thicker zones leading to lower mining recovery). A further 7.5% is subtracted from the grade with no change in tonnage to further account for other mining losses.

5. Mineral resources were estimated based on the use of cut and fill mining methods. The mining rate is projected to be a maximum of 940 tpd ore for the full year of 2008. The processing plants have the capacity to process more than this and are assumed to process all of the ore

produced by the mine in each year.

6. Mineral resources are estimated using polygonal methods on longitudinal sections.

7. Mineral reserves were estimated using a price of $11.00 per ounce of silver, $600 per ounce of gold, $2,100 per tonne of zinc and $1,700 per tonne of lead.

8. Environmental, permitting, legal, title, taxation, socio economic, political, marketing or other issues are not expected to materially affect the above estimate of mineral resources.

9. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The mineral resource estimation has been done using the polygonal method and corrections for mining method, mining recovery, dilution from wall rocks and dilution from backfill have been taken into account. Mining blocks were created from the variograms and classified as measured, indicated or inferred based on the relative confidence of the supporting data for each evaluated block.

Erratic high Ag values have been corrected for the mineral resource calculation. The La Colorada mineral deposit contains high grade, minable ore shoots and a simple arithmetic top cut to the database would eliminate entire high grade areas. In order to prevent that, the spatial location of a sample has been taken into account. Samples are collected along a structure and are plotted as silver gram per tonne (g/t) grade, width (m) of the vein and Ag grade multiplied by the width in order to identify minable ore shoots along the veins. For an easy method to locate the ore shoots, trend lines are plotted over the three datasets. Single outliers, or non-minable small ore shoots, are visually identified and the grades are replaced by the grades of the predicted trend line. The corrections are applied before the vein samples and mineralized footwall / hangingwall samples are composited. Although this method represents a rather unusual way of applying a top cut, the authors of this Technical Report agree that it represents a valid method for the La Colorada deposit, eliminating high grade outliers and with that reducing risk from the mineral resource estimation.

3.6. Mining Operations

Two operating mines exist within La Colorada: Candelaria Mine and La Estrella Mine. The underground Candelaria Mine has various veins that are currently in production. There are two types of ore (sulphide and oxide ore) that are processed separately in the processing plant within the Candelaria Mine. Ore that is favourable to flotation produces a lead / silver and zinc concentrates. Other ore (mostly highly oxidized but includes transitional and sulphide ore) is processed through a separate cyanide leach plant (termed the oxide plant) and produces dorè bars. The Estralla Mine has a single oxide vein that is also currently being mined and processed through the oxide plant.

A third mine, the Recompensa mine is currently the subject of an exploration project that includes surface and underground diamond drilling from a ramp to the surface.

The 2008 La Colorada mine plan is based on providing 540 tpd of ore to the oxide circuit (maximum capacity 650 tpd) that was commissioned in 2004 and a further 400 tonnes per day (tpd) of ore to the sulphide circuit which was recently expanded from 100 tpd production. Of the 540 tpd of oxide ore that is planned to be produced during 2008, it is estimated that 390 tpd will be mined from the Candelaria Mine and 150 tpd from the Estrella Mine. The expansion to the sulphide plant was commissioned in June 2007 and the plant now has a maximum capacity of 450 tpd. All of the sulphide production is scheduled to be mined from the Candelaria Mine at a rate of 400 tpd. The mining method used in both mines is mechanized cut and fill using waste rock as backfill.

The mines are being developed to permit fast and efficient movement of equipment, personnel and materials via a system of ramps that connect back to the shaft for haulage of ore and some waste to the surface. Main ramps that are used by haulage trucks are developed 3.6 metres wide by 3.6 metres high with the face drilling normally done by a one boom jumbo and bolting done either using hand held drills from the muck pile or using the one face jumbo that is equipped with a boom capable of drilling holes for split set bolts. The waste is removed using 3.5 cubic yard scooptrams, and where possible, is taken to a stope that is being backfilled for disposal. Stope accesses are typically 2.4 metres wide by 2.4 metres high to permit 2 cubic yard capacity scooptrams into the stopes. These accesses are normally drilled and bolted using hand held drills.

3.7. Authors Conclusions

This Technical Report demonstrates that the mineral reserves and mineral resources presented in this Technical Report will be economic with the forecast metal prices and other assumptions presented herein. Based on the current mineral reserve estimates, the mine is projected to operate until the end of 2011. This projected mine life may increase if future resources are converted to reserves. The undiscounted net present value (NPV) for the La Colorada mine is $9.77M based in the mineral reserves and mineral resources. The current realized metal prices are higher than those used for the mineral reserve and mineral resource calculations and for the economic analysis presented in this Technical Report.

In the opinion of the authors of this Technical Report, the diamond drilling and channel sampling information that has been collected is of sufficient density for mineral resource and mineral reserve estimation.

The QA/QC programs are conducted under the direct supervision of PAS geology staff and periodically revised by Michael Steinmann, P.Geo. The authors of this Technical Report have relied on the data verification work conducted by the geology staff at La Colorada Mine. Summary results used in the resource estimation have been verified by the authors of this Technical Report.

This report details the methodology employed and demonstrates why the authors of this Technical Report conclude that the continued operation of the La Colorada Mine is technically feasible and economically viable. It is the authors opinion that the data contained herein is of sufficient quality and reliability to make the conclusions stated.

3.8. Author s Recommendation

As the mine is currently in operation, the work programs necessary to maintain annual updates to the mineral reserve estimates are in place and being conducted on a daily basis by a full complement of technical and operating staff at the mine. The costs for these work programs are included in the annual operating budgets, mine plan and life of mine (LOM) plan that are shown in section 25.6 Capital and Operating Costs. Martin Wafforn and Michael Steinmann visit the La Colorada Mine on a regular basis throughout the year and make any necessary revisions or improvements to the estimation methodologies. Mr. Andrew Sharp has been based at the La Colorada mine for the last year and has worked there on a daily basis. It is recommended that PAS continues to follow the life of mine plan and make the capital investments that are detailed in that plan. It is further recommended to continue to follow the current sampling and quality control programs as may be revised from time to time by Michael Steinmann, P.Geo. It is also recommended to continue with the diamond drilling program and the related sampling and quality control programs in order to assure sufficient data density for future new resource estimations in deeper or lateral parts of the mine as well as for satellite deposits. The mine has a budget in 2008 of US $1.5M in order to conduct exploration and definition drilling programs in an attempt to convert resources to reserves and locate new ore bodies. These exploration programs are closely supervised and revised by the Senior V.P. of Geology and Exploration for PAS, Dr. Michael Steinmann, P.Geo.

The authors of this Technical Report recommend that the mine should continue to operate and that the mineral reserve and mineral resource statement presented herein be adopted.

4.0 INTRODUCTION

This Technical Report has been prepared for filing in accordance with NI 43-101 and the format and contents of this Technical Report are intended to conform to Form 43-101 F1. This Technical Report has been prepared for PAS for the purpose of updating the mineral reserve and mineral resource estimates for the La Colorada property. Mr. Andrew Sharp, AusIMM member, Planning Manager Mexico Operations of Minera Corner Bay, serves as the Qualified Person responsible for preparing sections 3, 4, 5, 6, 7, 8, 18, 20, 21, 22, 23, and 24 and Figures 8, 9, 10, 11, 12, 13, 14, 15 and 16 of this Technical Report. Dr. Michael Steinmann, P.Geo., Senior Vice President Geology and Exploration for PAS, serves as the Qualified Person responsible for sections 1, 2, 9, 10, 11, 12, 13, 14, 15, 16, 17 and 19 and Figures 1, 2, 3, 4, 5, 6, and 7 of this Technical Report. Mr. Martin Wafforn, P.Eng., Vice President of Mine Engineering for PAS, serves as the Qualified Person responsible for section 25 and figures 17, 18, 19 and 20.

Andrew Sharp has worked at the La Colorada Mine since November 2006 and Michael Steinmann and Martin Wafforn continuously supervise projects at La Colorada and visit the site on a regular basis.

Data, reports, and other information used for the compilation of this Technical Report were obtained from personnel in the PAS offices in Vancouver, British Columbia, the Plata office in Durango, México and from the La Colorada Mine offices in Zacatecas, México. This Technical Report is based on work conducted by PAS geologists, engineers and metallurgists, as well as third party consultants retained by PAS. Specifically, information and data for the mineral resource and mineral reserve estimates were obtained from La Colorada geology department personnel in México and information and data for matters pertaining to metallurgy and processing, cost estimates, environmental and geotechnical investigations, and economic analyses were provided by PAS.

Information and data was also obtained from certain corporate documents, including:

Feasibility Study, La Colorada Mine, México, June 22, 2000 (the Feasibility Study);

La Colorada Project, México. Feasibility Update, February 2002 (the Updated Feasibility Study);

La Colorada Mine Project, Zacatecas, Technical Report, August 29, 2003

Annual Information Forms of Pan American for 1999, 2000, 2001, 2002, 2003, 2004, 2005 and 2006; and

La Colorada Mine Project, Zacatecas, Technical Report, March 17, 2006

The Feasibility Study and the Updated Feasibility Study relied on various documents prepared by third party engineering and consulting firms, including:

Structural Analysis, La Colorada Mine, Lewis Geoscience, October 1998

La Colorada Project Geologic Modeling and Resource Estimation Report, MRDI, May 2000

Flotation and Cyanidation Study on Samples from La Colorada, Process Research Associates Ltd., May 2000

Estudio de Cianuración, Luismin Labs, April 2000

Updated Basic Engineering Report for the La Colorada Project, Agra Simons, May 2000

Hoisting System Evaluation, Beacon Hill Consultants, June 2000

Mining Calculations Detail, Beacon Hill Consultants, June 2000

Mina La Colorada Environmental Impact Assessment, Dew Point International, LLC, May 2000

Tailings Facility Design Report and Addendum, AGRA Earth and Environmental, May 2000

Mina la Colorada Environmental Action Plan, Dew Point International LLC, August 2002

Dewatering Requirements La Colorada Mine Golder Associates, July 2004

Dewatering Requirements November 2004, update La Colorada Mine Golder Associates, December 2004

All tonnages stated in this Technical Report are dry metric tonnes (dmt) unless otherwise specified. Ounces pertaining to silver metal content are expressed in troy ounces.

All dollar values stated in this report are U.S. dollars.

5.0 RELIANCE ON OTHER EXPERTS

Andrew Sharp, Michael Steinman and Martin Wafforn, as authors of this Technical Report, have relied upon the references, opinions and statements from various Qualified and Non-Qualified Persons contained within the reports referenced in Section 23 References. These reports, documents, and statements were found to be generally well organized and presented, and where applicable, the conclusions reached are judged to be reasonable.

7

It is assumed that these reports and documents were prepared by technically qualified and competent persons. It is also assumed that the information and explanations given verbally to the QPs by the employees of both PAS and Plata, and the various consultants and contractors who provided the reports listed in Section 23.0 during the time of preparation of this Technical Report were essentially complete and correct to the best of each employee s, contractor s, or consultant s knowledge, and that no information was intentionally withheld. It is the authors opinion that the referenced materials are prepared and presented according to Mining and Engineering Industry Standards.

6.0 PROPERTY DESCRIPTION AND LOCATION

The La Colorada property is located in the Chalchihuites district, Zacatecas State, México, approximately 99 km south of the city of Durango and 156 km north-west of the city of Zacatecas (Figure 1). The district s general co-ordinates are longitude 23°23 N and latitude 103°46 W. The following figures show the location of the La Colorada Mine:

Figure 1 Location of the La Colorada Mine in Mexico

Figure 2, 3 & 4 La Colorada Mine, Mining Concessions

Figure 5 La Colorada Mine, Mine Site General Layout (view of the mine area)

Figure 6 La Colorada Mine, Geology Map

Property boundaries are defined by field surveys. A survey starting point is established on each property to be claimed. This survey starting point must be constructed of concrete and have a base of at least 60 cm by 60 cm. From the starting point the property boundaries are surveyed by a surveyor registered by Direccion General de Minas (DGM) and the property to be claimed filed with DGM.

The locations of all know mineralized veins and structures containing the mineral reserves and mineral resources are shown in Figures 2, 3 and 4. The plant site, tailings facility, mine workings and other infrastructure are shown in Figure 5.

6.1. Mineral Tenure

The La Colorada property is comprised of 37 exploitation claims (7 awaiting title) totalling 2,864.1 ha. The Mexican law has changed as of last year pertaining to designation of the claims as either exploration or exploitation claims. The Mexican government has removed the exploration claim status and everything is currently listed under an exploitation claim. The extent of the mineral tenure is shown in Figures 2, 3 and 4. In addition, Plata also has control over approximately 571 ha of surface rights covering the main workings, namely the Candelaria, Campaña, Recompensa and Estrella Mines. Table 3 lists the mining concessions owned by Plata. The concessions have been legally surveyed.

Table 3 Mining concessions registered for exploitation (mining)

NAME OF THE CLAIM	TITLE	HECTARES	PESOS PER HECTARE	TOTAL PESOS	TOTAL PESOS ADJUSTED	DATE OF EXPIRATION
UNIF VICTORIA EUGENIA	188078	285.6230	100.79	28,787.94	28,788.00	11/21/2040
VICTORIA EUGENIA I	204862	23.3187	100.79	2,350.29	2,350.00	5/12/2047
VICTORIA EUGENIA II	211166	49.0000	28.64	1,403.36	1,403.00	4/10/2050
VICTORIA EUGENIA III	204756	1.1262	100.79	113.51	114.00	4/24/2047
VICTORIA EUGENIA IV	217627	36.9357	28.64	1,057.84	1,058.00	8/5/2052
MARIETA	171833	9.0000	100.79	907.11	907.00	6/14/2033
CRUZ DEL SUR	170155	11.0977	100.79	1,118.54	1,119.00	3/16/2032
UNIFICACION CANOAS	211969	18.5052	100.79	1,865.14	1,865.00	3/15/2023
SAN CRISTOBAL	170095	10.0000	100.79	1,007.90	1,008.00	3/15/2023
AMPL DE SN CRISTOBAL	170097	29.1223	100.79	2,935.24	2,935.00	3/15/2023
UNIF EL CONJURO	170592	44.8750	100.79	4,522.95	4,523.00	6/1/2023
TEPOZAN SEGUNDO	163260	13.5400	100.79	1,364.70	1,365.00	9/3/2028
AMPL. AL TEPOZAN	182730	10.7804	100.79	1,086.56	1,087.00	8/15/2038
VICTORIA 2	217628	16.7307	14.24	238.25	238.00	8/5/2052
VICTORIA 3 FRACC A	217629	459.3262	14.24	6,540.81	6,541.00	8/5/2052
VICTORIA 3 FRACC B	217630	14.1635	14.24	201.69	202.00	8/5/2052
EL REAL	214498	20.0000	28.64	572.80	573.00	10/1/2051
NUEVA ERA	214659	29.7151	28.64	851.04	851.00	10/25/2051
LA REFORMA	218667	135.5786	14.24	1,930.64	1,931.00	12/2/2052
PLATOSA	216290	41.0406	14.24	584.42	584.00	4/29/2052
SAN FRANCISCO	206567	7.7525	100.79	781.37	781.00	1/29/2048
VICTORIA 5	226310	693.4344	6.88	4,770.83	4,771.00	12/5/2055
VICTORIA EUGENIA	211587	36.0864	28.64	1,033.51	1,034.00	6/15/2050
SN FCO I FRACC 1	223953	165.5461	100.79	16,685.39	16,685.00	3/14/2055
SN FCO I FRACC 2	223952	3.3363	100.79	336.27	336.00	3/14/2055
LA CRUZ	211085	8.5121	28.64	243.79	244.00	3/30/2050
CRESTON	213594	9.0000	28.64	257.76	258.00	5/17/2051
ESCALERA FRACC 1	Awaiting Title	2.4573	0.00	0.00		
ESCALERA FRACC 2	Awaiting Title	2.9544	0.00	0.00		
ESCALERA FRACC 3	Awaiting Title	1.7926	0.00	0.00		
ESCALERA FRACC 4	Awaiting Title	1.1399	0.00	0.00		
ESCALERA FRACC 5	Awaiting Title	6.5872	0.00	0.00		
ESCALERA FRACC 6	Awaiting Title	6.0759	0.00	0.00		
ESCALERA FRACC 7		5.7413	0.00	0.00		

	Awaiting Title					
EL REAL 2	228945	561.2590	4.60	2,581.79	2,582.00	2/20/2057
MELISA	217670	69.5670	14.24	990.63	991.00	8/5/2052
LIZETTE	221172	23.3852	14.24	333.01	333.00	12/2/2053
TOTAL		**2,864.11**		**87,455.06**	**87,457.00**	

The concession Unificada Victoria Eugenia contains all of the mineral resources and mineral reserves, most of the mine workings, part of the mine plant, buildings and offices, the San Fermin Mine portal, the Candelaria Mine portal, the Recompensa Mine portal, the Estrella Mine workings, and the El Aguila shaft.

The Veta Dos portal, and some of the mine workings are located on Victoria 2. Victoria 3 Fraccion B contains some of the mine workings.

The tailings dam and storage area are located on Victoria 5 and Victoria 3 Fraccion A. The remainder of the mine plant, buildings and offices are located on Victoria 3 Fraccion A.

Concessions Escalera Fracc 1, 2, 3, 4, 5, 6 and 7 have been staked and are awaiting title from the Mexican government. The concession titles are expected within the following year after which PAS plans to explore and potentially develop the concessions.

6.2. Permits and Agreements

General Mining Office

To the best of the authors knowledge, all of the annual work commitments, payment of duties, and all other requirements to maintain the mining concessions held by Plata have been duly complied with.

Foreign Trade Services Department

On September 19, 2005, Plata was designated by the Ministry of Economy an ALTEX , or high level exporting company, and was registered as such with the Ministry of Economy under Certificate No. 2005/5838. As an ALTEX, Plata is entitled to carry out importing and exporting activities in relation to its operations and to obtain fiscal benefits and refunds related to such activities.

National Registry of Foreign Investment

To the best of the authors knowledge, Plata is in compliance with the quarterly and annual filing requirements of this registry.

Federal Labour Delegation

To the best of the authors knowledge, Plata is in compliance with the requirements of the applicable labour laws of Mexcio, and all registrations, as required, for the Federal Labour Delegation, in the State of Zacatecas, have been filed.

Federal Board of Conciliation and Labour Arbitration

To the best of the authors knowledge, there are no labour lawsuits against Plata.

Real Estate

To the best of the authors knowledge, title to the concessions held by Plata associated with La Colorada have been registered in the Public Registry of Property of Sombrerete, Zacatecas and are free of any liens or encumbrances.

Ministry of Finance

To the best of the authors knowledge, all filings with the Ministry in respect of income and sales taxes have been made on time and as prescribed.

Mexican Social Security Institute (IMSS)

To the best of the authors knowledge, Plata is in compliance with the payment of dues to IMSS in respect of both employer and employee withholdings.

Agreements

To the best of PAS knowledge, the La Colorada property is not subject to any royalties, back-in rights or encumbrances.

General Management of the Federal Registry of Firearms and Explosives (SECRETARIA DE LA DEFENSA NACIONAL (SEDENA)

Plata was granted General Permit (2917-Zacatecas) in 2000 authorizing the purchase, storage and use of explosives subject to Plata continuing to meet permit requirements. This is revalidated on an annual basis. To the best of the authors knowledge, Plata is in compliance with the monthly reporting requirements of this permit.

Federal Bureau of Environmental Protection (Secretaria de Medio Ambiente y Recursos Naturales: SEMARNAT) and National Ecology Institute (Institutio Nacional de Ecología: DIRECCION GENERAL DE ORDENAMIENTO ECOLOGICO E IMPACTO)

Following submission of an environmental impact statement, named the Manifestación de Impacto Ambiental-Modalidad General (EIS) and environmental risk assessment study, named the Estudio de Riesgo Ambiental Modalidad Análisis de Riesgo, the federal environmental authority granted approval (the Dictamen) for new project construction under D.O.O.DGOEIA.- 007244 on November 11, 1999. In October 2000, Plata received authorization by way of a change in use of soils (Cambio de Uso de Suelos) permit to construct a new tailings dam on land not previously impacted by historic mining operations.

National Water Commission (Comisión Nacional del Agua: Conagua)

Mining generates tailings, which are materials considered to be potentially hazardous wastes. Plata filed an application to become a hazardous waste generator in January 1999 and the required permit was received March 26, 2001.

Plata holds a permit (Concesión 03ZAC103761/11EQGE02) dated September 19, 2002, which permits the discharge of waters into the subsurface of the La Colorada property. Pursuant to a new National Waters Law (Ley de Aguas Nacionales), Plata is permitted to make use of waters obtained from the exploitation of a mine without having to apply to the National Water Commission for a permit or authorization.

6.3. Environmental Issues

An EIS and risk assessment was approved by the Mexican federal environmental authority in November of 1999. To the best of the authors knowledge, Plata is currently in compliance with all applicable environmental laws. Known environmental liabilities are associated with mining disturbances. The cost of closure of the La Colorada Mine is discussed in section 25.3.

7.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The closest municipality to the La Colorada property is the city of Chalchihuites, which is 16-km north-west of La Colorada Mine, with a population of approximately 1,000.

The La Colorada Mine site is accessible by road approximately 2 1/2 hours south-east of the city of Durango. The road consists of 120 km of a paved two-lane highway (Highway 45), and 23 km of public, all weather gravel road. Access from Zacatecas takes approximately the same time on similar types of roads. Durango and Zacatecas are serviced by daily flights from México City, other major centers in México and direct flights from some cities in the United States.

The physiography of the region is characterized by wide flat valleys and narrow, relatively low mountains ranges and hills. Topographic relief near the Candelaria, Recompensa and Campaña Mine sites is between 2,100 m and 2,550 m.

The climate is arid to semi-arid and vegetation typically includes mesquite and cactus. The rainy season is from July to September. Table 4 gives the precipitation statistics measured at the local government weather station. Winter temperatures are around freezing during the night. The mine operates throughout the entire year.

Table 4 Chalchihuites Statistics on Rain & Evaporation

Averages from 1962 to 1997 in millimetres.

Month	Max rain in 24 hours	Max rain per month	Evaporation
January	5.77	5.13	135.4
February	4.50	6.37	140.0
March	2.17	2.90	184.4
April	1.56	2.23	192.9
May	5.77	11.81	214.9
June	21.58	65.38	177.2
July	25.30	135.18	147.2
August	27.04	140.91	137.3
September	27.70	93.13	131.6
October	13.34	27.90	141.4
November	7.51	11.42	131.0
December	7.77	16.37	132.4

A long history of silver mining in Zacatecas State has resulted in an adequate infrastructure and an experienced workforce in the region. Two out of five of the largest silver mines in México are located in the area (San Martín, 6500 t/d, 35 km north of La Colorada and Fresnillo, 3800 t/d, 100 km south-east of La Colorada). There is another Zn-Pb-Cu-Ag mine 35 km north of La Colorada, Sabinas, 3,300 t/d. Durango and Zacatecas are the major industrial and supply centers in the area. Both are serviced by air and land routes. All facilities and services are available in these cities to support a mining operation.

The area is widely but not densely populated. The majority of the people engage in subsistence farming of predominately bean and corn crops.

Water sufficient to support the mining operation is available on site and is supplied from an underground source. As permitted by Mexican law, underground water is pumped to surface head tanks and used in the milling process, as well as for domestic services. The tailings dam and storage pond were approved in October of 2000 by the Federal Bureau of Environmental Protection and the construction was completed in June 2003. Power supply contracts for up to 3.5 MW are in place and operating. A new 9.0 MW power line is currently being installed from the town of Sombrerete and project completion is scheduled for the end of 2007 with start-up in January 2008. The contracts are already in place and the additional power in conjunction with the already installed the 3.5 MW power line will be sufficient for the mine and processing plant. The location of the tailings storage area, plant location, site buildings mine portals and mine shaft are shown in Figure 5.

8.0 HISTORY

The production history of the Chalchihuites district began during Pre-Colonial times when natives produced silver and malachite. During the 16th century Spanish colonization, the village of Chalchihuites was founded and intermittent exploitation of the mineral deposits in the area commenced. By the 19th century, the Spanish mines were operating continuously and important silver production was recorded. The War of Independence curtailed production from this and many other silver producing areas between 1910 and 1920.

Recent mining on the La Colorada property began in 1925. The Dorado family mined the La Colorada, Negrillas and Paloma breccia pipes.

In 1929, Candelaria y Canoas S.A. (Candelaria Co.), a subsidiary of Fresnillo S.A., started a 100 tpd flotation plant, processing dumps from the nearby San Rafael mine which was said to have produced ore containing 350 to 400 g/t Ag. The company also exploited the upper levels of the Candelaria mine with the main area of mining being the 252 level. The Candelaria Co. activities were suspended in 1955.

In 1935, a mining company, La Campaña de Industrias Peñoles, began operations on the Campaña breccia pipe, which lasted to the end of World War II.

In 1949, Compañía Minas Victoria Eugenia S.A. de C.V. (Eugenia), began mining activities and operated continuously until December 1993. In 1956, production reached 7,500 tonnes per month (tpm) with an average grade of 500 g/t Ag from various areas of the property. Eugenia exploited the mining properties Colorada, Campaña, Candelaria, Canoas, Dulces Nombres, El Conjuro, San Cristobal and San Fermín. All these properties returned high production grades, including Campaña where 400 g/t Ag was obtained in approximately 250,000 tonnes from the 60 level to surface. In the Colorada mine, breccia pipes reportedly produced lead ore containing between 55% and 60% lead (Pb) in addition to 1,250 to 1,500 g/t Ag.

In 1994, the properties of Eugenia were acquired by Minas La Colorada S.A. de C.V. (Minas) which operated three of the old mines, namely the Candelaria, Recompensa and Campaña. Production was at a rate of about 6,000 tpm up until March 1998.

PAS acquired the La Colorada property from Minas in April 1998, through Plata, its wholly owned subsidiary.

In 2000, development work at La Colorada included surface and underground diamond drilling for reserve definition, the preparation of a bankable feasibility study, negotiation with banks for project financing, independent engineering review, repairs to the existing shaft and rehabilitation of the existing mill to restart operation in 2001. In the fall of 2000 repairs involving shotcrete and steel were made on the existing shaft.

A bankable feasibility study was completed in June 2000 using H.A. Simons Ltd. for mill design, Agra Earth and Environmental Ltd. for tailing design, and Beacon Hill Consultants and R. Barnes Consultant for mine design. An environmental impact study (EIS) was prepared to World Bank standards by Dew Point International, LLC and reviewed by Clifton Associates Ltd.

PAS decided to rehabilitate the existing mill at La Colorada to allow for limited production in 2001. Limited production commenced in January 2001 at approximately 90 tpd, which increased to approximately 120 tpd as of March 2001 and reached a consistent production rate of 150 tpd in June 2001. In January 2002, the mill operating capacity was increased to 200 tpd following the addition of another small ball mill and additional lead flotation capacity. The feed for the mill consisted of underground sulphide ore from the La Colorada property.

In January 2002, PAS prepared the Updated Feasibility Study, which recommended the construction of a 210,000 tonne per year underground mining operation for oxide ore in conjunction with the continued mining of 70,000 tonnes per year of sulphide ore.

Construction of the new oxide mill commenced in July 2002 and produced the first dorè bars in August 2003. The rest of the facilities, including the surface areas and sulphides plant rehabilitation, road upgrades and the first phase of the tailings dam construction were 95% complete by December 31, 2003. Total project construction work, including the second phase of the tailings dam, was completed during 2004. The sulphide mill was expanded from its 2003 rehabilitation phase in 2007 to allow for a zinc flotation circuit.

Production continued during 2003 on sulphide ore. From 2003 to 2005, 408,061 tonnes of oxides with an average grade of 498 g/t Ag, 0.56 g/t gold (Au), and 74,063 tonnes of sulphides with an average grade of 482 g/t Ag, 0.46 g/t Au, 1.04% Pb and 1.45 % Zn were produced. From 2005 to 2007, 580,825 tonnes of oxides were mined with an average grade of 512.5 g/t Ag and 0.589 g/t Au. 100,062 tonnes of sulphide ore was also mined during 2005 to 2007 with average grades of 466 g/t Ag, 0.50 g/t Au, 0.84% Pb and 1.5% Zn. A new flotation circuit was added in 2007 to the sulphide circuit to recover zinc; therefore, only 79,504 tonnes of the sulphide ore was processed recovering Zn in a zinc concentrate.

The new information about ground conditions in the mine stopes and the ground water inflows led to changes of the mining assumptions, which were reflected in the last Technical Report. The mineral resource and mineral reserve estimates contained in this Technical Report replace the previous estimates.

9.0 GEOLOGICAL SETTING

9.1. Regional and Local Geology

The La Colorada property is located on the eastern flanks of the Sierra Madre Occidental at the contact between the Lower Volcanic Complex and the Upper Volcanic Supergroup. The La Colorada property lays 16km southeast of Chalchihuites and 30km south-southwest of Sombrerete, two mining camps with significant silver and base metal production from veins and associated skarn deposits.

The oldest rocks exposed in the mine area are Cretaceous carbonates and calcareous clastic rocks of the Cuesta del Cura and Indidura Formations (Figure 6). Overlying the calcareous rocks is a conglomerate unit containing clasts derived mostly from the subadjacent sedimentary rocks. In the Chalchihuites district this unit is called the Ahuichila Formation and is of Early Tertiary age.

Most of the outcrop in the mine area is represented by intermediate to felsic volcanic rocks of the regional Lower Volcanic Complex. This unit is identified as a trachyte in older mine data, although recent petrogrophy indicate that it is actually an altered dacite. There are several subgroups within this unit, including plagioclase porphyry, crystal to crystal-lapilli tuffs, and volcanic breccias. Generally these sub-units do not form mappable units.

The stratigraphically highest rocks in the mine area are felsic tuffs correlated with the Upper Volcanic Sequence. These tuffs unconformably overlie the trachyte along the southern property boundary and are distinctly maroon coloured and show varying degrees of welding.

Thirteen breccia pipes have been mapped at surface or in underground workings. All of the pipes are located along or to the south of the No Conocida Poniente (NCP/NC2) vein complex. The pipes are round to ovoid in shape, up to 100 metres in diameter, and can extend vertically more than 400 metres below the surface. The breccias contain clasts of limestone and trachyte (often mineralized) in an altered trachyte matrix. The ratio of limestone to trachyte clasts varies from pipe to pipe. Clasts of vein material have been found in the breccias suggesting that they postdate the vein emplacement.

CHRONOLOGY OF GEOLOGICAL EVENTS AT LA COLORADA

9.2. Structural Geology

The structures present at La Colorada represent a deformational sequence comprising at least three significant events:

- Laramide folding and faulting;
- Post-Laramide, east to northeast-striking faults; and
- Regional tilting events.

Regional deformation during the Laramide Orogeny is expressed by the widespread development of folds and contractional faults within the Cretaceous stratified sequence. These units show an abundance of folds and faults cutting shallowly to steeply across bedding where the rocks are exposed in the western portion of the La Colorada property and in the underground workings.

East to northeast striking faults form the dominant structures in the project area and play a strong role in local mineralization. Most of these faults dip moderately to steeply to the south and juxtapose younger hangingwall strata against older footwall rocks. Evidence suggests down-dip motion on these faults; however, most of the faults have been reactivated at some point so the movement direction during the initial formation is uncertain. Stratigraphic contacts are displaced from ten to over a hundred metres lower on down dropped blocks.

The trachyte unit displays an eastward tilting that may reflect displacements on regional, orogenparallel structures outside of the project area. This tilting probably reflects the final episode of deformation.

Dr. Peter Lewis, structural geology consultant, has proposed a structural model for La Colorada that suggests mineralization and alteration occurred in a tectonic regime dominated by gravitational forces and low horizontal stresses (Lewis, 1998). In this regime, the pre-existing steeply dipping structures were favourably orientated for re-activation and subsequent emplacement of mineralizing hydrothermal fluids. The dominantly eastern strike of the veins indicates slightly greater extension in a northerly direction. The north and north-easterly dipping faults accommodated mostly transverse movement associated with the dilation of the steeply dipping, easterly striking structures.

10.0 DEPOSIT TYPES

La Colorada represents a typical epithermal silver/gold deposit, with a transition in the lower reaches of the deposit to a more base metal predominant system. The geological model used for exploration as well as the mineral resource estimation is that of an epithermal vein deposit. There are indications of what might be skarn style mineralization in the deepest holes on the property. A local analogy of this type of deposit would be the San Martin Mine, where earlier in the mine life epithermal veins were mined and now the mine production comes from skarn mineralization hosted by the same limestone unit found in La Colorada Mine.

11.0 MINERALIZATION

There are 4 dominant styles of mineralization at La Colorada:

1 breccia pipes;

2 vein-hosted mineralization;

3 replacement mantos within limestone; and

4 deeper seated transitional mineralization (transition zone).

Mineralization in the breccia pipes generally has lower silver values and elevated base metal values. The core of the Campaña Breccia was bulk mined in previous years with reported grades of 80 g/t Ag and 5% combined Pb/Zn. Mineralization is associated with intense silicification and occurs as disseminated galena and sphalerite with minor chalcopyrite and bornite. Sulphides are found in the clasts and the matrix.

Most mineralized veins on the property strike east to northeast and dip moderately to steeply to the south (Figure 7). Veins occur in the trachyte and limestone units and cut across the bedding and contacts with little change in the width or grades of the vein. Mineralized widths in the veins are generally less than 2 metres, but may be wider if there is a halo of replacement or brecciated material. The No Conocida Poniente (NCP) Corridor strikes east west and dips moderately to the south, with average widths up to 15 metres, but most of the economic mineralization is located in quartz veins which are on average 1 to 2 m wide. In some cases the vein fillings consist of quartz, calcite, and locally barite and rhodochorosite. Where the veins are unoxidized, galena, sphalerite, pyrite, native silver and silver sulfosalts are present. The major mineralized veins, including the NCP Corridor, are strongly brecciated and locally oxidized, obscuring original textural features. Less deformed veins show mineralogical layering, crystal-lined open vugs, and hydrofracture vein breccias, indicating typical multi-stage growth**.**

The depth to the surface and the permeability of the mineralized zone control the level of oxidation in the veins. These factors result in an uneven, but generally well-defined redox boundary.

Manto style mineralization is found near vein contacts where the primary host rock is limestone. This style of mineralization was mined at Recompensa, but can also be seen in areas of the Candelaria Mine. At Recompensa the mantos appear to be controlled by thrust faulting adjacent to the veins and can form bodies up to 6 metres wide. Most commonly, they occur in the footwall north of the steeply dipping vein, but depending on the orientation of the fault they can occur in the footwall, the hangingwall, or both. The mineralogy of the mantos is characterized by galena and sphalerite with minor pyrite and chalcopyrite. Gangue minerals are quartz, rhodochorosite, pyrolusite and other manganese oxides.

The deep seated transition mineralization, also known as NC2E Deep, consists of both vein type mineralization and more diffuse stockwork and breccia zones. Peter Lewis (Lewis Geoscience, 1998) has suggested that there are 7 distinct zones within the transitional zone, and these can be sub-grouped into 3 main categories:

1. vein mineralization, including the down dip extension of NC2E and veins in the hangingwall and footwall of NC2E;
2. a peripheral stockwork vein zone that envelopes the NC2E structure; and
3. sub-horizontal mantos-like stockwork zones in the NC2E hangingwall.

At the time of the 1998 Lewis Geoscience report, due to limited drilling access, there was only 7 holes that intersected all or part of the sub-groups in the transition zone. During 2007 a new drill campaign was started to define this deeper mineralization. This work is in progress and is not yet reported.

11.1. Ore Zones

<u>Candelaria System</u>

NCP and NCP Corridor - Average orientation 75/60S 60deg dip. The Corridor consists of the NCP footwall and NCP hangingwall structures. There are currently 3 hangingwall structures defined named HW1, HW2 and Split. These zones are characterized by a broad mineralized shear within limestone containing one or more quartz veins parallel to the orientation of the shear. The majority of the silver mineralization is found in the quartz veins which in the NCP footwall vein are on average 2.9 metres wide and in the NCP hangingwall vein HW1 are on average 2.4 metres wide. HW2 vein is on average 2.0m wide and Split is 2.2m wide. Mining is in progress on various sublevels down to the 438 level.

NC2 - Average orientation 45/70S 60 deg dip. NC2 is a narrow (one-to-two metre) sulphide vein that contains an important part of the current sulphide resources. It has a strike length of over 700 metres and is open to the east where there is a wedge of inferred material below the east mine fault. NC2 is developed down to the 390 level and has been drilled to below the 495 level where inferred resources have been estimated.

NC2W - Average orientation 35/65S 60deg dip. NC2W is, in the opinion of Dr. Michael Steinmann, P.Geo., the faulted, western extension of NC2E. The western portion of NC2W is oxidized and averages 2.1 metres wide. The eastern portion is sulphide and averages 1.1 metres wide. This structure holds oxide reserves between the 150 and 220 levels and inferred sulphide resources between 220 and 270 levels.

4235 - Average orientation 90/75N dip 65deg. 4235 is a narrow (approximately one metre wide) vein which occurs in the hangingwall of the NCP and NC2 vein systems counter to the orientation of these major veins. It has a strike length of approximately 140 metres and has been exposed by development on the 295 level and by drilling above and below that level. The western half of 4235 is sulphide and the eastern half is oxide. Only resources have been estimated in this structure.

Inversa - this vein is a smaller version of the counter vein orientation V4325. Vein widths are around 2.3 metres. This vein has been defined by mining development on the 335 and 355 levels and has been partially mined in 2007.

Recompensa / Estrella System

The Estrella system was formerly referred to as the Amolillo system in the document titled La Colorada Mine Project, Zacatecas, Technical Report, March 17, 2006 . The Amolillo system will be referred to as the Estrella system within this Technical Report.

Recompensa - Average orientation 90/80N dip 75deg. Recompensa is a combination of vein and manto mineralization located more than one kilometre northwest of the NC2 and NCP vein complex. The vein mineralization is narrow (less than one metre and averages 1.8 metres for the economic zone). Recompensa contains a minor amount of oxide but mostly sulphide material.

Estrella - Average orientation 45/70S dip 59deg. Estralla is an oxide vein located 500 metres north of the NC2 and NCP vein complex and to the east (approximately along strike) of the Recompensa vein with an average width of 2.2 metres. The vein lies mostly within the trachite host rock and the limestone at depth.

12.0 EXPLORATION

The bulk of PAS exploration effort has been conducted through diamond drilling (surface and underground) and underground drifting on the veins and mineralized zones. Table 4, set out below, summarizes the drilling conducted by PAS from 1998 to September 2007 and by the previous owner in 1997. All drilling from 1998 to 2007 has been performed under the supervision of the PAS geology department. In 2007, approximately 7,056 metres of drilling was completed by REDRILLMEX REDRILSA Mexico S.A. D.E. CU. (Peru, Lima). All other drilling was preformed by PAS employees.

Table 5 List of Drilling Campaigns by Year:

	Surface Drilling		Underground Drilling		Total Drilling	
Year	**# of Holes**	**Metres**	**# of Holes**	**Metres**	**# of Holes**	**Metres**
1997	6	1,026	8	1,477	14	2,503
1998	28	8,026	28	7,853	56	15,879
1999	11	2,650	49	5,104	60	7,754
2000			42	5,228	42	5,228
2002	4	963			4	963
2005	17	2,380			17	2,380
2006	46	7,446	20	1,437	66	8,883
2007 (Sep)	33	4,608	61	5,056	94	9,664
Total	**145**	**27,099**	**208**	**26,155**	**353**	**53,254**

Underground drifting along the mineralized structures is the second method of exploration. By September 2007, approximately 18,550 metres of horizontal and ramp development was done in NC2W, NC2E, 4235, and San Fermin areas. The drifting allowed detailed mapping and structural interpretation of the ore zones, as well as providing key grade information. While some of the underground development is shown in Figure 8, the total mine development is too extensive to represent legibly in a simple plan view.

Drifting samples are taken in 3 metre intervals within the first sill drift. After the initial sill has been established, samples are expanded to every 5 metres on subsequent cuts above / below the original sill drift. In 2008, 1,750 metres of underground ore development are planned within Candelaria and Estrella Mines.

In 2008 another 18,232 metres of exploration and definition drilling are planned. 9,865 meters of underground exploration are planned to be drilled within the sulphide section of Candelaria Mine and 4,370 metres of underground exploration and definition drilling are planned for the oxide zone of the Candelaria Mine. A total of 3,997 metres of surface drilling is planned for 2008 in the San Fermin, Candelaria oxides and the La Estrella zones. The diamond drill patterns are variable and are dependent on the structural continuity and regularity of the vein system.

Dr. Peter Lewis did two structural studies at La Colorada by drill core analysis. The first one was done in September 1998, the objectives being a general evaluation of structural events and their relationship to vein emplacement; determination of controls on both grade and thickness of vein mineralization and the development of conceptual specific exploration targets. The authors of this Technical Report believe these objectives were successfully met. The second study (2000) targeted specific structural questions within the mine and structural controls on the oxidation boundary.

13.0 DRILLING

From 1997 to September 2007, Plata drilled 145 surface holes and 208 underground holes. Surface drilling was done with NQ sized core and underground was done with BQ sized core, except for the drilling in the NCP Corridor in 2000, which was done with HQ sized core in an attempt to improve recovery. Contractors under the direct supervision of Plata geologists performed all drilling for both surface and underground.

Prior to PAS involvement in the project, previous operators had drilled 131 holes for a total of 8,665 metres. Drill hole locations for these holes were scaled from plan maps. Assay information was taken from drill logs. These holes were not used in the resource calculation, with the exception of 4 holes, where the original core was found and assayed by PAS.

20

The holes generally range in length from 100 to 300 metres with dips from +45 degrees to -90 degrees. Standard logging and sampling processes were used to record information from the holes drilled by PAS. Intervals sampled were cut with a diamond saw and the entire remaining core is stored on-site. Hole collars were surveyed by a total station survey instrument.

La Colorada contracts out some of the exploration drilling to REDRILLMEX REDRILSA Mexico S.A. D.E. CU. (Peru, Lima). In 2007 the contractor was responsible for drilling approximately 7,056 metres. The contractor was under the direct supervision of Plata geologists. All other drilling was completed by Plata employees.

A listing of the La Colorada drill hole collars is given in Table 6:

Table 6 La Colorada Drill Hole Collars

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
BCH-1	5381.9	5409.9	2538.3	-55	230	286	5	3.67	264	0.43	0.22	0.05
BCH-2	5381.9	5409.4	2538.3	-55	190	N/A	N/A	N/A	N/A	N/A	N/A	N/A
BCH-3	5492.5	5127.5	2519.7	-50	260	346	23.57	10.5	14	0.1	1.34	0.09
BCH-4	5334	5229.4	2547.4	-55	220	143.5	23	14	104	0.37	5.38	1.24
BCH-5	5469.4	5045.7	2543.8	-55	270	N/A	N/A	N/A	N/A	N/A	N/A	N/A
BCH-6	5469.4	5145.7	2543.8	-65	75	178	23	21.24	84	0.18	0.89	0.21
BCH-7	5242	5220	2562	-55	210	335	N/A	N/A	N/A	N/A	N/A	N/A
BCH-8	5242	5220	2562	-50	240	316	68.51	62	22	0.11	0.67	0.42
BCH-9	5229.4	2547.4	2547.4	-65	210	N/A	N/A	N/A	N/A	N/A	N/A	N/A
BCH-9A	5334	5229.4	2547.4	-65	225	186.5	12	5	30	0.27	0.44	0.29
BH99-2	5151	3988.3	2400.2	-90	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
CAM-01	4963.8	4969	2197	-49	314	403.86	0.71	0.65	318	0.17	2.82	6.02
CAM-02	4963.8	4969	2197	-30	299	312.15	0.9	0.69	205	0.4	3.31	12.8
CAM-03	4961.8	5057.6	2185	-52	346	457.5	2.8	2.42	85	0.14	2.68	14.84
CAM-04	4961.8	5057.6	2185	-63	319	550.3	8.62	7.46	31	0.11	2.61	3.49
CAM-05	4963.8	4969	2197	-60	308	545.59	0.18	0.16	4	0.14	27.4	14.9
CAM-06	5004	4834.4	2216.7	-70	175	495	64.7	41.59	13	0.07	0.16	2.2
CAM-07	4963.8	4969	2197	-34	289	307.83	0.3	0.3	617	0.24	4.47	9.67
CAM-08	5067	4784	2217.8	-82	111	240.79	8.88	5.34	5	0.06	0.19	1.03
CAM-09	4961.8	5057.6	2185	-45	346	393.8	2.79	2.62	417	0.07	0.89	1.74
CAM-10	4961.8	5057.6	2185	-75	346	432.81	10.23	7.72	32	0.08	1.02	4.85
CAM-11	4961.8	5057.6	2185	-56	322	429.77	1.87	1.76	97	0.07	5.85	11.24
CAM-12	4951.4	5107	2182	-45	75	435.25	4.65	4.65	36	0.12	0.5	1.63
LIB-01	3472.4	4733.3	2401.5	-70	344	250.75	N/A	N/A	N/A	N/A	N/A	N/A
LIB-02	3593.7	4753	2405.7	-60	325	210.85	2.3	1.63	178	0.38	0.03	0.04
LIB-03	3593.7	4753	2405.7	-85	325	250.7	2.8	2.14	251	0.08	0.25	0.18
LIB-04	3472.4	4733.3	2401.5	-86	345	269.4	5.8	2.45	246	0.57	0.22	0.32
MW98-1*	4984.8	3549.2	2359.4	-90	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
MW98-2*	5380.1	3623.6	2362	-90	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
MW98-3*	4448.6	4195.7	2424.4	-90	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
MW98-4*	4910.9	3869.9	2368.7	-90	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
MW-99-1*	5097.1	3213.5	2343.5	-90	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
PIC-01	4220.9	4937.1	2187.1	-90	0	76.15	N/A	N/A	N/A	N/A	N/A	N/A
PIC-02	4374.7	5038.4	2186.1	-50	156	237.85	0.35	0.33	1,105	0.24	1.1	3.67
PIC-03	4458	4981.7	2186	-66	346	185.15	2.6	2.13	805	0.19	1.1	0.92
PIC-04	4612.1	5020.4	2184.3	-70	310	200.25	0.8	0.57	430	0	0.2	0.3
PIC-05	4612.1	5020.4	2184.3	-50	5	149.8	2.62	2.27	1,355	0.13	1.46	3.31
PIC-06	4822.9	5229.1	2186.1	-81	275	185.35	3	2.45	466	0.03	0.81	2.41
PIC-07	4822.9	5229.1	2186.1	-67	306	121.7	1.95	1.69	585	0.09	0.98	0.88
PIC-08	4966.5	5302	2186.3	-67	295	183.5	1	0.77	128	0.31	1.39	0.67
PIC-09	4845.7	5217.1	2186.2	-90	0	242.25	2.1	1.82	1,985	0.19	15.19	16.22
PIC-10	4845.4	5216.8	2186.1	-45	315	201.2	1	0.96	268	0.08	0.17	0.29

PIC-11	4845.4	5217.4	2186.1	-75	315	224.1	2.7	1.74	734	2.3	1.57	4.44
PIC-12	4845.4	5216.8	2186.1	-60	350	181.4	2	1.73	786	0.2	1.93	12.11
PIC-13	4968.5	5301.6	2186.4	-50	349	181.4	1.2	1.09	344	0.73	0.22	0.88

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
PIC-14	4968.5	5301.6	2186.4	-40	295	172.3	2.7	2.68	469	0.15	0.47	1.94
PIC-15	4747.2	5282.3	2184.5	-1	315	182.9	0.8	0.69	540	0.36	1.53	1.21
PIC-16	4848.3	5219.2	2186.1	-75	55	320.1	2.5	0.86	186	0.16	5.27	2.06
PIC-17	4218.4	4929.7	2188.9	0	165	301.8	1.3	1.13	4,232	11.65	2.88	3.91
PIC-19	4610.5	5019.9	2184.3	-50	310	134.15	1.5	1.48	179	0.03	0.69	1.17
PIC-20	4222.2	4922.2	2185.5	-51	342	165.7	5.25	4.87	1,039	0.02	1.26	1.22
PIC-21	4823.2	5336.1	2185.6	-90	0	288.04	2.36	1.67	3,706	2.89	8.95	19.14
PIC-22	4917	5379	2186.3	-90	0	246.89	0.9	0.45	1,326	1.1	8.44	35.6
PIC-23	4658	5022.3	2182.7	-65	5	234.69	2.31	1.89	382	0.39	0.69	1.98
PIC-24	4600.8	4968.5	2181.6	-60	270	195.68	2.21	1.81	963	0.07	4.35	11.14
PIC-25	4658	5022.3	2182.7	-90	0	499.87	3.3	1.89	207	0.09	0.37	1.62
PIC-26	4823.2	5336.1	2185.6	-60	315	219.46	0.51	0.37	112	0.07	0.44	0.57
PIC-27	4917	5379	2186.3	-60	315	242.31	2.1	1.54	320	0.32	0.13	0.29
PIC-28	4254.7	4812.2	2188.2	0	143	176.78	N/A	N/A	N/A	N/A	N/A	N/A
PIC-29	4254	4817.5	2189.4	-33	34	287.42	2.63	1.19	614	1.88	0.75	1.09
PIC-30	4254.7	4812.2	2189.2	10	154	208.48	2.22	2.12	820	2.5	0.62	1.59
PIC-31	4230	4911	2185.5	-40	42	184.4	N/A	N/A	N/A	N/A	N/A	N/A
PIC-32	4225.5	4911.8	2185	-70	328	201.17	0.6	0.4	276	0.21	0.84	3.68
PIC-33	4123.8	4903.7	2187.1	-77	303	167.64	5	3.54	346	0.47	0.97	1.33
PIC-34	3998.8	4899.8	2187.9	-65	0	105.16	1.28	1.05	1,289	0.25	3.09	4.44
PIC-35	3998.9	4897.7	2187.9	-90	0	145.08	13.38	6.7	589	0.23	0.29	0.48
PIC-36	4048	4910.5	2185	-62	0	100.58	3	2.46	349	0.14	0.95	2.71
PIC-37	4044.1	4900.2	2187.1	-86	0	158.49	4.92	3.8	654	0.29	0.54	1.65
PIC-38	4043.5	4896.2	2188.7	0	176	76.2	0.26	0.21	184	0.13	0.05	0.08
PIC-39	4123	4904.1	2187.1	-51	337	129.54	4.68	4.24	543	0.15	0.25	0.94
PIC-40	4144.4	4916.1	2187.1	-71	0	156.67	7.06	5.4	426	0.24	0.83	1.26
PIC-41	4145.4	4916	2187.1	-41	0	124.97	4.57	4.5	1,258	0.37	1.45	1.13
PIC-42	4226.7	4915.7	2188.5	-46	173	91.44	2.13	0.7	649	0.27	0.23	0.39
PIC-43	4312.5	5029.5	2186.1	-16	168	201.16	0.45	0.42	385	0.1	0.99	2.47
PIC-44	4262	4705	2187.5	32	343	161.24	0.36	0.27	45	0.04	1.81	1.5
PIC-45	3829.2	4927.6	2189.7	0	348	54.86	0.68	0.59	286	0.15	0.22	1.8
PIC-46	4047	4918.3	2187.8	-7	0	70.1	0.9	0.71	796	0.2	0.56	0.84
PIC-47	4043.7	4917.7	2187.8	-7	327	118.87	3.05	2.76	357	0.2	0.77	3.58
PIC-48	4144.8	4916.6	2187.8	-5	0	137.16	8.64	5.55	658	1.77	0.42	0.41
PIC-49	4192.8	4917	2187.8	-5	3	150.88	6.97	4	650	0.11	0.89	0.38
PIC-50	3952.3	4875.4	2188.1	-56	0	112.78	2.63	2.01	385	0.18	0.29	0.32
PIC-51	3952.1	4876.3	2188.1	-90	0	195.07	10.65	6.85	608	0.22	0.44	1.33
PIC-52	3952.3	4875.4	2190.8	11	0	141.73	4.39	2.08	286	0.12	0.34	0.61
PIC-53	4065.4	4849.1	2187.7	-90	0	309.37	0.83	0.5	227	0.51	1.1	10
PIC-54	4142.7	4916.7	2189.6	-6	335	135.65	9.91	7.59	642	1.11	0.71	1.32
PIC-55	4047.4	4918.2	2187.8	-7	2	13.11	N/A	N/A	N/A	N/A	N/A	N/A
PIC-56	4047.2	4818.2	2190.1	11	0	131.06	2.19	2.08	483	0.2	0.17	0.75
PIC-57	4142.1	4916.7	2188.8	-6	309	135.33	2.98	2.28	252	0.07	0.47	2.23

PIC-58	3898.7	4854.2	2188.7	-35	0	126.49	1.41	1.39	634	0.25	0.7	0.76
PIC-59	3898.7	4854.2	2188.7	-70	0	155.45	2.96	2.72	721	0.17	0.84	2

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
PIC-60	4341	4833	2324	-37	327	135.94	1.17	0.44	244	0.93	0.06	0.75
PIC-61	4339	4831	2324	-48	272	73.15	N/A	N/A	N/A	N/A	N/A	N/A
PIC-62	4145.8	4915.7	2187.7	-57	2.9	128.35	7.24	4.94	957	0.4	1.01	2.48
PIC-63	4145.8	4916.6	2188.1	-21	4.8	108.85	11.42	10.17	641	0.2	0.32	0.58
PIC-64	4192.8	4917	2188.5	-24	3.3	115.33	6	3.86	535	0.02	0.18	0.31
PIC-65	4122.3	4902.9	2187.1	-65	327	201.17	2.29	1.62	320	0.2	0.37	0.73
PIC-66	4122.8	4904	2188.1	-29	343	112.35	7.5	4.3	704	0.33	0.52	2.57
PIC-67	4045	4917.8	2187.1	-83	11	144	13.16	6.8	1,343	0.77	1.17	2.44
PIC-68	4046.7	4918.2	2189.1	-37	2	80.2	2.6	2.44	504	0.18	0.41	1.74
PIC-69	4046.7	4918.2	2187	9	1	97.4	3	2.82	479	0.12	0.2	0.61
PIC-70	3897.6	4855.3	2189	-15	0	110.1	1.8	1.77	817	0.31	0.03	0.61
PIC-71	3897.6	4855.3	2189	-90	0	147.4	1.43	1.1	1,017	0.38	2.01	6.43
PIC-72	3951.9	4876.4	2187.9	-75	0	153	1.8	1.47	113	0.06	0.06	1.06
PIC-73	3951.1	4876.2	2189.1	-17	358	94.4	1.1	1.1	1,244	0.84	1.93	4.52
PIC-74	3951.1	4876.2	2190.7	7	358	117.9	3.8	3.5	393	0.1	0.21	0.49
PIC-75	4007.8	4909.7	2189.7	12	354	98.4	5.2	4.97	1,250	0.11	0.31	3.13
PIF-01	3750	5084.5	2364.3	0	225	48.76	1.63	1.15	599	0.38	0.49	2.88
PIF-02	3691.8	5087.6	2356	-33	187	94.48	0.57	0.49	7	0.07	0.02	0.15
PIP-01	4130.3	4620.2	2377	0	135	26.58	N/A	N/A	N/A	N/A	N/A	N/A
PIP-02	3781.4	5087.1	2370	0	158	32.67	N/A	N/A	N/A	N/A	N/A	N/A
PIR-01	3021.6	5836.8	2395	-70	207	76.4	N/A	N/A	N/A	N/A	N/A	N/A
PIR-02	3317.3	5843.9	2423	-77	208	220.35	2.4	0.82	1,022	1.37	44.41	9.19
PIR-03	3026.1	5834.1	2395	-80	135	188.97	1.2	0.54	1,083	0.66	5.27	10.06
PIR-04	3317.3	5843.9	2423	-53	198	146.3	0.77	0.52	606	0.29	4.67	1.5
PIR-05	3317.3	5843.9	2423	-65	157	219.5	2	1.02	549	0.53	1.18	3.41
PIR-06	3021.6	5836.8	2395	-70	225	188.97	0.82	0.25	180	0.17	1.1	1.87
PIR-07	3007.5	5787.5	2398	48	317	47.24	0.71	0.61	431	1.2	0.14	0.16
PIR-08	3008.5	5783.5	2396.5	0	207	24.38	N/A	N/A	N/A	N/A	N/A	N/A
PIR-09	3029	5793	2397	50	0	48.77	N/A	N/A	N/A	N/A	N/A	N/A
PIR-10	3029	5789	2396	0	180	21.34	N/A	N/A	N/A	N/A	N/A	N/A
PIR-11	3050	5799.7	2397	50	0	52.73	N/A	N/A	N/A	N/A	N/A	N/A
PIR-12	3050	5794.4	2396	0	180	25.9	N/A	N/A	N/A	N/A	N/A	N/A
PIR-13	3161.5	5814.5	2389.5	0	200	40.53	N/A	N/A	N/A	N/A	N/A	N/A
PIR-14	3161.5	5814.5	2389.5	0	163	26.82	N/A	N/A	N/A	N/A	N/A	N/A
PIR-15	3201	5786.7	2381	0	0	15.24	N/A	N/A	N/A	N/A	N/A	N/A
PIR-16	3131	5809	2400	0	180	32	N/A	N/A	N/A	N/A	N/A	N/A
PIR-17	3371	5764	2432.5	0	212	50.29	N/A	N/A	N/A	N/A	N/A	N/A
PIR-18	3371	5764	2430.8	-59	222	65.84	0.38	0.35	165	0	0.09	0.17
PIR-19	3400	5759.5	2438.6	0	0	79.24	N/A	N/A	N/A	N/A	N/A	N/A
PIR-20	3007.5	5787.5	2397	0	317	60.19	1.3	1.1	198	0.15	0.57	0.25
PS-02	3318.2	5600.2	2532.2	-57	345	145.7	N/A	N/A	N/A	N/A	N/A	N/A
PS-07	4081.2	5649.2	2490.2	-70	320	220.2	4	0.94	757	0.62	0.31	0.1
PS-11	3479.2	5426.2	2518.2	-51	0	112.3	N/A	N/A	N/A	N/A	N/A	N/A

PS-12	3297.2	5304.2	2502.2	-53	0	276.5	4.8	3.93	171	0.1	1.12	2.22
PS-13	3432.2	5095.2	2496.2	-50	357	140	1.8	1.38	30	0	0.05	0.06

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
PS-14	4147.2	5363.2	2431.2	-60	144	150.9	2.5	1.92	26	0	0.05	0.17
PS-15	3344.2	4990.2	2446.2	-64	0	229.15	0.74	0.67	170	0.38	0.65	1.9
PS-16	3778.2	4969.2	2465.2	-66	0	N/A	N/A	N/A	N/A	N/A	N/A	N/A
PS-16A	3811.2	4991.2	2470.2	-72	0	265.96	1.26	1.03	1,919	0.69	14.09	4.1
PS-18	3727.2	4713.2	2427.2	-61	357	381.75	0.8	0.67	103	0	0.23	0.48
PS-19	5171.2	5600.2	2541.2	-70	330	263.6	N/A	N/A	N/A	N/A	N/A	N/A
PS-20	4159.2	4658.2	2495.2	-68	307	192.6	5.67	4.34	159	0.23	0.13	0.01
PS-22	5052.2	5509.2	2512.2	-73	322	268.4	N/A	N/A	N/A	N/A	N/A	N/A
PS-27	2892.2	5877.2	2488.2	-68	165	220.6	0.1	0.1	633		1.51	2.01
PS-30	4040.2	5012.2	2410.2	-67	0	195.2	3	2.6	237	0.15	0.14	0.2
PS-32	4149.2	5366.2	2432.2	-59	346	135.3	N/A	N/A	N/A	N/A	N/A	N/A
PS-35	5277.2	5410.2	2562	-71	345	347.21	4.8	4.35	69	0.04	0.06	0.56
PS-36	4899.2	5334.2	2497.2	-50	330	275.9	N/A	N/A	N/A	N/A	N/A	N/A
PS98-01	3791.8	4806	2433.2	-50	0	329.18	0.36	0.34	519	0.07	22.4	8.1
PS98-02	3713	4835.5	2423.2	-50	1	299.31	1.23	0.98	37	0.13	0.22	0.01
PS98-03	3601.9	4958.8	2463.2	-60	0	237.74	1.2	1.09	632	0.14	0.89	0.89
PS98-04	4278.5	4470.6	2481.2	-60	315	406.6	1.83	1.66	951	4.1	0.41	0.84
PS98-05	4311.1	4548.7	2496.2	-55	315	347.47	2.23	2.1	980	0.05	0.26	0.13
PS98-06	4346.8	4510.4	2487.2	-70	315	604.11	1.13	1.06	334	0.07	0.72	7.94
PS98-07	3407.4	6017.5	2538.2	-53	180	335.28	0.86	0.69	377	0.14	21.9	1.55
PS98-08	4269.2	4389.2	2443.2	-60	315	458.11	1.29	0.65	638	0.07	0.43	1.15
PS98-09	4602.5	4961.4	2480.2	-90	0	537.36	N/A	N/A	N/A	N/A	N/A	N/A
PS98-10	4190.2	4613.7	2494	-60	315	256.33	2.04	1.77	444	0.2	0.31	0.19
PS98-11	3152.2	6051.1	2563.2	-50	180	393.19	0.46	0.26	739		5.02	12.4
PS98-12	3000	5795	2484	-90	0	137.16	3.42	3	2,883	1.54	0.73	2.19
PS98-13	4142.8	5588.3	2456.2	-55	315	160.32	6.91	6.6	341	0.21	0.36	1.83
PS98-14	4070.9	5526.2	2479.2	-50	315	227.99	N/A	N/A	N/A	N/A	N/A	N/A
PS98-15	4168.3	5682.3	2459.2	-50	315	172.21	N/A	N/A	N/A	N/A	N/A	N/A
PS98-16	4209.2	5691.2	2480.2	-50	0	200.86	3.87	2.1	74	0	0.08	0.07
PS99-01	3959.4	4674.5	2397.7	-15	133	152.4	N/A	N/A	N/A	N/A	N/A	N/A
VN-01	4585.3	5066	2185.1	0	315	27.43	1	0.87	65	0.08	0.03	0.08
VN-02	4538.1	5078.7	2186.1	20	135	45.72	2.22	2.07	256	0.45	1.25	2.01
VN-03	4519.5	5056.4	2185.6	21	120	65.1	0.54	0.5	42	0.07	0.07	0.71
VN-04	4519.5	5056.4	2184.1	-30	90	48.46	1.66	1.56	811	1.86	1.64	0.97
SSF-01-05	4162.7	5069.2	2430.2	-67	280	171.85	2.02	1.89	564	0.4	0.64	0.33
SSF-02-05	4163.1	5069.2	2430.2	-72	280	182.35	N/A	N/A	N/A	N/A	N/A	N/A
SSF-03-05	4161.9	5069	2430.1	-76	18	146.4	0.6	0.46	159	0.73	0.25	0.22
SSF-04-05	4221.4	5102.6	2432.4	-66	337	105.3	9.7	7.95	347	0.35	0.71	0.38
SSF-05-05	4139.7	5118.2	2410.8	-51	356	72.1	N/A	N/A	N/A	N/A	N/A	N/A
SSF-06-05	3750.9	5084.9	2487.5	-50	355	103.35	3.3	3	75	0.07	0.2	1.49
SSF-07-05	3751	5084	2487.4	-80	355	119.55	1.45	1.11	119	0.12	0.58	2.81
SSF-08-05	3648.7	5076.3	2495.7	-59	357	102.6	2.05	1.82	39	0.01	0.12	0.21
SSF-09-05	3648.7	5075.6	2495.5	-80	357	127.35	N/A	N/A	N/A	N/A	N/A	N/A

SSF-10-05	3553	5107.8	2512.6	-78	12	108.25	1.2	0.9	185	0.08	0.88	0.83
SSF-11-05	4347.3	5016.7	2499.7	-62	349	225.5	4.1	3.55	108	0.2	0.44	0.33

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
SSF-12-05	4347.3	5017.2	2499.7	-52	355	228.6	1.8	1.75	238	0.39	0.34	0.64
SSF-13-05	4384.3	5140.8	2459.1	-46	328	104.6	2.32	2.3	632	1.87	0.43	0.31
SSF-14-05	4233.6	5113.9	2432.2	-42	18	100.05	3.65	3.64	98	0.04	0.16	0.1
SSF-15-05	4380.3	5020.1	2502.7	-66	360	271.4	1.2	1.04	75	0.12	0.6	0.33
SSF-17-05	4453.9	5173.6	2478.5	-32	334	129.8	1.01	0.99	649	0.7	0.31	0.86
SSF-18-05	4504.1	5243.9	2477.6	-34	13	80.5	N/A	N/A	N/A	N/A	N/A	N/A
SSF-18-05	4504.1	5243.9	2477.6	-34	13	80.5	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-01-06	4212.4	4753.0	2253.9	0	132	129.45	6.00	5.20	156	1.76	0.31	0.00
DDH-S-02-06	5740.9	2437.5	2493.0	-38	208	150.00	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-03-06	2439.1	5743.5	2492.7	-51	151	200.75	2.05	1.45	334	0.23	1.42	2.65
DDH-U-04-06	4212.3	4753.1	2253.7	-11	129	111.90	3.29	43.00	43	0.00	0.58	0.08
DDH-U-04-07	4212.3	4753.1	2253.7	-11	129	111.90	1.20	172.00	172	0.12	0.34	0.19
DDH-S-05-06	2439.2	5743.9	2492.7	-78	188	180.00	0.40	0.38	4	0.00	1.17	0.59
DDH-S-05-06	2439.2	5743.9	2492.7	-78	188	181.00	0.44	0.23	103	0.05	1.20	1.03
DDH-S-06-06	2644.7	5726.8	2452.7	-44	134	60.20	0.40	0.39	0	1.11	1.12	0.03
DDH-S-07-06	2594.6	5791.8	2456.5	-43.00	149	145.40	1.30	1.13	192	0.19	3.46	2.69
DDH-S-08-06	2763.3	5808.9	2476.0	-59	172	140.80	0.95	0.82	74	0.07	0.94	2.31
DDH-S-09-06	2763.5	5809.9	2476.2	-67	134	170.35	0.70	0.61	108	0.13	1.34	1.60
DDH-S-10-06	2763.5	5809.9	2476.2	-67	45	170.35	1.10	1.06	81	0.06	0.43	0.08
DDH-S-11-06	4136.4	5591.1	2458.3	-33	301	134.60	1.90	1.85	1349	0.51	0.30	0.11
DDH-S-12-06	4138.4	5592.5	2458.3	-37	336	130.00	2.05	1.92	940	1.41	0.48	0.66
DDH-U-13-06	4257.4	5019.1	2079.6	-3	183	54.20	2.95	2.26	988	0.28	2.62	6.62
DDH-S-14-06	4138.7	5590.6	2458.3	-75	314	180.00	1.70	1.09	481	0.21	0.83	0.63
DDH-U-15-06	4257.5	5019.5	2079.5	-21	179	68.50	1.50	0.84	130	0.15	0.61	1.58
DDH-S-16-06	4122.2	5546.3	2465.2	-46	317	170.25	1.35	1.27	594	0.37	0.30	0.40
DDH-U-17-06	2080.0	5019.4	4258.5	-5	158	45.85	1.35	1.17	2713	0.38	2.95	3.96
DDH-S-18-06	4122.6	5545.9	2465.1	-61	321	195.00	2.00	1.66	811	1.12	0.31	3.61
DDH-U-19-06	2079.4	5019.7	4258.5	-18	157	50.90	0.30	0.25	1471	0.78	3.66	3.72
DDH-U-20-06	4259.3	5020.0	2079.9	-13	133	64.80	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-21-06	4360.8	5062.9	2097.8	-23	204	127.55	1.70	0.98	1590	0.35	1.79	0.68
DDH-U-21-07	4360.8	5062.9	2097.8	-23	204	128.55	0.30	0.17	3661	4.80	4.86	9.40
DDH-S-22-06	4045.5	5602.8	2494.7	-62	339	128.60	3.05	2.34	442	0.15	0.13	0.32
DDH-S-23-06	4045.4	5603.5	2494.7	-48	339	80.20	0.75	0.67	356	0.40	1.50	0.12
DDH-S-24-06	4044.4	5602.6	2494.7	-61	290	100.25	1.60	1.23	188	0.09	0.41	0.14
DDH-S-24-06	4044.4	5602.6	2494.7	-61	290	101.25	3.15	3.15	711	0.23	0.62	1.04
DDH-S-25-06	4201.0	5644.4	2473.5	-58	304	147.75	1.75	1.50	340	0.30	0.82	1.93
DDH-U-26-06	4361.4	5062.5	2097.5	-6	183	95.40	1.60	1.13	22	0.08	0.38	0.17
DDH-S-27-06	4200.8	5664.0	2473.5	-47	284	130.15	1.85	1.81	1050	1.40	0.59	0.14
DDH-S-28-06	4214.8	5637.3	2473.2	-63	319	202.50	2.15	1.73	636	0.45	0.49	0.08
DDH-S-29-06	4204.4	5577.0	2461.6	-52	323	208.00	4.84	4.20	392	0.23	0.25	0.18
DDH-S-30-06	4204.7	5576.5	2461.5	-67	324	300.00	0.50	0.40	392	0.54	1.06	0.08
DDH-S-30-06	4204.7	5576.5	2461.5	-67	324	300.00	0.70	0.48	439	0.18	0.87	0.06
DDH-S-30-06	4204.7	5576.5	2461.5	-67	324	300.00	1.80	1.27	116	0.23	0.96	0.07

DDH-S-30-06	4204.7	5576.5	2461.5	-67	324	300.00	2.95	2.09	183	0.12	1.01	0.06
DDH-S-30-06	4204.7	5576.5	2461.5	-67	324	300.00	6.95	4.91	156	0.14	0.99	0.07

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
DDH-U-31-06	4563.1	5091.3	2147.9	0	348	20.75	1.45	1.26	92	0.19	0.61	1.46
DDH-U-31-06	4563.1	5091.3	2147.9	0	348	21.75	4.70	4.07	36	0.06	0.36	0.45
DDH-U-32-06	4560.8	5088.7	2147.8	0	304	22.30	2.20	1.91	14	0.01	0.23	0.14
DDH-U-33-06	4548.8	5077.2	2148.4	2	353	26.90	2.30	1.99	77	0.06	0.02	0.44
DDH-U-34-06	4545.6	5074.2	2148.3	0	282	24.05	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-35-06	4251.7	5680.4	2487.7	-42	317	200.50	0.70	0.54	249	0.36	0.17	0.26
DDH-U-36-06	3967.4	4934.8	2102.4	3	162	48.35	6.80	5.37	730	0.11	0.65	0.93
DDH-S-37-06	4252.6	5679.3	2487.7	-61	312	205.65	1.00	0.87	305	0.51	0.28	0.10
DDH-S-38-06	3782.3	5510.3	2518.9	-40	304	150.20	3.40	3.30	676	0.39	1.10	1.15
DDH-U-39-06	3967.4	4935.0	2102.4	6	129	16.15	2.25	1.84	406	0.38	0.71	1.78
DDH-S-40-06	3783.4	5509.5	2519.1	-63	305	123.50	0.60	0.50	212	0.07	0.90	0.45
DDH-S-41-06	3751.0	5528.7	2520.9	-39	314	67.60	2.55	2.48	514	0.40	0.98	0.87
DDH-S-42-06	3723.8	5514.4	2516.7	-41	309	100.90	1.95	1.92	660	0.46	4.78	0.47
DDH-S-43-06	3742.4	5498.5	2515.5	-68	313	76.35	1.25	1.04	93	0.20	0.33	0.37
DDH-U-44-06	3972.9	4855.6	2079.3	-23	1	94.15	0.40	0.40	84		0.88	1.13
DDH-U-44-06	3972.9	4855.6	2079.3	-23	1	95.15	2.70	2.54	236		0.67	0.60
DDH-U-44-06	3972.9	4855.6	2079.3	-23	1	96.15	1.50	1.49	435	0.50	0.33	0.18
DDH-U-44-06	3972.9	4855.6	2079.3	-23	1	97.15	0.90	0.90	317		1.68	2.23
DDH-S-45-06	3794.2	5453.3	2509.4	-54	310	139.00	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-46-06	3794.6	5538.1	2520.4	-47	312	80.10	3.60	3.48	344	0.40	1.41	0.16
DDH-S-47-06	3821.8	5510.8	2514.1	-69	209	131.00	2.15	1.80	395	0.61	0.99	0.27
DDH-S-47-06	3821.8	5510.8	2514.1	-69	309	131.00	3.30	2.77	1421	0.40	1.37	0.17
DDH-U-48-06	3972.9	4856.7	2080.4	-55	1	62.15	0.30	0.26	40	0.15	0.08	
DDH-U-48-06	3972.9	4856.7	2080.4	-55	1	63.15	3.20	3.10	952	2.07	0.31	0.87
DDH-U-48-06	3972.9	4856.7	2080.4	-55	1	64.15	4.30	4.27	531	0.30	0.16	0.48
DDH-S-49-06	3972.9	5510.4	2514.1	-81	305	145.40	1.50	1.10	262	0.26	0.15	0.40
DDH-U-50-06	3972.9	4855.5	2078.9	-60	307	93.50	2.90	2.63	534	0.82	0.68	1.31
DDH-U-50-06	3972.9	4855.5	2078.9	-60	7	93.50	5.70	5.51	1697	0.29	1.36	1.13
DDH-U-50-06	3972.9	4855.5	2078.9	-60	7	93.50	1.45	1.26	453	0.61	0.60	0.98
DDH-S-51-06	3853.5	5435.4	2501.0	-43	314	170.90	1.40	1.39	24	0.03	0.83	1.45
DDH-U-52-06	4654.8	5224.1	2120.7	-30	133	147.45	1.30	0.92	155	0.10	0.34	1.75
DDH-U-52-06	4654.8	5224.1	2120.7	-30	133	147.45	0.30	0.16	2225	0.33	8.57	5.30
DDH-S-53-06	3854.2	5434.6	2501.1	-66	316	182.35	2.00	1.70	690	0.28	2.58	3.08
DDH-S-54-06	4121.5	6037.2	2518.7	-39	151	81.00	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-55-06	4120.7	6038.6	2519.0	-74	150	136.00	0.50	0.25	496	0.24	2.12	0.18
DDH-S-56-06	4122.2	6038.0	2518.5	-37	125	90.10	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-57-06	4120.8	6038.9	2518.7	-62	119	170.70	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-58-06	2398.9	5639.3	2537.4	58	80	350.65	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-59-06	4571.8	5148.3	2115.4	-28	131	129.15	5.45	2.39	612	0.14	0.84	2.08
DDH-U-59-06	4571.8	5148.3	2115.4	-28	131	129.15	2.45	1.07	1032	0.21	1.34	3.37
DDH-S-60-06	2523.7	5648.9	2485.0	-57	76	150.60	2.05	2.05	30	0.26	0.50	0.32
DDH-S-60-06	2523.7	5648.9	2485.0	-57	76	150.60	2.15	2.15	111	0.28	0.06	0.44
DDH-S-61-06	3525.4	4552.8	2391.6	-53	346	227.70	1.20	1.17	214	0.10	1.00	0.16

DDH-S-62-06	3664.7	4654.5	2433.0	-60	322	230.20	1.40	1.17	442	0.23	0.19	0.65
DDH-U-01- 07	3752.0	4981.3	2196.9	-5	179	78.30	4.85	3.97	293	0.16	0.58	0.95

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
DDH-U-01- 07	3752.0	4981.3	2196.9	-5	179	78.30	0.80	0.66	117	0.20	3.04	2.01
DDH-U-01- 07	3752.0	4981.3	2196.9	-5	179	78.30	2.65	2.17	205	0.14	0.31	0.98
DDH-U-02- 07	3751.8	4981.1	2199.8	30	179	70.00	2.65	2.64	725	0.31	1.58	1.20
DDH-U-02- 07	3751.8	4981.1	2199.8	30	179	70.00	1.30	1.26	63	0.02	0.12	0.64
DDH-U-02- 07	3751.8	4981.1	2199.8	30	179	70.00	3.90	3.89	87	0.14	1.04	2.27
DDH-U-04- 07	3905.6	4927.2	2078.5	-19	156	122.95	1.30	1.00	1006	0.48	0.86	2.42
DDH-U-04- 07	3905.6	4927.2	2078.5	-19	156	122.95	1.35	1.17	292	0.17	0.73	1.51
DDH-U-04- 07	3905.6	4927.2	2078.5	-19	156	122.95	0.90	0.55	85	0.02	0.38	0.28
DDH-U-04- 07	3905.6	4927.2	2078.5	-19	156	122.95	8.60	6.59	461	0.26	0.13	0.68
DDH-U-04- 07	3905.6	4927.2	2078.5	-19	156	122.95	4.30	2.47	500	0.61	2.21	2.60
DDH-U-06- 07	3904.9	4927.0	2196.9	-18	181	128.45	3.90	2.99	214	0.11	0.27	0.78
DDH-S-07- 07	3911.0	5486.8	2466.5	-31	315	170.00	0.25	0.20	90	0.11	0.00	1.15
DDH-S-08- 07	3911.5	5486.3	2466.0	-53	317	186.65	0.90	0.69	630	0.16	0.46	1.10
DDH-U-09- 07	3823.2	5012.7	2208.8	0	178	100.20	2.30	21.60	159	0.18	0.64	0.68
DDH-S-10- 07	3912.2	5485.4	2467.2	-75	316	231.25	2.00	1.53	154	0.08	3.79	2.37
DDH-U-11- 07	3823.2	5012.4	2209.9	29	60	70.00	2.60	2.51	1494	0.52	1.29	1.63
DDH-U-12- 07	5832.4	3072.3	2394.4	-64	136	80.15	0.35	0.22	407	0.30	2.05	2.77
DDH-S-13- 07	3912.0	5485.9	2464.7	-66	318	188.70	2.30	1.76	122	0.05	0.01	0.41
DDH-S-13- 07	3912.0	5485.9	2464.7	-66	318	188.70	1.20	0.92	164	0.26	0.02	0.39
DDH-U-14- 07	3822.3	5012.4	2208.9	6	211	92.00	1.30	1.12	940	0.21	1.26	2.55
DDH-U-14- 07	3822.3	5012.4	2208.9	6	211	92.00	1.35	0.81	279	0.23	0.20	0.41
DDH-S-15- 07	3927.0	5523.7	2467.4	-54	317	103.50	0.90	0.78	40	0.14	0.44	0.30
DDH-U-16- 07	5831.0	3071.1	2395.9	-57	176	43.30	2.75	1.94	261	0.16	1.24	2.79
DDH-S-17- 07	3927.5	5523.1	2467.7	79	315	170.85	1.70	1.47	59	0.00	0.00	0.00
DDH-U-18- 07	3822.3	5012.4	2208.4	-25	211	117.25	1.20	0.66	110	0.11	0.25	0.64
DDH-U-19- 07	5831.6	3070.0	2394.5	-60	211	50.90	3.10	2.44	243	0.22	1.40	1.90
DDH-U-21- 07	5826.7	2990.6	2394.8	-43	178	60.60	1.05	0.79	104	0.07	0.65	1.73
DDH-U-22- 07	3854.9	5018.1	2221.5	-1	178	100.40	1.00	0.98	1362	0.43	3.34	4.46
DDH-U-22- 07	3854.9	5018.1	2221.5	-1	178	100.40	3.20	3.09	326	0.62	0.98	1.54
DDH-S-23- 07	3697.9	5477.0	2507.0	-31	315	76.00	0.50	0.50	621	0.30	0.91	1.61
DDH-U-24- 07	5828.0	2990.5	2394.9	-67	180	64.70	0.70	0.61	395	0.21	1.40	2.26
DDH-U-25- 07	3855.0	5017.9	2221.3	-20	178	78.55	3.85	3.15	447	0.16	0.44	0.53
DDH-S-26- 07	3699.5	5475.4	2507.1	-67	318	64.50	2.05	1.68	342	0.28	1.29	2.41
DDH-U-27- 07	3855.0	5018.3	2222.6	29	178	82.65	0.90	0.87	1185	0.28	3.03	10.40
DDH-U-27- 07	3855.0	5018.3	2222.6	29	178	82.65	1.75	1.64	158	0.02	0.20	0.16
DDH-U-28- 07	5827.9	2990.5	2395.0	-50	184	76.60	0.45	0.26	76	0.09	1.13	3.48
DDH-S-29- 07	3757.9	5417.2	2494.1	-50	315	109.65	1.10	1.06	271	0.24	0.43	0.27
DDH-U- 30- 07	3695.7	4985.6	2188.4	-20	177	107.60	3.15	2.41	226	0.22	1.10	0.58
DDH-S-31-07	3758.3	5416.3	2494.1	-68	318	140.00	0.85	0.74	144	0.11	0.28	0.96
DDH-U-32- 07	2990.1	5826.0	2394.9	-37	209	55.75	3.10	2.91	1305	3.43	0.59	1.53
DDH-U-32- 07	2990.1	5826.0	2394.9	-37	209	55.75	2.50	2.35	509	1.27	0.20	0.43
DDH-U-32- 07	2990.1	5826.0	2394.9	-37	209	55.75	2.55	2.40	205	0.55	0.10	0.22
DDH-S-33-07	3758.7	5416.0	2494.2	-82	318	156.00	1.80	1.56	109	0.07	0.63	0.15

DDH-U-34- 07	2990.1	5826.9	2394.9	-80	213	62.50	0.45	0.29	75	0.12	0.94	2.16
DDH-U-35-07	3695.7	4985.8	2189.8	-29	177	82.80	2.40	1.84	121	0.11	0.93	0.35

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
DDH-U-35-07	3695.7	4985.8	2189.8	-29	177	82.80	1.55	1.46	256	0.15	0.35	0.62
DDH-U-36- 07	2987.9	5826.2	2395.4	-23	245	41.25	3.95	3.03	587	31.47	0.48	0.36
DDH-U-36- 07	2987.9	5826.2	2395.4	-23	245	41.25	2.10	1.82	1801	49.84	0.70	0.76
DDH-U-36- 07	2987.9	5826.2	2395.4	-23	245	41.25	6.05	5.24	1008	37.85	0.55	0.50
DDH-U- 37- 07	3695.4	4985.6	2188.5	-15	210	124.65	0.80	0.69	717	0.19	2.61	11.30
DDH-U-38- 07	2989.2	5826.4	2394.8	-53	227	76.15	3.10	2.68	149	0.31	0.28	0.29
DDH-U-38- 07	2989.2	5826.4	2394.8	-53	227	76.15	1.50	1.15	1442	4.07	0.42	0.84
DDH-U-38- 07	2989.2	5826.4	2394.8	-53	227	76.15	1.60	1.39	676	1.32	1.14	2.34
DDH-S-39-07	5841.8	5277.7	2493.0	63	3	141.20	0.60	0.52	154	0.22	0.34	0.34
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	0.85	0.74	612	0.69	0.38	1.37
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	2.75	2.11	396	3.87	0.08	0.35
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	4.45	3.85	789	32.22	0.28	0.24
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	3.15	2.73	849	7.32	0.37	0.52
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	1.75	1.72	369	0.17	0.45	0.81
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	7.20	6.24	437	3.53	0.28	0.31
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	5.75	4.98	507	4.13	0.28	0.33
DDH-U-40- 07	2988.8	5826.0	2396.1	0	224	70.00	25.25	21.87	326	7.38	0.17	0.21
DDH-U-41-07	5841.8	5278.2	2493.0	50	1	131.85	1.90	1.65	205	0.29	0.30	0.14
DDH-U-41-07	5841.8	5278.2	2493.0	50	1	131.85	1.60	1.39	198	0.13	5.77	0.18
DDH-U-41-07	5841.8	5278.2	2493.0	50	1	131.85	4.60	3.98	320	0.11	0.33	0.10
DDH-U-42 - 07	3695.4	4985.8	2189.9	-31	29	80.35	0.80	0.61	569	0.10	0.46	0.43
DDH-U-43 - 07	3695.8	4985.6	2188.3	-26	153	35.90	0.80	0.75	127	0.24	0.28	0.08
DDH-U-44- 07	2991.3	5827.2	2395.0	-54	144	59.40	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-45 - 07	3695.8	4985.8	2190.0	31	147	65.25	1.55	1.19	379	0.43	1.05	0.48
DDH-U-45 - 07	3695.8	4985.8	2190.0	31	147	65.25	0.70	0.57	318	0.07	0.21	0.32
DDH-U-45 - 07	3695.8	4985.8	2190.0	31	147	65.25	3.00	2.46	360	0.06	1.06	0.88
DDH-S-46-07	5840.3	5276.8	2492.8	75	3	210.85	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-47-07	2990.4	5831.5	2396.2	0	357	50.10	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-48 - 07	3821.6	4916.5	2150.1	-51	1	50.10	1.90	1.65	108	0.04	0.45	0.26
DDH-U-48 - 07	3821.6	4916.5	2150.1	-51	1	50.10	1.60	1.39	18	0.01	0.31	0.12
DDH-U-49 - 07	3821.5	4916.2	2152.3	29	159	43.95	1.85	1.60	329	0.20	0.52	1.30
DDH-U-50 - 07	2989.3	5826.5	2397.7	26	226	82.30	1.35	1.27	1375	2.56	0.00	0.15
DDH-S-51-07	5890.5	5281.9	2492.6	-43	359	138.00	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-52 - 07	3825.3	4916.0	2150.1	21	41	57.45	6.25	3.13	495	0.15	0.54	0.77
DDH-U-53-07	3825.4	4916.2	2150.3	-49	40	36.05	4.05	3.51	498	0.29	0.50	1.79
DDH-S-54- 07	5900.5	5281.2	2492.6	-58	358	100.00	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-55-07	3821.6	4908.0	2150.8	-1	358	50.70	3.05	2.64	155	0.14	1.03	0.99
DDH-S-56- 07	5797.4	5287.8	2494.4	-42	3	166.65	1.50	1.10	83	0.09	0.05	0.07
DDH-U-57-07	4093.4	4946.0	2066.0	0	178	50.75	13.90	12.04	955	1.57	1.10	1.13
DDH-U-58-07	2987.9	5826.6	2397.4	9	234	50.70	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-59-07	3902.7	4931.7	2079.9	-56	329	65.25	2.30	1.99	26	0.11	0.10	0.15
DDH-U-60-07	3147.1	5866.9	2483.2	-58	183	76.90	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-62- 07	5772.2	5231.8	2496.7	-53	72	147.45	N/A	N/A	N/A	N/A	N/A	N/A

DDH-U-63 -07	3902.7	4932.0	2078.7	-19	331	121.25	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-64-07	3147.0	5867.7	2403.0	-79	188	106.05	N/A	N/A	N/A	N/A	N/A	N/A

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
DDH-U-65-07	3147.0	5866.0	2403.3	-34	1	69.40	0.90	0.89	103	0.07	0.27	0.34
DDH-U-65-07	3147.0	5866.0	2403.3	-34	1	69.40	0.60	0.59	254	0.20	0.44	0.30
DDH-S-66- 07	5772.8	5231.7	2496.7	-45	83	147.15	6.50	3.25	21	0.03	0.13	0.09
DDH-U-67-07	3146.7	5866.8	2403.2	-58	215	90.40	5.70	4.94	162	0.23	0.42	0.81
DDH-U-67-07	3146.7	5866.8	2403.2	-58	215	90.40	1.40	1.04	108	0.14	1.46	2.61
DDH-U- 68- 07	3957.7	4939.8	2081.0	30	329	82.30	4.15	3.48	301	0.10	0.74	0.34
DDH-S-69-07	5832.3	5345.1	2515.9	-54	68	80.05	18.20	15.76	140	0.40	0.28	0.11
DDH-U-70-07	3147.7	5866.9	2402.9	-57	150	110.70	1.20	0.77	225	0.28	0.61	2.16
DDH-S-71-07	5833.4	5345.5	2516.3	-28	68	80.50	13.80	13.75	376	0.16	0.36	0.58
DDH-S-71-07	5833.4	5345.5	2516.3	-28	68	80.50	14.25	14.20	147	0.24	0.81	0.04
DDH-S-72-07	5832.4	5345.5	2516.0	-40	58	80.60	8.40	8.40	114	0.04	0.38	0.10
DDH-S-73-07	5831.6	5345.7	2515.9	-28	42	50.40	10.40	10.40	85	0.05	0.19	0.05
DDH-S-73-07	5831.6	5345.7	2515.9	-28	42	50.40	12.25	12.25	96	0.07	0.15	0.06
DDH-S-73-07	5831.6	5345.7	2515.9	-28	42	50.40	7.95	7.95	259	0.19	0.07	0.06
DDH-S-74-07	5832.3	5346.4	2516.2	-58	46	60.00	1.40	0.70	123	0.38	0.78	0.14
DDH-U-75-07	3215.0	5835.7	2404.7	-47	184	81.20	0.75	0.52	128	0.19	1.27	3.03
DDH-U-75-07	3215.0	5835.7	2404.7	-47	184	81.20	0.20	0.16	2165	0.85	14.60	10.05
DDH-S-76-07	3732.7	5378.9	2473.3	-35	314	119.00	1.40	1.39	464	0.87	1.15	0.85
DDH-U-77-07	3215.1	5835.8	2404.2	-64	186	135.40	0.30	0.14	6636	0.77	17.40	6.90
DDH-U-79-07	3217.0	5836.2	2404.0	-55	159	95.05	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-80-07	4872.9	5210.9	2109.3	-61	313	147.20	2.20	2.20	194	0.11	0.29	0.21
DDH-U-80-07	4872.9	5210.9	2109.3	-61	313	147.20	2.12	2.12	156	0.15	0.28	0.36
DDH-S-81- 07	3733.8	5377.5	2473.2	-68	316	151.50	1.25	0.96	205	0.23	0.24	0.33
DDH-U-82-07	3283.6	5818.8	2422.4	-42	181	80.00	6.10	6.10	376	0.11	1.42	2.12
DDH-U-83-07	3283.7	5819.5	2422.3	-63	176	80.20	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-84-07	3284.2	5819.6	2422.9	-41	145	90.50	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-85-07	2929.4	5842.3	2486.2	-56	182	151.75	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-86-07	2929.4	5841.3	2486.0	-42	181	117.25	N/A	N/A	N/A	N/A	N/A	N/A
DDH-U-87- 07	4873.0	5210.5	2107.7	-73	341	116.00	2.40	1.84	1307	0.30	5.42	7.84
DDH-U-87- 07	4873.0	5210.5	2107.7	-73	341	116.00	2.25	1.59	481	0.18	1.30	3.61
DDH-S-88-07	2929.3	5841.5	2486.2	-69	6	160.65	10.20	7.09	500	14.33	0.69	1.11
DDH-U-89-07	3281.9	5819.8	2422.4	-45	211	68.95	N/A	N/A	N/A	N/A	N/A	N/A
DDH-S-90-07	2929.6	5843.6	2487.3	-80	185	215.45	1.25	0.37	551	0.22	1.63	3.37
DDH-S-90-07	2929.6	5843.6	2487.3	-80	185	215.45	2.85	1.51	98	0.16	0.27	0.58
DDH-S-90-07	2929.6	5843.6	2487.3	-80	185	215.45	2.15	1.14	275	0.75	0.26	0.31
DDH-U-91-07	3284.1	5819.1	2422.9	-23	185	80.45.	1.25	1.25	812	0.17	2.00	2.35
DDH-S-93-07	2900.3	5861.6	2483.7	-63	182	185.85	0.90	0.50	734	1.21	3.20	14.25
DDH-S-93-07	2900.3	5861.6	2483.7	-63	182	185.85	6.05	3.64	213	11.78	0.66	0.78
DDH-S-93-07	2900.3	5861.6	2483.7	-63	182	185.85	2.25	1.35	546	267.38	0.62	0.68
DDH-U-94-07	2899.8	5860.2	2484.7	-51	182	144.25	1.50	1.36	110	0.12	0.46	2.03
DDH-S-96-07	2899.8	5860.2	2484.7	-51	182	144.25	0.65	0.51	171	0.13	1.47	
DDH-U-97-07	3360.6	5845.4	2423.3	-52	2	143.65	1.10	0.78	1025	0.23	1.45	2.29
DDH-U-97-07	3360.6	5845.4	2423.3	-52	2	143.65	0.80	0.57	1486	0.86	2.45	0.60

DDH-S-98-07	2900.2	5881.6	2483.7	-70	2	100.00	0.90	0.70	376	0.74	0.57	0.94
DDH-S-98-07	2900.2	5881.6	2483.7	-70	2	100.00	0.85	0.32	132	0.56	0.80	3.05

Hole#	Easting	Northing	Elevation	Dip	Azimuth	Length	Down hole intersect	True width	Ag g/t	Au g/t	Pb %	Zn %
DDH-U-99-07	3360.4	5845.2	2423.1	-49	195	120.00	1.50	1.36	280	0.09	1.31	1.28
DDH-U-99-07	3360.4	5845.2	2423.1	-49	195	120.00	3.00	2.72	54	0.15	1.96	1.42

*Notes:

(1) The drill core and results are no longer available, for the purposes of resource estimation these holes have not been used.

(2) A N/A designator means the drill hole did not intersect mineralization.

14.0 SAMPLING METHOD AND APPROACH

The La Colorada database consists of 2 types of samples underground channel samples and diamond drill core samples.

The sampling method for the underground exploration development along mineralized structures consists of channel sampling the back or roof of the tunnel every 3 metres. Vein and wall rocks are sampled separately. Sample lines are marked by the geologist and then the sample is taken by a helper using a hammer and chisel. Sample size is approximately 3 kilograms. To provide an accurate representation of vein grades, samples are taken regardless of whether the vein appears to be above cut-off or not.

In addition to the samples taken from sampling of the exploration development, the database now includes stope samples taken from mining from 2001 to September 2007 in the NC2E, NC2W, and 4235 veins. Stope sampling methodology is the same as the exploration development sampling, except that samples are taken at 5 metre intervals. The reason for the 5 metre spacing as opposed to the 3 metre spacing is that the stope samples normally represent a smaller tonnage.

In the opinion of the authors of this Technical Report the sample spacing used in the exploration development headings and the stopes is appropriate and provides sufficient data density for mineral resource estimation.

Drill holes are sampled and logged according to industry-accepted standards. A staff geologist logs the holes for lithology, alteration, mineralogy, and recovery. Sample intervals are marked by the geologist who also assigns a sample number. As with the underground sampling, the samples are broken out by geology and vein and wall rock are sampled separately. Samples are split using a diamond saw.

Recovery in the drill holes was generally high (+80%), with the exception of the holes drilled into the NCP Corridor ore zone where the recovery averaged 67%. In the opinion of Michael Steinmann, P.Geo., there was no significant bias in the drill holes that had poorer recovery.

The structural controls of the La Colorada orebodies are relatively simple. The mineralized zone is encountered in the Inferior and Superior Volcanic Formations of the Eocene and Oligocene periods respectively. Vein mineralization generally occurs within dilation faults crosscutting the host rock. The Inferior Volcanic formation is a sequence of porphyritic dacite-trachytes of subvolcanic origin, which interclasts with pyroclastic and volcanoclastic sediments. The Superior Volcanic formation consists of volcanic tuffs and pyroclastic flows of rhyolitic composition placed in disconformity with the Inferior Volcanic formation. Mineralized widths in the veins are generally less than 2 metres, but may be wider if there is a halo of replacement or brecciated material. Sample width is based on visible vein width, which varies generally from less than a metre to 15 metres. Wide vein intersections are sampled in several intervals dependant on the visible mineralization changes. In areas where the wall rock shows disseminated mineralization, additional samples are taken.

In the opinion of the authors of this Technical Report, the samples are of an acceptable quality for mineral resource and mineral reserve estimation. To the best of the authors knowledge, there are no factors that may have resulted in a sample bias and the samples are representative.

Channel samples are the main contributor for the calculation of mineral resources and mineral reserves. They are also an important tool in determining mining constraints. As there are over 10,000 channel samples in the Plata database, it is not practical to list them or a summary of them in his Technical Report.

In conclusion, it is author s opinion that the sampling method applied at La Colorada by Plata gives representative results and meets industry standards.

15.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

Plata employees prepared all the various drill and channel samples and analyzed the samples at the La Colorada Laboratory.

Geology and exploration samples are sent to the laboratory as either geology channel samples (chip samples) or as diamond drill core. The diamond drill core has been logged to industry standards and is then cut in half with a diamond saw with half of the sample being used for the assay. The other half is stored by the geology department for future reference. The diamond drill core is reduced to a size of minus 1/4 by passing it through a cone crusher. After the diamond drill core has been crushed, the chip samples and crushed drill core are treated in the same manner. If either of the sample types are visible wet at this time, they are put in a 110 120°C oven to remove the humidity. Approximately, 250 gram samples are then taken and passed through a John Quarterer to ensure sample homogenization providing a representative sample. The sample is then dried in an oven at 110 120°C making sure that the sample identification is with the actual sample. When dry, the samples are put in a properly identified paper envelope where they await pulverization. The samples are eventually ground to 100% passing a 200 mesh screen and are replaced within the same envelope to await analysis. Generally a 200 gram sample will be taken from the pulverised sample and the remainder of the pulverised sample is given to the geology department to enable random check samples.

The La Colorada mine laboratory uses fire assay for gold and silver on a ten gram charge with a gravimetric finish. Base metals were assayed by the La Colorada laboratory using acid digestion and titration prior to the acquisition of atomic absorption (AA) equipment in 2003. Since then, the base metals have been assayed using acid digestion and AA determination. All assaying by the commercial labs for gold and silver was done using fire assay with either an AA or gravimetric finish on a one-assay tonne charge. Base metals were assayed by the commercial labs using acid digestion and AA determination.

Various steps are taken to ensure the quality and reproducibility of data within the La Colorada laboratory. The LIMS system automates the majority of the data input eliminating some of the error encountered with manual data input. Samples of known concentrations are tested at random within the AA and fire assay processes to verify no significant biases exist. Ph buffer solutions are used for calibration within the AA process. Additionally, there is a QA/QC program supervised by the geology department. It includes the submission of random samples to secondary laboratories for check assays (mostly to ALS Chemex) on 2-5% of the samples and 1-2% of check samples. This program was started with the new diamond drilling campaign in 2005. Historically, the deviation between Chemex and La Colorada lab has been within acceptable limits and no significant biases have been encountered.

Pan American has used the following four commercial labs in the past for the exploration assaying at La Colorada:

> ITS Bondar Clegg, 130 Pemberton Ave., North Vancouver, BC, Canada. This laboratory is registered to ISO 9001: 2000 for the provision of assay and geochemical analytical services by QMI Quality Registers. This laboratory has also received ISO 17025 accreditation from the Standards Council of Canada. This laboratory was acquired by ALS Chemex on December 01, 2001.
>
> ALS Chemex, 212 Brooksbank Ave., North Vancouver, BC, Canada. This laboratory is registered to ISO 9001: 2000 for the provision of assay and geochemical analytical services by QMI Quality Registers. This laboratory has also received ISO 17025 accreditation from the Standards Council of Canada.
>
> Luismin, Laboratories, De Selenio y Aluminio, Cd Industrial Durango, Durango, México. Prior to 2003 Luismin laboratory was certified under ISO 9000. In February, 2006 the laboratory was acquired by SGS and operates as SGS de México S.A. de C.V. Laboratorio de Durango. The laboratory is currently in the process of re-certification.
>
> ALS Chemex de México, Ignacio Slazar 688, Hermosillo, Sonora, México. This laboratory is used for sample preparation with prepared samples sent to the ALS Chemex laboratory in North Vancouver, British Columbia, Canada. This laboratory is registered to ISO 9001: 2000 for the provision of assay and geochemical analytical services .

All of the drilling, sampling and QA/QC programs were conducted under the direct supervision of PAS geology staff.

In the opinion of Michael Steinmann, P.Geo., the sample preparation, security and analytical procedures are of adequate quality for mineral resource and mineral reserve estimation.

16.0 DATA VERIFICATION

The sampling and analytical data entered into the La Colorada database has been checked with only minor transcription errors found. The assays were checked against the plan maps in the case of older data and against the assay certificates in the case of newer data.

As part of an ongoing check program, the production assays entered into the database are routinely checked against the assay certificates (in the case of tertiary labs) and against the check assays (in the case of the La Colorada lab).

The QA/QC programs are conducted under the direct supervision of PAS geology staff and periodically revised by Michael Steinmann, P.Geo.

The authors of this Technical Report have relied on the data verification work conducted by the geology staff at La Colorada Mine. Summary results used in the resource estimation have been verified by Michael Steinmann, P.Geo.

17.0 ADJACENT PROPERTIES

Adjacent properties are not relevant for this review of the La Colorada property.

18.0 MINERAL PROCESSING AND METALLURGICAL TESTING

18.1. Mineral Processing

Two distinct types of ore are being treated at the La Colorada mine: one is classified as oxide ore and the other is classified as sulphide ore. A description of the mineral processing method is presented in section 25.2 Bench scale metallurgical testing and full-scale plant operations have determined optimum processing methods of cyanidation for the oxide ore and selective lead/zinc sulphide flotation for the sulphide ore. Tables 7 and 8 illustrate the metal recovery estimates and the predicted head grades used in the 2008 2011 La Colorada LOM plan based on metal recoveries achieved in 2007.

Table 7 Predicted Metal Recoveries for La Colorada Ores

2008 - 2011 Projected Plant Recovery	**Au**	**Ag**	**Pb**	**Zn**
Sulphide Circuit	67.0%	92.0%	82.0%	73.0%
Oxide Circuit	77.0%	82.0%		

Table 8 Predicted Head Grades

		2008	**2009**	**2010**	**2011**
Oxide Plant	***Tonnes***	199,484	193,497	166,541	100,520
	Silver g/t	421	404	454	484
	Gold g/t	0.50	0.47	0.54	0.59
	Lead %	0.55	0.66	0.72	0.86
	Zinc %	0.95	1.06	0.95	1.03
Sulphide Plant	***Tonnes***	115,010	143,919	111,285	67,923
	Silver g/t	443	399	386	396
	Gold g/t	0.50	0.43	0.40	0.43
	Lead %	0.88	1.00	0.96	0.94
	Zinc %	1.54	1.95	1.91	2.09

Actual Oxide Plant Results

Table 9 shows the actual metal recoveries achieved in the oxide plant during 2005, 2006 and 2007 as of September 30. In 2007, 81.5% of the contained silver and 78.2% of the contained gold was recovered into dorè.

Table 9 Actual Metal Recoveries of Oxide Ore Achieved from 2005 Sep 30, 2007

Year	Tonnes	Au % Recovery	Ag % Recovery
2005	211,854	80.96	82.62
2006	213,187	79.46	85.69
2007 (as of Oct 17, 2007)	155,784	78.20	81.49

Actual Sulphide Plant Results

When PAS first purchased the La Colorada property, an existing 120 tpd sulphide flotation circuit produced lead concentrate. In 2004, PAS decided to temporarily cease operations within the sulphide circuit. This was as a result of a lack of sulphide ore available to feed the sulphide circuit and the circuit needed repair work to make operating conditions more reliable. In July 2006, the sulphide circuit was re-opened for lead concentrate at a feed rate of 100 tpd. While producing lead concentrate, a zinc flotation circuit and another ball mill were being added to the sulphide circuit increasing the feed capability to 250 tpd and enabling the production of zinc concentrate. By June 2007, the construction of the zinc circuit and the implementation of the additional ball mill were completed. Since June, there has been an increase in capacity of 200 tpd to a total of 450 tpd. This was made possible by increasing underground mining equipment efficiency and installing a dewatering system allowing more sulphide feed into the circuit and improving and maintaining mechanical equipment within the sulphide plant itself. Table 10 shows actual metal recoveries achieved in the sulphide plant during 2006 and 2007 (to Oct 17, 2007).

Table 10 Actual Metal Recoveries of Sulphides Ore Achieved from 2005 Oct 17, 07

Year	Tonnes Processed	% Au Recovered	% Ag Recovered	% Pb Recovered	% Zn Recovered
2006	**20,557.03**	**62.80**	**87.73**	**71.77**	
2007 Zn con	*77,108.54*	*66.87*	*85.43*	*77.07*	*13.18*
2007 Pb con	*77,108.54*	*5.29*	*5.37*	*3.53*	*48.05*
2007 Totals	***77,108.54***	**72.16**	**90.80**	**80.60**	**61.23**

18.2. Lab Sampling and Analysis

Two different processes are used to test plant samples in determining head grade and plant recovery. Samples from the plant are initially received with solids suspended in solution and both the solids and the water is tested for metal concentrations. The solution is decanted taking a 200 ml sample and placing it within a 600ml beaker along with 5ml of lead salt, 3 grams of zinc and 15ml of hydrochloric acid. The solution is then heated until an amalgamation is formed containing precipitated silver and it is moulded into a spherical shape. It is then wrapped in a thin lead sheet and is ready for fire assaying. The solids of the initial plant solution are filtered, rinsed with neutral water and then dried. 200 grams of the solids are taken and placed in a properly labelled envelope and await atomic absorption (AA) and fire assaying.

The La Colorada mine laboratory uses fire assay for gold and silver on a ten gram charge with a gravimetric finish. Base metals were assayed by the La Colorada laboratory using acid digestion and titration prior to the acquisition of AA equipment in 2003, and since then the base metals have been assayed using acid digestion and AA determination. All assaying by the commercial labs for gold and silver was done using fire assay with either an AA or gravimetric finish on a one-assay tonne charge. Base metals were assayed by the commercial labs using acid digestion and AA determination.

18.3. Metallurgical Testing

Bench scale metallurgical testing and full-scale plant operations have determined optimum processing methods of cyanidation for the oxide ore and selective lead/zinc sulphide flotation for the sulphide ore. Projected recoveries are based on actual recoveries achieved in 2007. A description of the metallurgical testwork that has been completed for PAS to date is described below.

The metallurgical assumptions used for the economic analysis in this Technical Report are based on actual plant performance comprising of hundreds of thousands of tonnes of the La Colorada ore material. These samples are considered by the co-authors to be representative.

Pre-operational Bench Scale Testing:

Bench scale metallurgical test-work conducted prior to the commissioning of the oxide cyanidation plant in mid-2003 was completed at the La Colorada mine site laboratory and the following commercial laboratories:

Process Research Associates Ltd. 9145 Shaughnessy Street, Vancouver, BC, Canada, (Professional engineers in Canada).

Luismin, Laboratories, De Selenio y Aluminio, Cd Industrial Durango, Durango, México Prior to 2003 Luismin laboratory was certified under ISO 9000. In February, 2006 the laboratory was acquired by SGS and operates as SGS de México S.A. de C.V. Laboratorio de Durango. The laboratory is currently in the process of re-certification.

Table 11 shows the initially predicted metal recoveries in oxide ore based on all available test work at that time.

The Feasibility Study recoveries were reviewed by an independent engineer (KD Engineering, 7701 N. Business Park Drive, of Tucson, Arizona professional registered under the laws of the state of Arizona) and, based on this review, were adjusted for the feasibility update.

Table 11 Predicted Metal Recoveries of Oxide Ore

	Feasibility Study		*Independent Engineer*		*Update*	
Structure	***% Ag***	***% Au***	***%Ag***	***%Au***	***%Ag***	***%Au***
NC2W	90.00	88	90.00	87.45	90.00	87.5
4235	90.00	88	90.00	87.45	90.00	87.5
NCP	84.75	80	84.44	75.00	84.45	75.0
Tailings	76.00	80	72.77	80.00	73.00	80.0

1999 Testwork

During 1999, metallurgical testwork was concentrated on flotation and cyanidation leach testing. Fresh samples of vein material were obtained from the back of the mineralized 295 level drift and from diamond drill intercepts for both the oxide and sulphide ore types. These samples provide representative material of the mineralogy of each the sulphide and oxide ore type.

Selective flotation tests by Process Research Associates Ltd. were conducted on both ore types as well as gravity plus cyanidation tests for the oxide type.

For the oxide type, both near surface drill intercepts and along the 295 level drift, a 15 kg/tonne silver low grade flotation concentrate with limited marketability could be produced. Recoveries of silver were significantly less than combined gravity-leaching processing.

Locked cycle cyanide leach testing was conducted on a narrow vein oxide ore from above the 295 level drift at a grind of approximately 80% minus 70 micron. This indicated excellent recoveries for silver as shown in Table 12.

Table 12 Bottle Roll Test Results (Oxide Ore Above 295 Level)

Sample	***Type***	***Cycle***	***Grind 80% Passing***	**96 hr Silver Recovery *%***	**96 hr Gold Recovery *%***	***Calc Silver Grade gpt***	***Calc Gold Grade gpt***
3	Oxide	1	66	94.5	91.2	657	1.91
		2	70	92.5	90.6	660	1.80
		3	68	93.0	87.7	640	1.38
		4		90.6	88.5	639	1.48

For the sulphide type ore, a clean bulk Ag-Pb concentrate can be floated with relatively high recoveries. It is also possible to produce a Zinc concentrate, but difficulties in depressing zinc reporting to the lead concentrate were encountered. In addition, preliminary testing indicated significant deportment of silver to the zinc concentrate. Table 18 provides a metallurgical balance of the flotation procedure.

Table 13 Flotation Results: Stage 1

		Weight	Assay (g/t)		Assay (%)		Recoveries %			
Sample	***Product***	***(%)***	***Au***	***Ag***	***Pb***	***Zn***	***Au***	***Ag***	***Pb***	***Zn***
DH4	Cu/Pb Cln Conc	8.6	6.5	13,383	38.0	7.2	57.1	87.2	83.0	18.5
	Zn Cln Conc	1.8	2.3	671	3.60	26.6	4.3	0.9	1.7	14.6
	Rougher Tails	63.1	0.14	71	0.30	0.13	9.0	3.4	4.8	2.5
	Head	100	0.98	1324	3.95	3.36	100	100	100	100

Continued work with collectors lead to significantly improved zinc recovery and concentrate grade, as shown in Table 14.

<u>Notes to tables 13 through 20:</u>

Cu/Pb Cln Conc:	copper lead cleaner concentrate
Zn Cln:	zinc cleaner concentrate
Rougher Tails:	tailings from the rougher flotation circuit
Head:	ore feed to the circuit
Cu/Pb Rougher:	copper lead rougher flotation product
Zn Rougher:	zinc rougher flotation product
Pb Cln 2:	second or final lead cleaner concentrate
Pb Cln 1:	first or intermediate lead cleaner concentrate
Pb Rougher:	lead rougher product
Zn Cln 2:	zinc cleaner 2 concentrate

Table 14 Flotation Results: Stage 2

		Weight	Assay (g/t)		Assay (%)		Recoveries %			
Sample	***Product***	***(%)***	***Au***	***Ag***	***Pb***	***Zn***	***Au***	***Ag***	***Pb***	***Zn***
DH1	Cu/Pb Cln Conc	2.6	4.5	8455	35.2	1.14	43.5	66.0	90.7	2.0
	Cu/Pb Rougher	9.0	1.7	3184	10.8	1.44	54.6	84.7	94.6	8.8
	Zn Cln Conc	2.3	0.5	1475	0.22	54.8	4.3	10.2	0.5	86.5
	Zn Rougher	6.8	0.34	547	0.16	19.06	8.4	11.0	1.1	87.6
	Rougher Tails	84.2	0.12	17	0.05	0.06	37.0	4.3	4.3	3.6
	Head	100	0.27	338	1.02	1.48	100	100	100	100

Flotation recovery and, more importantly, metal deportment are seriously affected by grind size and reagent use. Testwork has shown that combining these factors can produce saleable concentrates with excellent recoveries for the sulphide ore. Table 15 shows projected sulphide ore metallurgy.

Table 15 Projected Typical Sulphide Metallurgical Balance

Product	*Weight (%)*	*Assay (g/t) Au*	*Ag*	*Assay (%) Pb*	*Zn*	*Recoveries % Au*	*Ag*	*Pb*	*Zn*
Cu/Pb Cln Conc	2.67	8.43	14,044	44.9	13.0	75.0	75.0	85.0	14.4
Zinc Conc	3.73	0.87	2,178	1.70	52.1	10.8	16.25	4.50	80.0
Tails	93.6	0.05	46.7	0.16	0.15	14.2	8.75	10.5	5.60
Head	100	0.30	500	1.41	2.43	100	100	100	100

Bond work index tests were run on both oxide and sulphide material from narrow vein areas. Results ranged from 15.9 to 20.0 kilowatt hours per tonne, with the majority of samples approximately 18 kilowatt hours per tonne. All flotation testwork after initial scoping tests were conducted targeted a grind of 80% minus 74 micron.

2000 Testwork

Testwork in 2000 on the NCP corridor material, which represents the majority of the proven and probable reserves, has consisted of bottle roll tests on drill core, flotation followed by cyanidation on the drill core, and locked cycle tests on composite samples made from the drill core. All of this testwork was carried out at Process Research Associates Ltd. (PRA) in Vancouver, BC, Canada.

Six drill holes were shipped to PRA s Vancouver facilities for testing and included drill holes PIC 35, 36, 37, 39, 40 and 41. All holes except PIC 35 were complete mineralized intercepts. The drill holes were characterized geologically to represent material that ranged from mostly oxide to a mixed oxide and sulphide mineralization.

Preliminary bottle roll cyanidation testing was performed on all six drill holes, and the results are shown in Table 16. These scoping tests indicated that the ore was amenable to cyanidation leaching, but like previous testwork, showed that recovery was grind dependent.

Table 16 Drill Core Scoping Bottle Roll Tests

Drill Hole #	*Type*	***Grind 80% Passing***	**96 hr Silver Recovery %**	***Calc Silver Grade Gpt***
35	Oxide	82	79.0	358
36	Oxide	77	76.1	227
37	Oxide	75	79.2	200
39	Mixed	111	79.7	768
40	Mixed	64	64.0	505
41	Mixed	82	73.7	801

Utilizing parameters established on previous metallurgical samples, a four cycle (96 hours per cycle) locked cycle test was conducted. The drill holes were composited into two samples, one representing highly oxidized material and one representing the mixed mineralization. Results confirmed the increase in recovery at a finer grind (see Table 17). Both composites exhibited similar leaching characteristics and ultimate recoveries. Losses of precious metals are likely due to both silica encapsulation and the presence of sub micron silver. The consistency of recoveries during the cycles aided in establishing projected plant recovery levels.

Table 17 Locked Cycle Results

				Leach Recovery Silver %			Silver Grade g/t
Sample	***Type***	***Cycle***	***24 hr***	***48 hr***	***72 hr***	***96 hr***	***Calculated***
1	Mixed	1		80.2	81.3	81.6	646
		2		68.2	74.3	79.5	661
		3		73.1	79.2	78.3	643
		4		64.5	75.1	80.3	684
2	Oxide	1		77.4	83.5	84.0	266
		2		79.8	81.0	84.3	256
		3		79.2	82.4	82.9	265
		4		77.6	80.7	84.1	277

Gold recovery for both composites during locked cycle testing is shown in Table 18. Gold recoveries are difficult at these low grades. Cycle 3 on the mixed composite was ignored when projecting design recoveries largely due to the discrepancy between calculated grades in this cycle versus the other three cycles.

Table 18 Gold Recoveries in Locked Cycle Testing

			Gold Recovery (%) Versus Time		Gold Grade (g/t)
Sample	***Type***	***Cycle***	***48 hr***	***96 hr***	***Calculated***
1	Mixed	1	70.5	77.0	0.65
		2	57.8	81.7	0.65
		3	37.0	42.8	0.95
		4	68.8	74.7	0.55
2	Oxide	1	66.3	67.9	0.40
		2	55.7	71.3	0.35
		3	72.0	68.6	0.38
		4	63.0	70.3	0.47

Flotation testwork was conducted on all individual drill core samples and also on both composite samples. During preliminary testwork, flotation tails were also cyanided to determine the results of a combination flotation/leaching plant. Individual drill core results varied with silver recoveries to a bulk lead silver concentrate ranging from 53% to 80%. Leaching of these tailings recovered a further 65% to 72%.

The oxide and mixed ore composites were subjected to differential flotation to recover both lead and zinc concentrates. Tables 19 and 20 illustrate the metallurgical balances obtained for both composites. On the mixed composite, saleable concentrates could be made recovering 50% of the silver in the lead concentrate, and 11.7% of the silver in the zinc concentrate. Gold recovery from the two concentrates was approximately 30%. On the oxide composite, a saleable lead concentrate could be made, but precious metal recovery was poor and no saleable zinc concentrate could be made.

Table 19 Flotation Results on NCP Oxide Ore

	Weight		*Assay (g/t)*		*Assay (%)*		*Recoveries %*			
Product	***(g)***	***%***	***Au***	***Ag***	***Pb***	***Zn***	***Au***	***Ag***	***Pb***	***Zn***
Pb Cln 2	8.74	0.22	4.12	4,272	28.6	3.05	3.41	3.80	17.9	0.77
Pb Cln 1	42.5	1.07	2.11	2,749	7.74	2.91	8.48	11.9	23.6	3.56
Pb Rougher	333	8.42	1.12	1,175	1.30	1.63	35.4	39.8	31.1	15.6
Zn Cln	88.5	2.24	2.43	1,869	0.37	10.8	20.4	16.8	2.35	27.4
Zn Rougher	461	11.7	0.66	495	0.30	2.74	28.7	23.3	10.0	36.3
Rougher Tails	3159	79.9	0.12	115	0.26	0.53	35.9	36.9	58.9	48.1
Head	3953	100	0.27	248	0.35	0.88	100	100	100	100

Table 20 Flotation Results on NCP Mixed Ore

	Weight		*Assay (g/t)*		*Assay (%)*		*Recoveries %*			
Product	***(g)***	***%***	***Au***	***Ag***	***Pb***	***Zn***	***Au***	***Ag***	***Pb***	***Zn***
Pb Cln 2	20.9	0.53	10.6	32,900	40.0	2.98	13.6	27.1	29.0	0.84
Pb Cln 1	54.0	1.37	7.17	23,340	19.1	4.13	23.7	49.6	35.9	3.00
Pb Rougher	250	6.34	2.25	6,720	4.60	2.86	34.5	66.1	40.0	9.60
Zn Cln 2	40.6	1.03	2.43	1,869	0.37	10.8	20.4	16.8	2.35	27.4
Zn Rougher	504	12.77	0.42	752	0.45	4.80	12.9	14.9	7.93	32.6
Rougher Tails	3190	80.9	0.27	151	0.47	1.35	52.6	18.9	52.1	57.8
Head	3944	100	0.41	645	0.73	1.89	100	100	100	100

<u>Post Oxide Plant Construction Plant Support Testwork</u>

Additional testing was conducted on-site following the completion of the construction and commissioning of the oxide cyanidation plant in mid-2003. The test programs were designed to further examine the deposit metallurgical variability, optimum conditions for metal extractions, methods to reduce operating costs, and potential to increase productivities. In general, the results of the various site test programs conclude:

The primary characteristic of the ore deposit that controls the metallurgical responsiveness either in the cyanidation or flotation circuit is the degree of oxidation;

The metallurgical performance of either the cyanidation or flotation circuits are inversely proportional to the intensity of grinding with marked metallurgical improvements obtained with finer grinds;

The amount of clay material in the ore and the intensity of grinding can negatively impact the performance of the thickening wash circuit in the oxide plant;

The primary ore bodies that contribute the high clay materials are the San Fermin and Veta Dos deposits;

The concentration of true-free cyanide in the leaching and Merrill Crowe circuits is crucial to optimum cyanidation plant performance; and

Additional flotation studies conducted on-site support the conclusions of the various historical laboratory test results presented above.

19.0 MINERAL RESOURCE AND MINERAL RESERVE ESTIMATES

The mineral resource modelling in 2007 was done at the mine site under the supervision of the authors of this Technical Report. All mineral resources and mineral reserves quoted below have been done in accordance with accepted industry practices and are in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum definitions on Mineral Resources and Mineral Reserves.

The resource calculation is based on both diamond drilling and underground channel sampling. The channel sampling is a combination of PAS sampling (NC2W and 4235) and previous sampling campaigns (NC2E, Recompensa, Estrella). The only mineral resources in the current mine plan that include older channel samples are in NC2E. In this area PAS did a check-sampling program that confirmed grades and widths of the original samples are adequate for mineral resource estimation. Samples were taken across the structures every 3 metres in principal development and every 5 metres in stopes. Samples are analyzed for Ag, Au, Pb, Zn, Cu, Fe and Mn at the on-site laboratory using AA and fire assay methods. The laboratory conducts a routine internal QA/QC program that includes external check samples and the routine submission of standards.

Additionally, there is a QA/QC program supervised by the geology department. It includes the submission of samples for check assays. The secondary lab checks are analysed by ALS Chemex in Hermosillo.

The mineral resource estimation has been done using the polygonal method. Corrections for mining method, mining recovery, dilution from wall rocks and dilution from backfill have been taken into account.

Specific gravities of the different oxide ore and sulphide ore are a function of the lead and zinc grade contained within the ore. Table 21 shows the specific gravity formulas used to calculate the tonnes of mineral reserves and mineral resources for the La Colorada Mine.

Table 21 Specific Gravity Testwork

Ore type	SG ore	SG wallrock
Oxides	= 2.7 + (%Pb + %Zn) * 0.024	2.30
Sulphides	= 2.77 + (%Pb + %Zn) * 0.024	2.30

The mineral resource classification scheme used differentiated between channel sample and drill hole data. Measured mineral resources extended 25 metres above and/or below a drift. Indicated mineral resources from drift data extend 35 metres below the bottom of the measured drift (for a total of 60 metres measured and indicated) or 20 metres from a drill hole composite. For areas that have 2 or more drillholes within the search criteria, the indicated resource category is extended to a total of up to 50 metres. Inferred mineral resources are based on vein continuity for a particular area and are generally strike and dip extensions of measured and indicated resources.

The minimum mining width is 2.0 metres in order to permit access for the scooptrams. Veins of less than 2 metres true thickness have added wall dilution to a total of 2 metres true thickness. Veins greater than 2 metres have no additional wall dilution added. Wall dilution is added at no grade, even though sampling of this material shows some sub-economic mineralization.

The tonnes in each block are reduced by a factor of between 5% and 15% depending on width (i.e. mining recovery is assumed to be between 85% ad 95%). This figure is based on experience and observation at the La Colorada Mine and takes into account losses of ore in permanent pillars, losses into the backfill and other losses such as those that may be caused by ground failures or other geomechanical conditions. The mine workers attempt, where possible, to recover all pillars, however, some crown pillar ore and some safety pillars inevitably remain to ensure safe working conditions for the miners in the stopes.

The calculated mineral reserves assume that backfill is inadvertently mined during the process and that there is some loss of ore. This is equivalent to a 7.5% reduction in mined grade for no net change in tonnage. At the end of 2006, there was also a difference between the head grade of the ore at the mill and the LOM plan grade. The source of the error is unknown, but it could be a result of overstating the mineral reserve and mineral resource grades or it could be a result of under-achieving the LOM plan. As a result, PAS has further increased the amount of dilution reducing the grade by 13% to account for this difference since it was the most likely source of error.

The grades for each of the blocks are then re-calculated taking into account the dilution from the wall rock, where applicable, and the backfill.

La Colorada Mine consists of polymetalic deposits with production of 4 different metals; hence no simple cut off grade can be used to determine the economic viability of a block. Instead, net smelter return (NSR) calculations are used to assign a value per tonne (VPT) for each block. Oxide and sulphide ore blocks are assessed differently due to different factors within the NSR calculation and therefore oxide and sulhide metals have different VPT factors. A value per tonne is calculated using reserve metal price assumptions and using separate formulae for the oxide and the sulphide ore. The metal price assumptions for the reserve calculation at the end of September 2007 were as follows:

Table 22 Metal Price Assumptions

Silver	US$/ Ounce	$	11.00
Gold	US$/ Ounce	$	600.00
Zinc	US$/ Tonne	$	2,100
Lead	US$/ Tonne	$	1,700

In the case of the oxide ore, the product is dorè bars and the valuable metals are silver and gold only. The factors are calculated by taking the product of the relevant price, the metallurgical recovery and the refinery recovery and subtracting the refinery fees. The factors for the oxide ore are as follows:

Table 23 Value per Tonne Factors for Oxide Ore

Silver	US$per 1 g/t	$	0.2913
Gold	US$per 1 g/t	$	15.2420
Zinc	US$per 1%	$	0
Lead	US$per 1%	$	0

In the case of the sulphide ore, the products are lead / silver and zinc concentrates and the valuable metals are silver, gold, lead and zinc. The factors are calculated taking into account the metallurgical recoveries for each metal into the flotation concentrates, the distribution of the metals between the concentrates, the smelter terms for each of the concentrates (including percent payable, any penalty element deductions, treatment charges and refinery deductions), concentrate freight, insurance and warehouse charges. All of these factors are taken from the actual concentrate sales agreements that were in place at the mine in 2007. Pan American continues to sell this type of concentrate from its operations in Peru and Bolivia, and is familiar with the current concentrate market conditions. On this basis the factors used in this study are considered to be appropriate. The calculated factors for the sulphide ore are as follows:

Table 24 Value per Tonne Factors for Sulphide Ore

Silver	US$per 1 g/t	$	0.3007
Gold	US$per 1 g/t	$	15.9921
Zinc	US$per 1%	$	6.2595
Lead	US$per 1%	$	5.0394

The appropriate factors are then applied to the block grade in order to calculate a value per tonne for each of the blocks.

A cut off value per tonne for both sulphide and oxide ores within the different mines has been established. The cut off VPT for the sulphides (all within the Candelaria Mine) is $58 per tonne. The cut off VPT for the Candelaria oxides is $67 per tonne and for the Estrella oxides it is $58 per tonne. The increase in cut off value within the Candelaria oxide zone is due largely to higher water pumping costs and weaker ground conditions that require additional ground support.

Some of the blocks have not been converted from mineral resources to mineral reserves because they are either remote or inaccessible and the cost of development would be excessive, or because there are other uncertainties that call their economic viability into question. The mineral resource blocks that exceed the cut off grade and have not been eliminated for other reasons are converted to mineral reserves on the basis that a measured mineral resource is converted to a proven mineral reserve and an indicated mineral resource is converted to a probable mineral reserve.

There are no known issues relating to the environmental, permitting, legal, title, taxation, socio-economic, marketing, political, metallurgical, infrastructure or other relevant factors that would materially affect the resource and reserve estimates reported in this Technical Report.

The resource calculation methodology used the various long section projections within the polygonal methodology:

NCP HW (hangingwall vein) Long section projection (Fig. 9);
NCP HW2 (hangingwall vein 2) Long section projection (Fig. 10);
NCP Vein Long section projection (Fig. 11);
NCP Inverse Vein Long section projection (Fig. 12);
NCP Split Vein Long section projection (Fig. 13);
NC2 Vein Long section projection (Fig. 14);
Recompensa Vein Long section projection (Fig. 15); and
Estralla Vein Long section projection (Fig. 16).

19.1. Mineral Reserves

The proven and probable mineral reserves at the La Colorada mine as of September 30, 2007 are estimated to be as follows:

Table 25 La Colorada Mineral Reserves

		Grade			
Category	**Tonnage kT**	**Ag g/t**	**Au g/t**	**Pb %**	**Zn %**
Proven	449.40	421.57	0.46	0.01	0.01
Probable	566.50	460.12	0.53	0.01	0.01
Total Reserve	**1,015.90**	**443.07**	**0.50**	**0.01**	**0.01**

Notes:

1. Total grades of silver and zinc are shown as contained metal before mill recoveries are applied.

2. La Colorada mineral reserves have been estimated at a cut off value per tonne of $66.53 in the Calendaria Mine and $58.31 in the Estrella Mine for oxide ore and $58.48 per tonne in sulphide ore.

3. The geological model employed for La Colorada involves geological interpretations on sections and plans derived from core drill hole information and channel sampling.

4. Mineral reserves have been estimated using the O Hara dilution formula, which typically adds 20% to 50% dilution at zero grade depending on dip angle and vein width. As a result of reconciliation to actual production the mining dilution is increased by a further 13%.

5. Mineral reserves have been estimated using a mining recovery of 85-94% (pillars are left in some thicker zones leading to lower mining recovery). A further 7.5% subtracted from the grade with no change in tonnage to further account for other mining losses.

6. Mineral reserves were estimated based on the use of cut and fill mining methods. The mining rate is projected to be a maximum of 940 tpd ore for the full year of 2008. The processing plants have the capacity to process more than this and are assumed to process all of the ore produced by the mine in each year.

7. Mineral reserves are estimated using polygonal methods on longitudinal sections.

8. Mineral reserves were estimated using a price of $11.00 per ounce of silver, $600 per ounce of gold, $2,100 per tonne of zinc and $1,700 per tonne of lead.

9. Environmental, permitting, legal, title, taxation, socio economic, political, marketing or other issues are not expected to materially affect the above estimate of mineral reserves.

19.2. Mineral Resources

The measured and indicated mineral resource estimates for the La Colorada mine as of September 30, 2007 are in addition to the mineral reserves and are estimated to be as follows:

Table 26 La Colorada Mineral Resources

Category	Tonnage k tonne	Ag g/t	Grade Au g/t	Pb %	Zn %
Measured	186.79	329.66	0.51	0.01	0.01
Indicated	547.33	243.62	0.30	0.01	0.01
Total Resource	**734.11**	**265.51**	**0.36**	**0.01**	**0.01**
Inferred	1701.18	346.46	0.39	0.02	0.02

Notes:

1. PAS reports mineral resources and mineral reserves separately. Reported mineral resources do not include amounts identified as mineral reserves.

2. The geological model employed for La Colorada involves geological interpretations on sections and plans derived from core drill hole information and channel sampling.

3. Mineral resources have been estimated using the O Hara dilution formula, which typically adds 20% to 50% dilution at zero grade depending on dip angle and vein width. .As a result of reconciliation to actual production the mining dilution is increased by a further 13%.

4. Mineral resources have been estimated using a mining recovery of 85-94% (pillars are left in some thicker zones leading to lower mining recovery). A further 7.5% is subtracted from the grade with no change in tonnage to further account for other mining losses.

5. Mineral resources were estimated based on the use of cut and fill mining methods. The mining rate is projected to be a maximum of 940 tpd ore for the full year of 2008. The processing plants have the capacity to process more than this and are assumed to process all of the ore produced by the mine in each year.

6. Mineral resources are estimated using polygonal methods on longitudinal sections.

7. Mineral reserves were estimated using a price of $11.00 per ounce of silver, $600 per ounce of gold, $2,100 per tonne of zinc and $1,700 per tonne of lead.

8. Environmental, permitting, legal, title, taxation, socio economic, political, marketing or other issues are not expected to materially affect the above estimate of mineral resources.

9. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Mining blocks were created from the variograms and classified as measured, indicated or inferred based on the relative confidence of the supporting data for each evaluated block. Erratic high Ag values have been corrected for the mineral resource calculation. The La Colorada mineral deposit contains high grade, minable ore shoots and a simple arithmetic top cut to the database would eliminate entire high grade areas. In order to prevent that, the spatial location of a sample has been taken into account. Samples are collected along a structure and are plotted as silver gram per tonne (g/t) grade, width (m) of the vein and Ag grade multiplied by the width in order to identify minable ore shoots along the veins. For an easy method to locate the ore shoots, trend lines are plotted over the three datasets. Single outliers, or non-minable small ore shoots, are visually identified and the grades are replaced by the grades of the predicted trend line. The corrections are applied before the vein samples and mineralized footwall / hangingwall samples are composited. Although this method represents a rather unusual way of applying a top cut, the authors of this Technical Report agree that it represents a valid method for the La Colorada deposit, eliminating high grade outliers and with that reducing risk from the mineral resource estimation.

20.0 OTHER RELEVANT DATA AND INFORMATION

No other data or information is relevant for the review of the La Colorada project.

21.0 INTERPRETATION AND CONCLUSIONS

This Technical Report demonstrates that the proven and probable mineral reserves and measured and indicated mineral resources presented in this report will be economic with the forecast metal prices and other assumptions presented herein. Based on the current mineral reserve and mineral resource estimates, the mine is projected to operate until the end of 2011. This projected mine life may increase if future resources are converted to reserves. The undiscounted net present value (NPV) for the La Colorada mine is $9.77M based on the LOM plan production. The current realized metal prices are higher than those used for the reserve calculations and for the economic analysis presented in this Technical Report.

In the authors opinion, the diamond drilling and channel sampling information that has been collected is of sufficient density for resource and reserve estimation.

The QA/QC programs are conducted under the direct supervision of Plata s geology staff and periodically revised by Michael Steinmann, P.Geo. The authors of this Technical Report have relied on the data verification work conducted by the geology staff at La Colorada. Summary results used in the mineral resource estimation have been verified by Michael Steinmann, P.Geo..

This report details the methodology employed and demonstrates why the authors conclude that the continued operation of the La Colorada mine is technically feasible and economically viable. It is the opinion of the authors of this Technical Report that the data contained herein is of sufficient quality and reliability to make the conclusions stated.

22.0 RECOMMENDATIONS

As the mine is currently in operation the work programs necessary to maintain annual updates to the mineral reserve estimates are in place and being conducted on a daily basis by a full complement of technical and operating staff at the mine. The costs for these work programs are included in the annual operating budgets, mine plan and life of mine plan that are shown in section 25.6 Capital and Operating Costs. Andrew Sharp has worked at the La Colorada Mine for the past year and Michael Steinmann and Martin Wafforn visit the La Colorada Mine and supervise various projects on a regular basis throughout the year. It is recommended that PAS continue to follow the LOM plan and make the capital investments that are detailed in that plan. It is further recommended to continue to follow the current sampling and quality control programs as may be revised from time to time by the authors of this Technical Report. It is also recommended to continue with the diamond drilling program and the related sampling and quality control programs in order to assure sufficient data density for future new resource estimations in deeper or lateral parts of the mine as well as for satellite deposits. The mine has a budget in 2008 of US$1.5M in order to conduct exploration and definition drilling programs in an attempt to convert resources to reserves and locate new orebodies. These exploration programs are closely supervised and revised by the Senior V.P. of Geology and Exploration for PAS, Michael Steinmann, P.Geo.

The authors of this Technical Report recommend that the mine should continue to operate and that the mineral reserve and mineral resource statement presented herein be adopted.

23.0 REFERENCES

AGRA Earth and Environmental, *Tailings Facility Design Report and Addendum, May 2000*
Agra Simons, *Updated Basic Engineering Report for the La Colorada Project, May 2000*
Beacon Hill Consultants, *Hoisting System Evaluation, June 2000*
Beacon Hill Consultants, *Mining Calculations Detail, June 2000*
Dew Point International LLC, *Mina La Colorada Environmental Action Plan, August 2002*
Dew Point International LLC, *Mina La Colorada Environmental Impact Assessment, May 2000*
Golder Associates, *Dewatering Requirements La Colorada Mine, July 2004*
Golder Associates, *Dewatering Requirements November 2004, update La Colorada Mine, December 2004*
Lewis Geoscience, *Structural Analysis, La Colorada Mine , October 1998*
Luismin Labs, *Estudio de Cianuración, April 2000*
MRDI (now a division of AMEC), *La Colorada Project Geologic Modelling and Resource Estimation Report, May 2000*
Pan American Silver Corp., *Annual Information Forms for 1999, 2000, 2001, 2002, 2003, 2004, 2005 and 2006*
Pan American Silver Corp., *Feasibility Study, La Colorada Mine, México, June 22, 2000*
Pan American Silver Corp., *La Colorada Mine Project, Zacatecas, Technical Report, August 29, 2003*
Pan American Silver Corp., *La Colorada Mine Project, Zacatecas, Technical Report, March 17, 2006*
Pan American Silver Corp., *La Colorada Project, México. Feasibility Update, February 2002*
Process Research Associates Ltd., *Flotation and Cyanidation Study on Samples from La Colorada, May 2000*

24.0 ADDITIONAL REQUIREMENTS FOR TECHNICAL REPORTS ON DEVELOPMENT PROPERTIES AND PRODUCTION PROPERTIES

24.1. Underground Mine Operations

The underground mining operations at the La Colorada mine provide up to 1,000 tpd of ore to the processing plants. The 2008 La Colorada mine plan is based on providing 540 tpd to the oxide plant and a further 400 tpd to the sulphide plant. The oxide plant was commissioned in 2004 and has a maximum capacity of 650 tpd. The sulphide plant has a maximum capacity of 450 tpd, having been expanded from 250 tpd in June 2007. The mining operations currently have slightly less capacity than the combined capacity of the plants.

The Candalaria underground mine produces the majority of the oxide ore and all of the sulphide ore at La Colorada. There are two main vein systems at Candalaria called NCP and NC2E. The NCP vein contains the majority of the oxide mineral reserves and the NC2E the majority of the sulphide mineral reserves. Transition areas containing a mixture of sulphide and oxide ores are typically processed in the oxide plant depending on the degree of oxidation. Ore and waste from the Candalaria mine is hoisted to a transfer level at the same elevation as the surface facilities where it is hauled by train and mine cars a short distance to surface. Wherever possible waste development rock is stored underground for use as backfill in the mechanized cut and fill mining method that is employed.

The La Estrella underground mine is located 0.65 kilometres from the Candalaria Mine. Production averages 150 tpd of oxide ore from a single vein. Ore and waste are hauled to the surface using 15 ton capacity 4 wheel drive mine trucks and dumped on to stockpile areas. The ore is later loaded into surface dump trucks and hauled to the oxide plant. Waste is typically re-handled underground and used as backfill in the mechanized cut and fill stopes.

Mining operations at the San Fermin and the Veta Dos underground mines are now complete. The San Fermin ramp was previously connected to the Candalaria mine and now forms the upper portion of the Western decline access to Candalaria. In addition to providing access the ramp is used to haul ore and waste from the west portion of the NCP vein to surface.

Access to all of the mining areas is via declines from the surface that are nominally 3.6 metres wide by 3.6 metres high grading at -15% (Figure 8). There are 2 declines from the surface accessing the La Estrella Mine, and a further 2 declines accessing the Candalaria Mine. The Candalaria ramp system connects to all of the currently active areas of the mine providing a back up method of haulage to the hoisting shaft (El Aguila shaft), escapeways in the East and West as well as a ventilation route.

The Candelaria s El Aguila shaft extends from the surface to the 353 level (353 metres below the surface). The shaft was upgraded and rehabilitated in 2004 and 2005 by PAS and contractor crews (Dynatec Corporation now FNX Mining Company Inc.). This work, when combined with the preventative maintenance program in place, should ensure safe and reliable operation for the life of the mine. Work done on the El Aguila shaft included scaling and supporting the walls, replacing timber sets as required, straightening out the compartments and aligning the guides to vertical positions, installing larger diameter pipelines for services and dewatering, extending the shaft signalling system, deepening the shaft to allow installation of new loading pockets, development and installation of new truck dump grizzlies, ore and waste pass raises, and slusher operated measuring bins. The new loading pockets are at the 343 level. The ore and waste is hoisted in rock skips to separate bins located just below the surface. Ore and waste are trammed out of the mine on the 20 level (20 metres below surface) in rail cars to dump points. The ore is taken either directly to the crusher feed bin or to a storage area adjacent to the feed bin using a front end loader. Waste is hauled away and is typically returned to the mine via a backfill raise and placed in the stopes as backfill.

The mine has recently replaced the drum hoist with a larger reconditioned unit in order to provide more reliable operations and greater hoisting capacity. The new hoist started production in 2007 and is capable of skipping approximately 2,000 tpd. It is a 1.68 metre diameter double drum unit with mechanical clutches that is equipped with a 400 HP direct current, thyristor controlled motor. It is installed in a new hoist room that was constructed behind and above the previous hoist to minimize downtime when changing from the old hoist. The skips dump underground, meaning that the sheave wheels are installed directly on top of the shaft collar and there is no head frame. Other than the hoist replacement, few other modifications were required. The skip capacity remains at 2 tonnes per skip for sulphide ore and 1.7 tonnes per skip for oxide ore. The difference is attributable to different densities of broken sulphide and oxide ores. The new hoist has sufficient capacity to meet the ore and waste production requirements that are scheduled in the life of mine plan that forms part of the economic analysis for this Technical Report.

The longitudinal section of the mine in Figure 8 shows the hilly surface topography, the El Aguila shaft, the mine portal entrances and declines, as well as the other principal development in the mine. Not all of the orebody of the previous stope mining is shown in order to retain some clarity in the view. Longitudinal sections showing the portions of the orebody already extracted and those remaining in known reserves and resources are shown in Figures 9 to 16.

The ventilation network is designed to provide ventilation for the diesel equipment, to provide sufficient air volume to remove heat in the sulphide portion of the mine, and to prevent build up of gasses in other areas. The life of mine plan presented in the economic analysis contains capital investments for ventilation raise bore holes and additional primary ventilation fans in order to provide the necessary expansions to the ventilation circuit as the mine is developed to depth. These investments are required in order to comply with the mining regulations for worker health and safety in Mexico. The heat in the sulphide portion comes from hot water inflows into the mine and to a lesser degree from the operation of diesel equipment in the mine. At the 345 level, the water temperature is approximately 30 degrees Celsius and requires excellent ventilation in order to remove the heat to achieve legislated (and tolerable) working conditions. Sulphide mining is currently on the 370 and 390 levels where water temperatures are slightly higher. Ventilating air is exhausted from the mine using fixed exhaust fans via the El Aguila shaft (59,000 CFM) and the San Fermin exhaust raise (82,000 CFM). Air intakes are via the surface declines and a raisebore hole with air distributed around the mine using fixed fans in bulkheads as well as auxiliary fans and ducting. The current ventilation system has some inherent problems however. There are three fresh air entrances ventilating the mine and only one exhaust raise located within the oxide section of the mine. All of the exhaust air from the sulphide portion of the mine mixes with the fresh air provided for the oxide section. In 2008 there are plans to expand the ventilation system within the Calendaria Mine by providing separate ventilation systems for the sulphide and oxide sections of the mine. By early 2008, one 2.4 metre diameter raisebore hole will be drilled down to the 345 level within the sulphide section and will serve as an exhaust raise. There are also two raises planned and included in the capital investments presented in the economic analysis within the oxide section of Calendaria Mine, which when driven, will improve ventilating air distribution.

Over the course of the recent mining history at the Candalaria Mine, dewatering requirements have increased as mining became deeper. The mine has constructed a system of pumping stations to pump the water to surface. As mining progresses to access deeper mineral reserves and mineral resources, the pumping requirements to dewater the mine are expected to increase the capital cost of the additional pump stations and are included in the economic analysis in the life of mine plan. The increase in the pumping requirements also leads to an increase in the amount of power required.

The sulphide plant operation was halted at the end of 2004 because the pumping capacity was not available at that time to dewater the NC2E zone below the 345 metre elevation. A new high capacity dewatering system was constructed and the sulphide section was re-opened in 2006. Production has now progressed in the sulphide section to the 390 Level with plans to develop the ramp down to the 420 Level in 2008. The oxide section of the Candelaria Mine is being mined at the 390 Level currently. Development is in progress on the 438 Level and downramp to the 468 Level where the pumping system will be expanded to dewater the lower levels of the mine. The pumping system is currently rated for 1800 USGpm.

Electrical power is supplied from the CFE grid (CFE is a Mexican power utility) in Zacatecas via a 34 KV power line to a 5 MVA 34KV/13.2 KV transformer that feeds the mine property distribution system. There is also a 34 KV feeder tap to a second transformer used for the underground transformers. The agreement with CFE is for the mine to take 3.5 MW. Distribution to the underground is via 13.2KV/4.16KV and 13.2KV/2.3KV transformers. Local transformers reduce the voltage to the mine and plant normal operating 480 volts. With the additional electrical loads from the mine dewatering system and the expanding ventilation network, the mine has reached the capacity of the current distribution system, and will need additional power for the further expansion of the ventilation and dewatering pumping systems. The CFE grid is already heavily loaded and is subject to power outages particularly during the late summer months when there are rain storms. An additional 90 KV power line is currently being constructed running 54 km from Sombrerete to the La Colorada Mine and is scheduled to be completed by the end of 2007 with startup in January 2008. This line will provide additional power needed to run the dewatering pumps and air compressors while eliminating the need for the diesel generators. The capital expenditures are included in the economic analysis; the expenditures are justified on the basis of reducing the amount of higher unit cost diesel generated power.

The mine has 300 HP and 400 HP electric air compressors on the surface. The compressed air is distributed throughout the underground via a network of steel pipes. There is ample water available for mining and process. This water is distributed via a network of pipes.

The explosives magazine is approved by the military and is presently operating at the mine site. The commercial magazine is located in Fresnillo, which is only 3 hours away by truck.

The mining method for both Estrella and Candelaria Mines is mechanized cut and fill with waste rock being utilized for backfill. The development access scheme is shown in Figure 17. This diagram depicts a crosscut being developed to the orebody from an access ramp that is 64 metres in the footwall of the orebody. In the lower part of the mine, where dewatering is conducted as mining progresses, the initial cut is developed at +3% (as shown in the Figure 17) in order to promote drainage from the stope. In the upper parts of the mine that have been dewatered by mining below, the initial access is driven at -20% to the orebody. After the initial cut has been mined and filled the stope is re-accessed either by slashing the back of the initial crosscut, if ground conditions permit, or by developing a new crosscut. The cuts are mined and backfilled progressively until the upper crosscut is at the maximum positive gradient for the scooptrams which is typically +22.5%. At this point either a new access is started further up the access ramp or, as is the case shown in Figure 14, the crown pillar between the cuts is recovered. In the newer parts of the mine, the backfill in the first cut has been cemented and the pillar can be recovered completely. In the older parts of the mine, it has been found that the backfill compacts extremely well and has some cohesion. This permits La Colorada personnel to blast the ore up to the backfill in 8 metre long slices and muck out the ore with a scooptram without exposing the operator. A 2 metre pillar is then left and another 8 metre long portion of the crown pillar extracted. The ore reserves account for leaving these pillars and other random support pillars in the stopes by reducing the overall stope tonnes by 5-15% depending on the mining width.

Figure 18 shows the details of the cut and fill mining method that is in use at La Colorada. The first panel of the figure shows the miners drilling horizontal drill holes using hand held jackleg drills. Once completed, these holes are loaded with explosives and blasted. As shown in the second panel, roof and wall support in the form of split set bolts are then installed working off of the pile of broken ore. The drillers are required to use a mechanical roof jack to provide additional roof support until the split sets are in place. The third panel in the figure shows the ore being extracted using a 2 cubic yard capacity scooptram and finally the ore is replaced with waste backfill in order to repeat the cycle. The ore reserves assume that additional non-mineralized backfill is inadvertently mined during this process and some ore is left behind effectively lowering the grade by 7.5% for no net change in tonnage. At the end of 2006, there was also a difference between the head grade of the ore at the mill and the LOM plan grade. The source of the error is unknown, but it could be a result of overstating the mineral reserve and mineral resource grades or it could be a result of under-achieving the LOM plan. As a result, PAS has further increased the amount of dilution, reducing the grade by 13% to account for this difference since this was the most likely source of error.

PAS personnel employ their best efforts in order to control and minimize the amount of waste rock and low grade dilution. The NCP zone has 2 main vein structures; the NCP FW and NCP HW, both of which typically dip at 62°. The NCP FW contains the biggest reserve and varies from 1 metre to 9 metres wide with an average diluted width of 2.7 metres. The NCP HW varies from 2 to 5 metres wide, averaging 3 metres. The more heavily altered and oxidized portions of these zones present some challenges to mining because the higher grade vein portion is often within a very weak corridor characterized by high clay content. Where this corridor is exposed, it is a support problem that requires it to be shotcreted as soon as possible before it starts to deteriorate. La Colorada underground technical and supervisory personnel identify areas where the conditions are particularly bad from experience on previous mining lifts and in those areas leave a 0.3 metre skin of the stronger vein material on the walls. The walls are regularly test holed to ensure that excess ore is not left behind. This ore loss is accounted for in the resource calculation. The NC2 zone dips at between 55° and 70° and varies between 1 and 3 metres wide with an average width of 2.2 metres. The sulphide portions of the NC2 zone are typically good ground and mining can progress quickly. The La Estrella mine is close to the surface and mines a heavily oxidized ore vein. In areas of particularly high clay content the split sets bolts that would otherwise have low pull test resistance are cemented to provide additional support. As mining progresses deeper at La Estrella ground conditions appear to be improved over those close to the surface.

The mines are being developed to permit fast and efficient movement of equipment, personnel and materials via a system of ramps that connect back to the shaft for haulage of ore and some waste to the surface. Main ramps that will have trucks on them are developed at 3.6 metres wide by 3.6 metres high with the face drilling normally done by a one boom jumbo and bolting done using hand held drills from the muck pile or bolts are installed by the one boom jumbo that has a boom configuration to allow it to drill the holes required to install split sets. The waste is removed using 3.5 cubic yard scooptrams and where possible is taken directly to a stope that is being backfilled for disposal. Stope accesses are typically 2.4 metres wide by 2.4 metres high to permit 2 cubic yard capacity scooptrams into the stopes. These accesses are normally drilled and bolted using hand held drills.

PAS prepared a long range plan in 2007 utilizing the mineral reserve and mineral resource information available at that time. Some assumptions were also made for the exploration success that is likely at the mine and in the surrounding properties. This plan was used to determine mining and processing rates, production profile, the timing of investments in major items such as the dewatering equipment, the sulphide plant and the tailings dam. From this long range plan, a valuation was derived which demonstrates that the mine has positive economics at the assumed metal prices. The authors of this Technical Report conclude that the La Colorada Mine remains a viable ongoing operation and the proven and probable mineral reserves presented in this Technical Report comply with the requirement that they are economic based on their assumptions.

The long range mine plan shown in Table 27 uses mineral reserves and mineral resources at the end of 2007 to calculate the production, revenue, operating cost, capital costs, taxes and other financial information. The mine plan is based on mining and processing parameters that are currently being achieved in the mine.

Table 27 Long range mine plan based on Sep 30, 2007 reserves

	2008	2009	2010	2011
Oxide Plant Tonnes	199,484	193,497	166,541	100,520
Ag Grade (g/t)	421	404	454	484
Au Grade (g/t)	0.50	0.47	0.54	0.59
Ag Recovery %	82.05	83.04	81.87	81.40
Au Recovery %	77.00	77.00	77.00	77.00
Silver Ounces Produced	2,216,682	2,089,496	1,988,255	1,274,362
Gold Ounces Produced	2,479	2,233	2,215	1,477
Sulphide Plant Tonnes	143,919	111,285	67,923	46,786
Ag Grade (g/t)	399	386	396	441
Au Grade (g/t)	0.43	0.40	0.43	0.48
Pb Grade %	1.00	0.96	0.94	1.28
Zn Grade %	1.95	1.91	2.09	2.95
Ag Recovery %	92.9	97.0	97.0	97.0
Au Recovery %	71.1	83.5	83.5	83.5
Pb Recovery %	79.0	79.0	79.0	79.0
Zn Recovery %	73.0	73.0	73.0	73.0
Lead Concentrate Tonnes	3,059	2,267	1,362	1,274
Zinc Concentrate Tonnes	3,166	2,395	1,598	1,556
Silver Ounces Produced in Concentrate	1,715,931	1,340,740	838,696	644,288
Gold Ounces Produced in Concentrate	1,427	1,182	776	604
Lead Tonnes Produced in Concentrate	3,059	2,267	1,362	1,274
Zinc Tonnes Produced in Concentrate	3,166	2,395	1,598	1,556
Total Silver Ounces Produced	3,932,613	3,430,236	2,826,951	1,918,650
Payable Silver Ounces	3,806,915	3,335,001	2,765,674	1,872,209
Payable Lead Tonnes	1,077	798	480	449
Payable Zinc tonnes	1,742	1,318	879	856
Payable Gold Ounces	3,716	3,249	2,848	1,984

24.2. Milling

Two distinct types of ore are being treated at the La Colorada Processing Plant: one is classified as oxide ore and the other is classified as sulphide ore. Two processes exist to treat the different ore types. Figures 19 and 20 demonstrate the process flowsheets for the sulphide and oxide ore circuits.

Sulphide Area

Sulphide ore containing valuable silver, lead, gold and zinc is recovered as concentrate in this circuit via froth flotation. The sulphide plant capacity is currently 450 tpd and the process flowsheet is shown in Figure 20. The activities required within this circuit to produce a concentrate include:

Crushing:
The plant operation for both the sulphide and oxide ore begins with the crushing circuit. The sulphide ore is brought to a stockpile via locomotives where it is then mucked into a 24 x 36 grizzly feeding a coarse ore bin. The ore passes through a jaw crusher feeding a conveyor belt that transports the ore to a 4 x 8 vibrating screen. Oversize is fed to a gyratory crusher forming a closed circuit with the jaw crusher. The undersize of the vibrating screen passes through a fine ore bin if it is sulphide ore or a stockpile if it is oxide ore.

Grinding:
The fine ore bin feeds the first ball mill via a feed belt. The feed belt weighs the ore with a Ramsey scale to control the production rate. The ball mill (2.5 Dia. Steel balls) feeds the slurry via two 5 x 4 pumps (one on stand-by) to a 10 diameter cyclone which classifies the slurry particle size. Each pump has its own cyclone and the undersize is fed into the flotation circuit. The oversize slurry is fed to the second ball mill (5 X 8) which then feeds another 10 diameter cyclone. This cyclone is fed by two 4 x 3 pumps (one on stand-by) and the cyclone undersize is sent to the flotation circuit.

Flotation:

Lead Circuit

The fines from the grinding process are fed to the lead flotation circuit composed by a primary flotation cell and a secondary flotation cell and are complemented by two Sub-A flotation banks that serve as first and second lead cleaners. The concentrate produced in the primary flotation is treated in the two cleaning stages and their concentrate is the final product, which is sent to the corresponding thickening tank. The concentrate from the secondary cell is fed to the primary flotation together with the cleaned tailings. The tailings from the secondary flotation are fed to the zinc conditioner.

Zinc Circuit

The slurry from the conditioner is fed to the primary flotation and the secondary flotation cell. The circuit is complemented by three cleaning stages; the first two are Sub-A flotation cells and the third one is a hybrid cell. The primary concentrate is treated in three cleaning stages and the resulting product is the final zinc concentrate, which is sent to a thickening tank. The concentrate from the secondary flotation cell and the tailings from the cleaning stages are returned to the primary flotation. The tailings from the secondary cell are pumped to the final tailings thickening tank.

Thickening and Filtering the Concentrates:
The underflow slurry of the lead thickening tank is pumped to filtering discs where water is extracted from the slurry. The resulting concentrate is transported to the concentrate storage area where it awaits shipping. The same process is applied for the zinc thickener and underflow slurry.

Tailings Disposal:
The tailings produced from the zinc circuit are sent to a thickening tank. The thickener underflow is sent underground via a pump to be used as backfill. The thickener overflow is reused as clarified water in the milling processing circuit.

<u>Oxide Area:</u>
Mineral containing valuable gold and silver is recovered through a precipitate leaching process and is later converted into Dore bars. The oxide plant has a capacity of 650 tpd and the process flowsheet is shown in Figure 19.

Crushing:
The crushing circuit is the same circuit as described in the sulphide process.

Grinding:
The stockpile is loaded into a feeder that in turn feeds a belt into ball mill #1 (9.5 x 11). The belt has an integrated Ramsey weighting system to control the production rate. The mill works with 3 diameter balls and the slurry produced is sent by two 5 x 4 pumps (one in stand-by) to the 20 diameter cyclone for classification. The underflow feeds ball mill #2 (8 x 10) with 2.5 diameter balls. The slurry that passes through mill #2 feeds the same bin as mill #1, thus forming a closed grinding circuit.

Chemical Treatment:
The slurry is sent to a primary leaching process comprised of 7 agitation stages. Afterwards the slurry is fed to the intermediate thickener which generates a pregnant solution that is sent to precipitation and slurry, which is sent to a secondary leaching circuit comprised by 4 agitation stages. Finally, the slurry is sent to a backpressure washing process comprised by 4 thickening stages recovering valuable dissolved mineral. The underflow of the fourth thickening tank is pumped to the tailings dam. The first thickening tank s overflow is back fed to the grinding circuit to recycle water.

Precipitation:
The pregnant solution is filtered, clarified and subsequently vacuum fed to a de-oxygenation process. Once the solid particles and the air are removed, the pregnant solution is mixed with zinc powder and sent to the press filter for the gold and silver precipitation.

Dore Bar Production:
The precipitated gold and silver are mixed with different chemicals and are fed to the gas furnace for the production of Dore bars.

Tailings Disposal:
The slurry from the fourth thickening tank of the backpressure washing is distributed along the tailings dam s starting board so that the solid particles form a beach when they set and a clarified solution is obtained, which is then pumped to the leaching process

24.3. Environmental Considerations
The La Colorada Mine is within an area of historic mining activity. The mine was constructed and is operated using internationally recognized techniques and practices designed to minimize new environmental impacts.