

C.S.I. AG

PIEDRA AMARILLA PROPERTIES
REGION 3 - CHILE
TITANIUM, SULFUR, AND GOLD
PRE - FEASIBILITY UPDATE
PRELIMINARY DRAFT

PIEDRA AMARILLA MINERAL PROPERTIES PRE-FEASIBILITY UPDATE

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Chile - Location and Geography.

Chile is located on the west coast of southern South America, bordered by the Pacific Ocean on the west, Argentina and Bolivia on the east, and Peru on the north. (See Map 1.) Chile covers a geographical area slightly larger than the state of Texas. Because of its elongated north-south dimension, when traveling within Chile the impression is of a much larger geographical country. The average width from east to west is only about 110 miles, but the north-south dimension of 2,650 miles is equivalent to the distance between Washington, DC and the U.S. West Coast.

Chile is divided politically into 13 regions, including the Metropolitan Region of Santiago, which is not numbered like the other 12 regions, beginning with Region I at the northern border with Peru and continuing in sequence to Region XII at the southern end, with each region having a capital. This report deals largely with Region III, whose capital is Copiapo, outlined in the boxed area of Map 1.

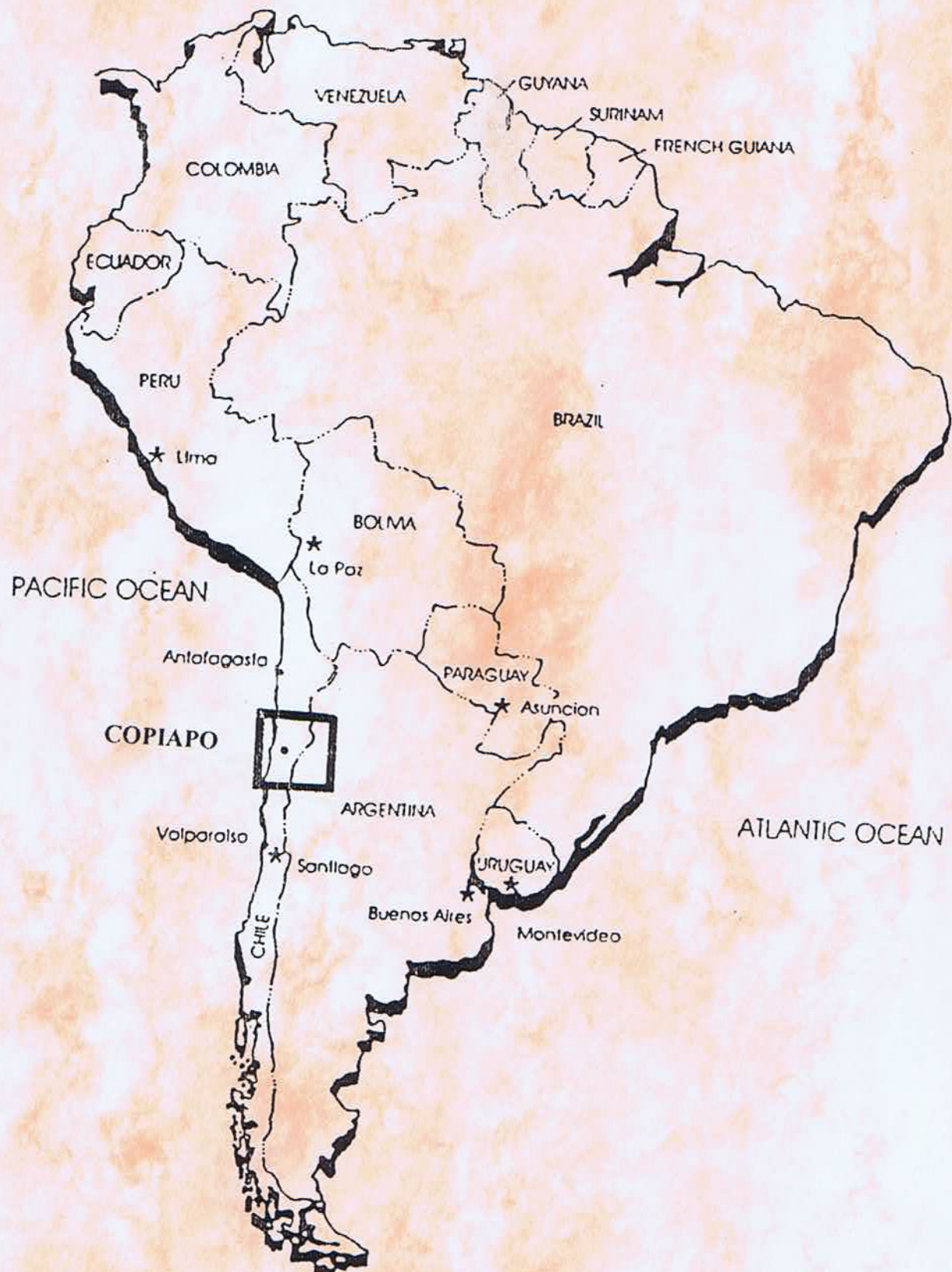
Chile has three main geographical areas which vary dramatically in climate, natural resources, and population. Each of these three areas are briefly outlined below:

1. The northern area of Region I beginning with the Peruvian border and continuing on through Regions II, III, and IV include the Atacama desert, one of the world's driest areas. Farming and agriculture are limited to a few valleys irrigated from streams originating high in the Andes. However, abundant and varied mineral resources are found in this area, including the world's largest reserves of copper, nitrates, lithium carbonate, and important resources of iron ore, gold, silver, molybdenum, sulfur, and a variety of commercial salts. Regions I - IV contain about 10 to 12 percent of the population of Chile.
2. From Region V continuing south through Region X is the Central Area, where more than 87% of the population resides. This area is composed of an elongated valley, known as the central valley, between the coastal mountains on the west and the Andes to the east. Near the northern end of the valley lies Santiago, Chile's capital and home to one third of the country's population. Important resources include agriculture, large copper deposits, coal, and hydropower from streams and rivers descending from the central Andes.
3. The southern area of Chile from Region XI to Region XII is one of the wettest and stormiest areas in the world. Less than 3% of the population resides here. Important resources include forestry, coal, and to a lesser extent natural gas, and petroleum.

The two dominant geographical features of Chile are the Pacific Ocean and the Andes mountain range. Chile ranks seventh in the world with more than 8,000 miles of coastline, compared to about 5,000 miles for the continental U.S. The cool Humboldt Ocean Current flows north from the Antarctic along Chile's entire western coast, helping to keep temperatures moderate in the central and northern sections of the country. The cool waters are an ideal breeding ground for a number of fish and seafood species, and as a result has spawned a large fishing industry which annually is one of the three or four largest in the world in terms of tonnage caught.

The Andes run the entire length of Chile's eastern border, with several peaks in the northern and central Chile reaching more than 20,000 feet above sea level. The tallest mountain in the Americas at more than 23,000 feet is Aconcagua, found along the border with Argentina, just northeast of Santiago.

Map 1. Location of Chile Relative to South America.



Chile's Region III

Chile's 3rd Region is located in the northern part of the country. (See Map 2.) The Atacama desert, which is one of the world's driest deserts, begins in southern Peru and ends in the northern portion of the 3rd Region. From about latitude 27° southward the desert gradually transitions to a more hospitable landscape. While the northern portion of the 3rd Region is generally devoid of most plant-life, the southern portion of the Region is reminiscent of the U.S. southwest deserts.

The 3rd Region is about three times as long from north to south as it is wide from east to west. The Region covers a total of more than 30,000 square miles, about the same size as South Carolina. The capital and largest city is Copiapo, located near the geographical center of the 3rd Region.

Politically, the Region, like all regions in Chile, is divided into provinces and then smaller communities or counties. The 3rd Region has three provinces, beginning in the north with Chanaral, Copiapo in the center, and Vallenar in the south, each of the provinces named after the provincial capital. (See Map 3.) The provinces are divided into a total of nine counties, each county having a county seat.

Each county has an assigned courthouse, one or more public notaries, and a mining conservator which is sometimes shared with more than one county, but always within the province. Mining claims in a geographical area are filed at the assigned courthouse for each of the corresponding counties. As the mining property undergoes various stages of constitution, copies are inscribed at the office of the mining conservator. Once the property becomes constituted, the Conservator records the deeded property on the real estate records.

Each of Chile's regions has an intendant or governor, who resides at the capital and coordinates with the various counties. Most of the larger cities have a mayoral system of government. The capital of each region also houses a court of appeals, and various regional government agencies such as education, justice, internal revenue, and so on.

Historical Background of Region III.

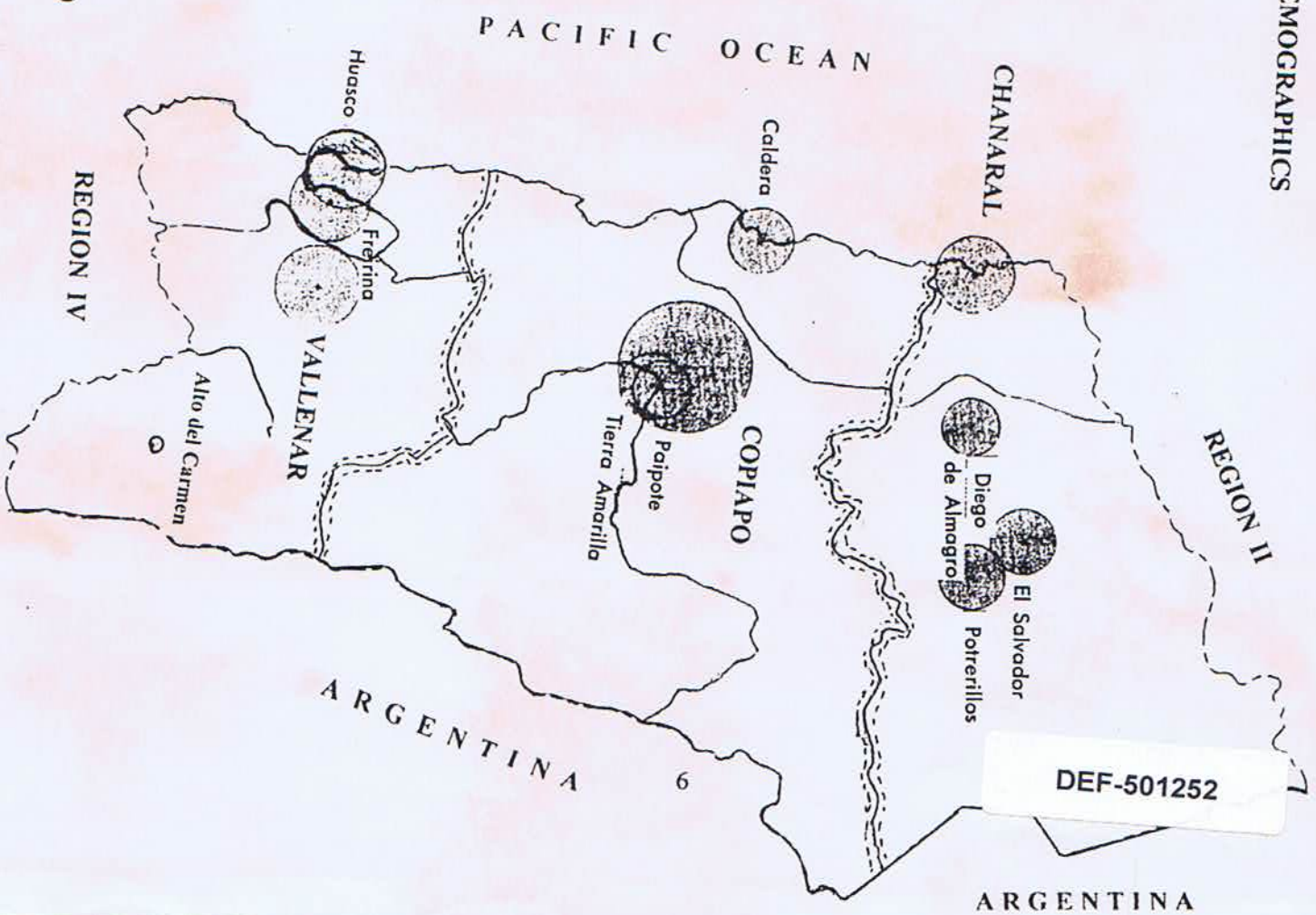
Although not founded until the mid 1740's, Copiapo was actually the site of the first camps of the Spaniards who entered Chile from Peru beginning in 1536, and subsequently served as a way-station for Spanish Conquistadors traveling from Lima to Santiago. The name "Copiapo" was derived by the Spanish from the local indigenous Indians, and means "Cup of Gold", referring to the numerous gold veins and workings in the nearby hills.

Although Copiapo's mining roots go back hundreds of years, it wasn't until 1832 that mining became a dominant economic force. In that year, a huge silver strike was found about 40 miles south at Chanarcillo, and the first large commercial mining operation in Chile was initiated. Chanarcillo would eventually become the world's third largest silver producer, netting sales through the 1870's of more than two hundred million dollars in silver bullion, which adjusted for inflation is equivalent to several billion dollars today.

The Chanarcillo mine attracted world wide attention and brought mining interests from Europe and other areas, and helped fuel industrial development in the Copiapo area. The first railroad in South

MAP 3. REGION III POLITICAL DIVISION AND DEMOGRAPHICS

<u>Provinces</u>	<u>Population</u>	<u>Counties</u>
Chanaral	52,000	Chanaral Diego de Almagro
Copiapo	175,000	Caldera Copiapo Tierra Amarilla
Vallenar	65,000	Huasco Freirina Vallenar Alto del Carmen
Region III Total Population : 292,000		



----- Provincial Boundaries

Scale 1 : 2,550,000

America was constructed during the 1850's to transport ore and silver bullion from the Chanarcillo mine to various processing plants and to the port of Caldera for shipment to European buyers.

Although the mine began to play out in the 1880's, a number of miners stayed in the area to work other subsequent strikes of gold and silver. A number of small copper operations were initiated prior to the turn of the century in the area, the most notable being the mines and smelter at Canto de Agua and Cerro Blanco, located about 60 miles south of Copiapo. The crude smelters produced a copper dore containing 4 - 5 lbs. per ton of gold and silver. The dore was shipped to U.S. buyers, principally Asarco, for further refining.

Most of the small mining operations in the Region ceased during World War I. This began a cycle of boom and bust which would continue on through the 1970's, wherein local mining activity expanded and contracted depending on world economic conditions and demand.

Towards the end of World War I, an American company, known as Andes Copper Company began exploration and development of a large porphyry copper deposit located about 100 miles northeast of Copiapo, near what is today known as Potrerillos. Port facilities were built in Chanaral, and an open pit mine engineered along with facilities for ore processing and smelting. A railroad was subsequently added for transporting smelted copper to port.

This was one of the first major construction projects undertaken in the world following construction of the Panama Canal in 1914 and initiation of the hostilities of World War I the same year. The scale of the mining project at Potrerillos, its relative remoteness and lack of infrastructure made this one of the great engineering feats in mining history, and is a credit to the "can do" attitude of American engineering in its infancy.

During the 1950's, a number of small copper projects were started in the Region, due in large part to strong copper prices prevalent during the decade. Many of these projects remain in operation today, and include the operations of Sali Hochschild, San Rafael, San Jose, and others. These projects have expanded and contracted along with world market conditions over the last 40 years.

The Chilean government agency, Enami, or Empresa Nacional de Minería, constructed a smelter and refinery during the latter part of the 1950's, at Paipote, some five miles south of Copiapo, to handle production of copper and gold concentrates from small and medium size producers, and to encourage development of other small mining operations.

Today, Enami buys from dozens of small mills and producers, some of which may produce only a few kilos a day of mineral concentrates. In addition, Enami built concentrating plants at El Salado, north of Copiapo, and at Domeyko, located in the southern portion of the Region to help foment mineral production from small and medium sized mining operations.

Recent Economical Developments.

Declining copper prices during the 1970's and early 1980's put the Copiapo area in a prolonged recession. In 1983, as part of an overall economic reform, Chile adopted a new mining law, which gave ownership guarantees to foreign companies and projects. By 1985, a number of the world's foremost mining companies were active in the Region, exploring and developing different projects.

More than one billion dollars has been spent in the last decade in the Copiapo area by international mining companies developing a variety of mineral prospects. A list of the most important international projects and companies is found in Table 1, and their locations are shown in Map 4.

State owned mining enterprises in the Region have also undergone modernization of existing facilities and infrastructure in the last decade. A list of the important State run mining operations and production capabilities is found in Table 2.

In 1995, the 3rd Region produced some 1.9 billion dollars worth of mineral commodities, almost all of which was exported to offshore buyers in Japan, Europe, and the United States. Of this amount, about 60%, or 1.2 billion dollars was copper. About one half of the copper is refined in the Region, at the smelters located at Potrerillos and Paipote; the balance is exported as concentrates. The remaining metal exports are as follows : gold - 450 million dollars; silver - 140 million dollars; and iron ore - 120 million dollars.

Metals and mineral concentrates are mostly exported from the ports of Chanaral and Caldera, while the iron ore is largely exported from the port of Huasco, located in the southern portion of the 3rd Region. These ports also serve to import fuel and supplies for the mining industry. The ports of Caldera and Huasco are also used to export fruit from the Region grown in the Vallenar and Copiapo river valleys. Fruit exports, largely to the United States and Japan, total millions of boxes per year, mostly as table grapes.

Regional Demographics and Infrastructure.

Growth of the mineral industry in the last twenty years, and to a lesser extent agricultural fruit exports and the coastal fishing industry, has caused Copiapo and the Region as a whole to double in population during the same period. The regional population for 1996 is estimated at 292,000 inhabitants. The greater Copiapo area has a current population of approximately 140,000 inhabitants. Table 4 shows gives the population statistics for the largest cities in the Region.

In order to assist the major industries in the Region, a number of equipment, supply and service related companies have opened offices in Copiapo in the recent past. The Chilean Chamber of Commerce estimates that each new mining job creates 3 new jobs in related services.

Communications. All forms of modern communications are available in Copiapo and most other areas of Chile. This includes phone service from national and international carriers, such as Bell South and VTR, cordless and cellular phone service, fax, Internet, E-mail, and so on.

Transportation. Copiapo has day-time only jet service from Santiago, with connections to Antofagasta, La Serena, and El Salvador. Three airlines provide 737 jet service, including National, Lan Chile, and Ladeco. Frequency varies from 15 to 20 flights per week to Santiago, 10 flights per week to La Serena, and 3 flights per week to Antofagasta and El Salvador.

In addition, a 19 passenger commuter airline provides twice daily service to Antofagasta and La Serena. Also, a number of inter-communal bus lines provide different levels of service to all points in Chile. Freight from Santiago is available overnight from a number of different trucking lines, and world-wide package service is provided by Federal Express, UPS, TNT, and others.

TABLE 1.
MAJOR FOREIGN MINING OPERATIONS - COPIAPO DISTRICT

<u>Location</u>	<u>Companies</u>	<u>Home Country</u>	<u>Minerals</u>	<u>Annual Production</u>
El Hueso	Homestake	United States	Gold Silver	75,000 troy oz. 300,000 troy oz.
La Coipa	TVX Placer Dome	Brazil Canada	Gold Silver	175,000 troy oz. 17,000,000 troy oz.
Can Can	COCAR Chevron	Various USA	Gold Silver	66,500 troy oz. 140,000 troy oz.
Coipa Norte	Niugini	Australia	Gold Silver	60,000 troy oz. 200,000 troy oz.
Guanaco	Amax Gold	United States	Gold Silver	80,000 troy oz. 1,000,000 troy oz.
La Pepa	Bridger Resources Cluff Mineral	Canada UK	Gold Silver	30,000 troy oz. 500,000 troy oz.
El Refugio	Bema Gold Amax Gold	Canada United States	Gold Silver	245,000 troy oz. 2,000,000 troy oz.
Ojos del Salado	Phelps Dodge	United States	Copper Gold Silver	75,000 metric tons 10,000 troy oz. 450,000 troy oz.
Candelaria	Phelps Dodge Sumitomo	United States Japan	Copper Gold Silver	125,000 metric tons 89,000 troy oz. 1,000,000 troy oz.
Manto Verde	Anglo American	South Africa	Copper	53,000 metric tons

Production Sub-Totals :

Gold 830,500 troy oz.
Silver 22,590,000 troy oz.
Copper 253,000 metric tons

TABLE 2.

MAJOR CHILEAN STATE OWNED MINING OPERATIONS - COPIAPO DISTRICT

<u>Location</u>	<u>Company</u>	<u>Minerals</u>	<u>Annual Production</u>
El Salvador	CODELCO	Copper	95,000 metric tons
		Gold	10,000 troy oz.
		Silver	350,000 troy oz.
		Molybdenum	5,000 metric tons
Paipote	ENAMI	Copper	80,000 metric tons
		Gold	188,000 troy oz.
		Silver	2,000,000 troy oz.
		Sulfuric Acid	60,000 metric tons
Salado / Domeyko	ENAMI	Copper	45,000 metric tons
		Gold	92,000 troy oz.
		Silver	1,000,000 troy oz.
Algarrobo	CAP	Iron ore	3,000,000 metric tons

Production Sub-Totals : **Gold** **290,000 troy oz.**
 Silver **3,350,000 troy oz.**
 Copper **220,000 metric tons**

Other : **Sulfuric Acid** **60,000 metric tons**
 Molybdenum **5,000 metric tons**
 Iron Ore **3,000,000 metric tons**

TABLE 3.

**TOTAL ANNUAL MINERAL PRODUCTION - REGION III
STATE OWNED AND FOREIGN OPERATIONS COMBINED**

Gold	1,120,500 troy oz.
Silver	25,940,000 troy oz.
Copper	473,000 metric tons
Sulfuric Acid	60,000 metric tons
Molybdenum	5,000 metric tons
Iron Ore	3,000,000 metric tons

TABLE 4.
CHILEAN COPPER PRODUCTION

<u>Company</u>	<u>Mines</u>	<u>Region</u>	<u>Annual Production</u>	<u>Type of Product</u>	<u>Refining Method</u>
CODELCO	Chuquicamata	II	600,000	Blister	Smelter
	El Teniente	VI	300,000	Blister	Smelter
	Andina	V	136,000	Blister	Smelter
	El Salvador	III	95,000	Blister	Smelter
	El Abra	II	225,000	Cathode	SX-EW
Exxon	Los Bronces	V	130,000	Blister	Smelter
	El Soldado	V	75,000	Concentrates	-
Anglo American	Elvira/M. Blancos	II	100,000	Concentrates	-
	Manto Verde	III	53,000	Cathode	SX-EW
	Collahuasi	I	250,000	Concentrates	
			50,000	Cathodes	SX-EW
Phelps Dodge	Candelaria	III	125,000	Concentrates	-
	Ojos del Salado	III	75,000	Concentrates	-
ENAMI	Paipote	III	80,000	Blister	Smelter
	Ventanas	V	120,000	Blister	Smelter
	El Salado/Domeyko	III	45,000	Precipitates	SX
RTZ	Escondida	II	320,000	Concentrates	-
			380,000	Cathode	SX-EW
	Cerro Colorado	I	45,000	Cathode	SX-EW
OutoKumpu	Lo Zaldívar	II	40,000	Cathode	SX-EW
Cominco	Quebrada Blanca	I	75,000	Cathode	SX-EW
Lucky Star	Los Pelambres	IV	22,000	Concentrates	-
RayRock	Ivan Zar	II	87,000	Cathode	SX-EW
Lince	Zaldívar	II	20,000	Cathode	SX-EW

Total Copper Production by 1997 : **3,448,000 metric tons**

Blister copper	1,461,000 metric tons
Copper in Concentrates	967,000 metric tons
SX-EW Cathode Copper	975,000 metric tons
Copper in Precipitates	45,000 metric tons

By Region :	I	420,000 metric tons
	II	1,772,000 metric tons
	III	473,000 metric tons
	IV	22,000 metric tons
	V	461,000 metric tons
	VI	300,000 metric tons

REGION III GEOLOGY

A regional geologic map on a scale of 1 : 1,000,000 covering the northern one-third of Region III is shown in figure RG-1 (Mercado, 1979). The major geological features run north-south, such as the Atacama fault system, morphological units consisting of granitic extrusions and batholiths, volcanic systems, and so on.

Elevation increases are from west to east, with a general decrease in the age of morphological units from west to east. Drainage from the coastal to pre-cordillera areas is from east to west, while the cordillera areas consist mostly of confined and closed drainage basins, with a large portion of the drainage running from south to north.

The plutonic magma orogen in this portion of the Andes is greatly influenced in the Pacific Ocean along the Atacama Trench, formed where the Pacific "Nazca" tectonic plate collides with the Continental or "South American" plate. The Nazca plate dives underneath the Continental plate, forming a very deep offshore trench, which reaches a depth of more than 8,000 meters, the deepest in all of the Americas. The Atacama Trench extends just offshore from Antofagasta to Caldera.

As a result of the tectonic collisions, the Continental plate has been lifted upward through constant block faulting. The plates grind laterally across one another, with the upward faulting of the Continental plate producing seismic activity. The suppression of the Nazca plate causes tremendous friction and pressures, causing portions of the orogen to melt into magmatic material.

Geographical Units.

The portion of the Chilean Andes from the Pacific Ocean to the Argentina border as shown in figure RG-1 can be subdivided into four distinct geographical units :

1. The coastal ranges, which parallel the coast and reach inland some 70 kilometers, from roughly 70° to 70° 45' W. These ranges may reach 1,500 meters in elevation, but are for the most part less than 1,000 meters above sea level.
2. The interior ranges, which are found between 69° 30' to 70° W. These ranges are below 3,000 meters in elevation in the northern portion of the region, but reach elevations of 4,500 meters at 27° S latitude.
3. The pre-cordillera, found between 69° and 69° 30' W. These ranges are generally between 3,000 to 4,000 meters in elevation, although the southern portion of the pre-cordillera at 27° S latitude has several peaks over 5,000 meters.
4. The cordillera, found between 68° and 69° W. The cordillera ranges are generally above 4,000 meters in elevation, with several volcanic peaks over 6,000 meters.

- Coastal Ranges.

The coastal ranges can be subdivided into four distinct geological units.

- a. The coastal mountains which are adjacent to the ocean consist of granitic extrusions and sediments, both of Paleozoic age, and which are interspersed along the coastline, having an approximate width of some 25 kms. Great blocks of granite and sediments have been uplifted by block faulting action related to the subduction of the Nazca plate. These blocks were

subsequently cut by erosion from storm runoff and drainage from the interior basins. These mountains fall precipitously towards the coastline, with some slopes greater than 50°. Few if any economic mineral deposits of any consequence have been located to date within these Paleozoic formations.

- b. Bordering the Paleozoic formations is a granitic extrusive body of upper Jurassic age, which forms the western boundary or edge of the Atacama fault system. The Jurassic system varies in width, from 10 to 20 kms. A number of copper deposits are found along the fault, characterized by a deep oxide zone and low gold values. Mineralization is generally confined to deposition within vein and dike channel-ways, with little dissemination within the granitic host-rock, although in some areas repeated intrusions of hydro-thermal fluid has caused formation of deep stocks.
- c. A granitic extrusive body of Cretaceous age forms the eastern boundary or edge of the Atacama fault system. This system is severely cut by volcanics of lower Cretaceous age (see d.) which were the catalysts for the introduction of the mineralized fluids along the fault system. Mineralized deposits along the eastern portion of the Atacama fault system are characterized by shallow oxide copper zone and higher gold values. The mines at Carrizalillo and Las Bombas are indicative of this type of mineralization.
- d. Volcanics of lower Cretaceous age, which intrude the granitic body of the same period and extend eastward some 30 kms. A number of silver bearing deposits were exploited during the 19th century in this area, including Chanarcillo. Within these volcanics were also found many of the iron ore deposits which were extensively exploited during the 1950's and 1960's.

- Interior Ranges.

The northern portion of the interior ranges are dominated by volcanics of upper Cretaceous to lower Tertiary age. These ranges are separated from the coastal ranges by a central basin covered with deep Tertiary sediments. The large copper porphyries at Potrerillos and El Salvador were found along contact zones related to these volcanics.

The southern portion of the interior ranges are dominated by volcanics of Jurassic age. Some minor gold deposits have been found and exploited within these volcanics near Inca del Oro.

- Pre-Cordillera.

The pre-cordillera is dominated by volcanics of middle to upper Tertiary age, interspersed and atop of marine sediments, largely metamorphosed, of Jurassic age. Some splendid gold deposits have been discovered along the contact zones between these formations, including the Coipa deposit, the El Hueso deposit, the Tinajas deposit, and the Esperanza deposit. Some large granitic blocks of Paleozoic age are found among the marine sediments.

- Cordillera.

The cordillera is dominated by volcanics of very late Tertiary to early Quaternary age. The western edge of these volcanics border volcanics of late Paleozoic to early Permian age. Granitic blocks of Paleozoic age appear in the southern portions of these formations. Only in the last few years have any extensive explorations been undertaken in the cordillera regions. The large gold

deposits at Marte, Lobo, Pantanillo, and Refugio have been found in these Quaternary volcanic formations.

A number of large, international mining companies have prospected areas of the cordillera, with some of them filing extensive mineral property claims. These include Cyprus-Amax, Phelps Dodge, Anglo American, Cominco, Goldfields, Placer Dome, Codelco, and others. The Chilean petroleum company Enap and a consortium of international companies extensively drilled the basin area around the Salar de Pedernales. Although showings of natural gas were found, they were apparently not of economic importance. The drilling logs of Enap would be of help in understanding the geological basement formations in the area.

Geology of the Piedra Parada Salar Basin

There are no detailed geological studies of the Piedra Parada Basin area. Available information such as that shown by Mercado is skeletal at best, and based on black and white aerial photography using scales of between 1 : 50,000 to 1 : 60,000.

Minexco Ltda., a Chilean company, carried out exploration activities around the Piedra Parada Salar Basin from 1988 - 1990. Minexco obtained color aerial photographs of the basin on a scale of 1 : 40,000. Senior geologist Carlos Ulriksen (M.Sci., Dalhousie University, Canada) created photo-geological maps on a scale of 1 : 40,000 covering much of the basin area. Aids used in the mapping project included :

- Landsat photographs, black and white, infrared, scale 1 : 250,000.
- 45 aerial photographs, black and white, scale 1 : 60,000.
- 78 aerial photographs, color, scale 1 : 40,000.
- El Salvador-Laguna Verde Maps, scale 1 : 250,000.
- Laguna Brava, Panteon de Aliste, Rio Juncalito, La Ola, Cerros Colorados, and Cerro Leon Muerto topographical maps, scale 1 : 50,000.

These geological aids were combined with a number of field trips by Ulriksen to the Piedra Parada basin area. No extensive field mapping was undertaken. Excerpts from a pre-feasibility study completed by Minexco in 1990 are included hereafter in Appendix 1. These detail some of the geological features of the Piedra Parada Basin area.

Mineral Properties.

C.S.I. Ag has been actively prospecting in the Copiapo area since 1986. Most of the exploration efforts have been centered in a remote area known as the Piedra Parada Salar, an enclosed drainage basin located some 250 kilometers northeast of Copiapo. A number of important mineral properties have been constituted, and are known collectively as the Piedra Amarilla properties, which means "yellow stone", and refers to the outcrops of native sulfur found on the mineral properties.

The Piedra Parada Salar is found at 26° 20' S latitude and 68° 45' W longitude. The salar is bordered by two large volcanic ranges to the south, the Cuyanos and Sierra Nevada ranges, which reach up to 6,000 meters in elevation.

The Claudio Gay range borders to the west, and consists of rhyolitic volcanics of late Permian age, the oldest volcanics in the 3rd region. The range is largely composed of remnants of two large caldera type features, wherein the volcanics either subsided or eroded into caldera structures.

The Piedra Amarilla properties form the eastern border of the Piedra Parada Basin. The properties lie only a few kilometers from the Argentina border.

Prior to the exploration activities by C.S.I., there were no constituted mineral properties in the Piedra Parada area. The Piedra Amarilla properties were initially claimed in August of 1987 and have been fully constituted since May of 1990. A number of large mining companies have claimed exploration properties in the general area, but none of these have been fully constituted. Some of these companies are Phelps Dodge, Anglo American, Goldfields, Codelco, and Cyprus Mines.

The first access roads near the Piedra Parada Salar were built by Andes Copper during the 1920's. A pumping station and aqueduct were constructed at La Ola, and water from the Rio Leoncito was pumped to Potrerillos for use in ore beneficiation. A modest recreational area was developed at the Rio Negro hot springs for use by mining personnel.

Andes Copper geologists studied the Cuyanos range in the late 1920's while searching for native sulfur. Old reports indicate blocked out sulfur reserves of 500,000 metric tons, grading 50% sulfur. Two adits were driven into the caliche, and a few hundred tons of high grade sulfur were extracted for use in copper refining. The adits and remnants of an old mining camp are still visible.

Codelco began exploration activities north of the Salar about 1985. An access road was built to the Panteon de Aliste area

and extended northward to the Azufrera Tres Puntas. Portions of the Tres Puntas formation were trenched, and Codelco claimed several properties from the Tres Puntas area eastward to the Agua de Morales formation.

In 1986, Codelco constructed a road partway up the southern half of the Cuyanos range, and initiated a drilling program. Codelco also drilled a small formation near the Agua de la Falda range just to the south of the Cuyanos range.

During the last part of 1987 and 1988, Anglo American carried out an extensive geo-chemical sampling program throughout the Piedra Parada Salar area. Several areas were identified as altered anomalies and were claimed as exploration properties.

Several claims were filed on the Carolina properties and portions of the Piedra Amarilla properties, but C.S.I. had filed much earlier and Anglo American had to abandon their claims. Anglo still has substantial exploration properties throughout the basin. As a courtesy to C.S.I., lab results from 196 samples covering much of the basin area were made available on an unofficial basis by certain Anglo exploration personnel.

A list of the constituted mineral properties owned by C.S.I. in the Piedra Parada area are listed below:

- | | |
|--|--|
| 1. Piedra Amarilla 41
300 hectares | 6. Piedra Amarilla 191
300 hectares |
| 2. Piedra Amarilla 71
288 hectares | 7. Piedra Amarilla 221
300 hectares |
| 3. Piedra Amarilla 101
300 hectares | 8. Piedra Amarilla 251
300 hectares |
| 4. Piedra Amarilla 131
300 hectares | 9. Piedra Amarilla 281
100 hectares |
| 5. Piedra Amarilla 161
300 hectares | |

The properties total some 2,488 hectares, or 6,195 acres. Properties 3 - 9 are contiguous and comprise about two thirds of the total land amount. Properties 4, 5, 7, and 8 have been extensively studied during the pre-feasibility stage and contain the bulk of the information discovered in the geological and metallurgical studies.

LOCAL GEOLOGY

There are no detailed geological studies of the Piedra Parada Basin area. Available information such as that shown by Mercado is skeletal, (figure RG-1) and based on black and white aerial photography taken by SAF (Servicio Aereo Fotografico), a branch of the Chilean Air Force. These photographs are generally on a scale of 1:50,000 - 1:60,000.

SAF produces high quality topographical maps, on scales of 1:250,000 and 1:50,000. Minexco senior geologist Carlos Ulriksen has mapped all of the geological features of the Piedra Parada Basin area on a scale of 1:50,000. Although the geological mapping is preliminary and lacks some of the detail of mapping done on bigger scales, such as 1:10,000, it was deemed sufficient for the needs of this pre-feasibility study.

Minexco was able to obtain color aerial photographs of the basin on a scale of 1:40,000. Photocopies of these aerial photos are shown in figures 4-7. These cover the Piedra Amarilla and Carolina mineral properties. Minexco has additional color aeriels which cover approximately 1500 km² of the basin area. The aeriels, which are available for study at Minexco's offices in Santiago, were the basis of the geological interpretations by Ulriksen. Aids used for geological interpretation include the following:

- Landsat photographs, black and white, infrared - scale 1:250,000
- 45 SAF aerial photographs, black and white - scale 1:60,000
- 78 aerial photographs, color - scale 1:40,000
- El Salvador and Laguna Verde maps - scale 1:250,000
- Laguna Brava, Panteon de Aliste, Rio Juncalito, La Ola, Cerros Colorados, and Cerro Leon Muerto maps - scale 1:50,000

Geological aids were combined with a number of field trips to the Piedra Parada basin area during 1988. The geological work done to date is only the first step and forms the basis of detailed field work which should be carried out during feasibility studies.

Volcanics.

The majority of the volcanics in the Piedra Parada area are of dacitic and of dacitic-andesitic composition, (intermediate chemistry) characterized by modest or low iron content, and dominant quartz silica content. Most of these volcanics are believed to have originated from a large granitic batholith of Paleozoic age. The batholith is exposed as occasional laccoliths along the entire cordillera of the 3rd Region, with a greater preponderance of exposed laccoliths found along the border with Argentina in the southern half of the Region.

The batholith in the Piedra Parada area and northward is believed to have undergone a process known as isostatic compensation, in which the lower portions of the batholith sank to sufficient depth to melt the components into mass magma. The melting of the lower portions of the batholith were aided by the friction and pressures created by the subduction of the Nazca plate under the continental plate.

During the late Tertiary and early Quaternary ages, the mass magma surfaced through deep faults and cracks, largely as ash, lapilli, and small volcanic blocks, and forming the volcanic structures found in the basin today. The Permian volcanics of the adjacent Claudio Gay range are believed to have formed in a similar fashion, but at a much earlier time period.

The lack of basaltic flows, which contain a higher iron content and predominate in some of the other areas of Chile, would seem to eliminate primary magma as the source of the Piedra Parada volcanics.

Age.

No rock dating has been done to determine the specific formative ages of the different volcanic episodes within the basin. Mercado (figure RG-1) shows a preponderance of Quaternary volcanics. Field observations by Ulriksen indicate that many of these structures are probably much older, from at least late Tertiary times.

Ulriksen constructed a relative time scale, found in figure 8, which shows different periods of volcanic activity relative to one another in the main property group of the Piedra Amarilla properties. As can be seen in the aerial photographs, many of the older features in the basin were changed or reworked by the continuous introduction over a substantial period of time of newer volcanics.

It is common to observe these newer volcanics partially covering older structures. A check with the Chilean National Mining Service and other records indicates that there are no active volcanoes in this area of Chile.

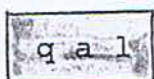
Volcanic types.

The volcanoes within the Piedra Parada Basin can be generally classified into two different categories, strato-volcanoes and cinder cones.

1. Strato-volcanoes are the dominant structures in terms of individual mass and height within the basin area. These consist of alternating layers of pyroclastic material and lava flows. The volcanic slopes are steep at the top, exceeding 30 degrees.

PIEDRA AMARILLA PROPERTIES

RELATIVE TIME SCALE



Quaternary Alluvium



Piedra Parada Salar
and Alluvial Scarps
of Interbedded Sediments

- Volcanics -

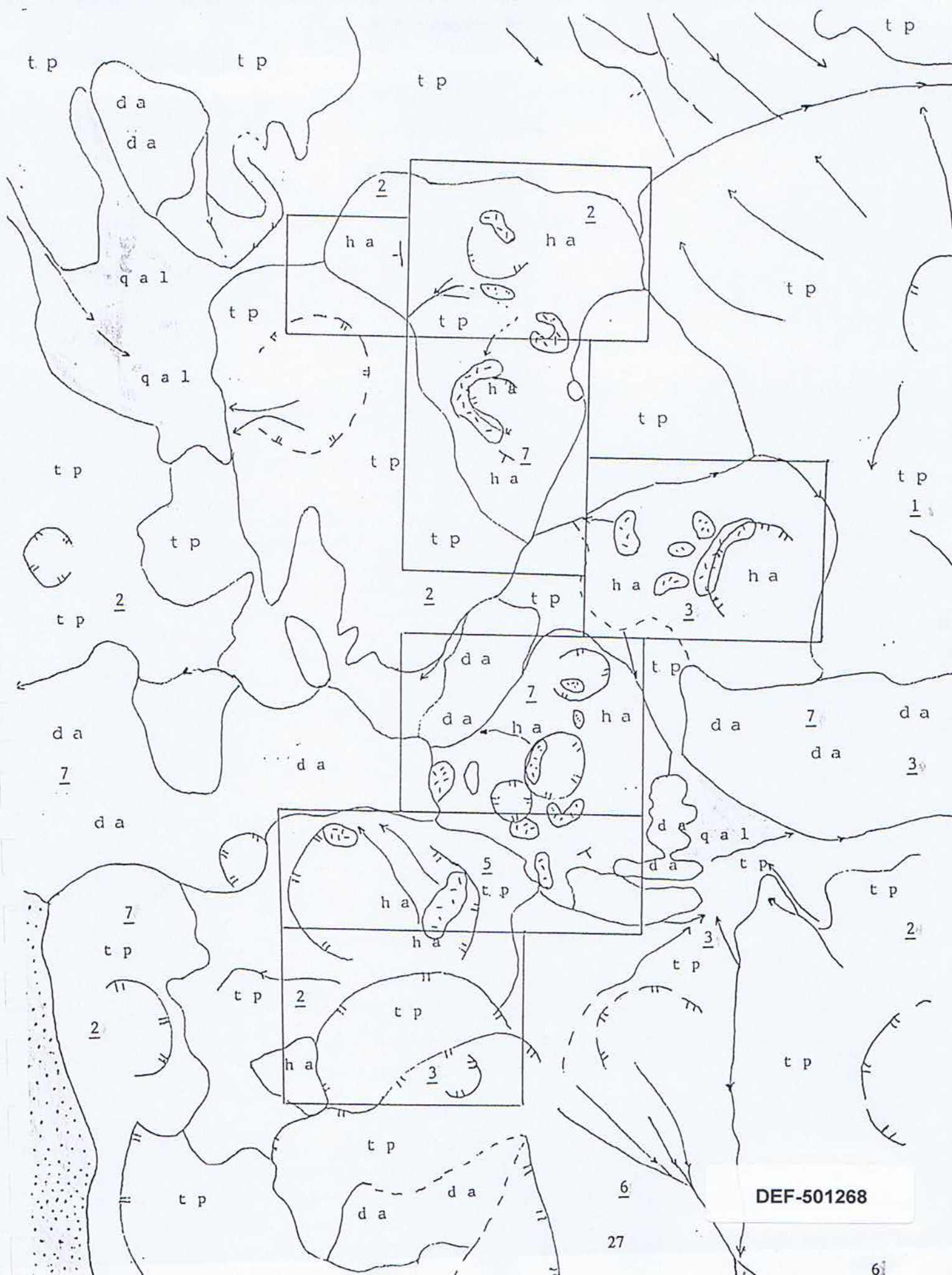
1 - Youngest Volcanic Formations

7 - Oldest Volcanic Formations

Inferred Age of volcanics -

Late Tertiary to Early and Middle Quaternary

Scale = 1:40,000. Geology by Carlos Ulriksen G. 1988



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Slopes at the bottom of strato-volcanoes are modest. The layered strata are indicative of cyclical changes in the nature of the eruptions. The strato-volcanoes found in the southern half of the basin, such as those which compose the Cuyanós and Sierra Nevada ranges, have a greater proportion of lava flows than the those found in the northern portions of the basin. The Panteón de Aliste and Cerros Colorados formations, for example, are more evenly built by alternating layers of pyroclastic ejecta and lava flows.

2. Cinder cone volcanoes are common throughout the middle areas of the basin, and are formed largely from tephra or ejecta in the form of ash, dust, lapilli, and larger volcanic blocks. These are blown out of the fissures of the volcanic conduit by high pressure gasses as pyroclastic material. Lava flows are rare or intermittent and comprise a much smaller portion of the total volcanic material.

The pyroclastic material has compacted into breccia tuffs near the volcanic cones or conduits, and into finer tuffs away from the cones. These tuffs are often layered, indicating cyclical deposition of pyroclastic material. The Piedra Amarilla properties largely consist of cinder cone volcanoes. A more thorough understanding of the formative volcanic processes is necessary in order to comprehend the mineralization of these formations.

The cinder cone volcanics which comprise the Piedra Amarilla properties have been chemically altered by the subsequent introduction of hydrothermal intrusions of gas and super-heated liquids. Although all three forms of matter - solid, liquid, and gas - are ejected from most volcanic structures, the nature of the formation produced by the volcano depends largely upon the proportion of each of these states of matter and the order in which they are expelled.

Piedra Amarilla Properties.

The Piedra Amarilla Properties are composed of a group of related volcanic cinder cones of late Tertiary age. The cinder cones were created largely by dacitic flows of ash, lapilli, and small volcanic blocks. These cinder cones sit atop a basement formation not well identified, but presumed to consist of rhyolitic flows from the Claudio Guy range and a combination of salar and marine sediments. Drilling logs from ENAP taken during exploration of the Salar de Pedernales might better identify the possible makeup of the basement formation.

During several separate periods of hydrothermal mineralization, super heated solutions, charged with steam and other gasses escaped from the congealing magma in enormous quantities, carrying with it mineral matter which was deposited at higher

levels where the pressure and temperature were less intense. Fissures and cracks in the dome formations provided channel ways for the movement of these solutions, and became the sites where deposition took place. The gas and liquid phases were intimately associated, but some predominantly gaseous activity occurred separately.

Water vapor as super-heated steam proves to be the main component of magmatic gasses. Magma typically contains up to 11% steam at great pressures. Other volcanic gasses include carbon dioxide, compounds of sulfur (sulfur dioxide and hydrogen sulfide), chlorine (ammonium chloride, hydrochloric acid), fluorine, and boron. Temperature of these gasses often exceeds 650o C.

The introduction of hydrothermal fluids is among the last expressions of volcanic activity. Large areas of the Piedra Amarilla properties have been altered by the introduction of these fluids, changing the texture and composition of the volcanic tuffs into caliche (a Chilean term).

Most of the original orthoclase feldspars have been decomposed into white argillous kaolinite, giving the silica formations a whiteness which is distinctive and helps to distinguish them from the surrounding country rock. Prospecting activities initially incorporated location of caliche formations in finding altered geological areas.

The host-rock for the Piedra Amarilla deposit is considered dacitic tending to rhyolitic in composition, consisting largely of ash flows and pyroclastic breccias compacted into tuffs. These tuffs have been hydrothermally altered by subsequent introduction of large amounts of gas, steam, and other liquids. The hydrothermal activity may have been related to the last episodes of the cinder cones, as well as to the many volcanic formations which were later introduced in the adjacent area.

The morphology is similar to that found at the Coipa formation, wherein acid volcanics are composed of altered breccia tuffs overlying a sedimentary bed.

Analysis of the host-rock indicate that silicates comprise the vast majority of pyroclastic material. Diffraction has identified quartz and its polymorphs, cristobolite and tridymite as the major host-rock components. The presence of cristobolite indicates a very hot formative environment as crystallization begins at a relatively high temperature (1470o C.).

A number of feldspar minerals are present, with the calcium-sodium plagioclase feldspars dominating the potassium orthoclase feldspars. These latter have been largely altered to kaolinite by intrusions of sulfur and carbonate solutions. The feldspars, which include alumina as a component, together with kaolinite

make up a composite of approximately 3.5% of the host-rock content as accessory silicates. The deposit is geologically classified as acid volcanics, meaning that the total silica content is greater than 65%. If elemental sulfur is excluded, the actual silica content of the Piedra Amarilla deposit is >90%.

Economic Minerals.

Mineralization deposited within the structures of the Piedra Amarilla properties has been identified and in many cases quantified using a variety of analytical means. These include atomic absorption, fire assay, x-ray fluorescence, wet chemical leaching analysis, and microscopic studies. Table 2 identifies typical mineralization found in the varying grades of caliche contained in the properties.

- Titanium -

Titanium is widely and abundantly distributed throughout the properties. Microscopic analysis has identified rutile (TiO_2) as the principal titanium mineral in the Piedra Amarilla deposit, accounting for >95% of all titanium present. Rutile is the most economically desirable of all titanium minerals because of the ease of its conversion into commercial grade pigment and metal.

Other titanium minerals present but accounting for less than 5% of titanium content include in order of importance ilmenite (FeTiO_3), and sphene (Si_5TiCa). TiO_2 content averages 3.20%.

The rutile mineralization is intimately associated with the silica content of the deposit, overlying the quartz crystals as clasts and elongated prismatic inclusions. It is presumed that the rutile is present as an accessory mineral of the silica host-rock. The titanium is distributed over the entire deposit, and grade distribution is too consistent to have developed from hydrothermal intrusions alone.

Particle size of the rutile generally varies from 80 μm to 20 μm , with a predominance of the finer sized particles. A small portion of the rutile mineralization averaging 5 μm is finely disseminated throughout the quartz host-rock and is probably uneconomical to recover. Much of the rutile may have formerly been sphene or ilmenite. Post intrusions of sulfur and carbonate solutions may have transformed most of this into rutile by releasing the calcium and iron as sulfates and carbonates.

Rutile and sphene are known to be accessory minerals of extrusive volcanics which are products of mass magma, especially acid volcanics which have cut through intrusive batholiths. On the other hand, primary magma contains principally sphene, ilmenite and titanium associated with biotite.

TABLE 2

33

<u>Mineral / Element</u>	<u>Content - Low Grade Caliche</u>	<u>Content - High Grade Caliche</u>
SiO ₂	75.10%	70.02%
S	10.50%	20.10%
TiO ₂	2.64%	2.74%
Al ₂ O ₃	0.85%	0.50%
H ₂ O (Moisture)	1.82%	1.65%
CaO	0.26%	0.19%
Na ₂ O	0.10%	0.06%
MgO	0.07%	0.05%
K ₂ O	0.07%	0.03%
Fe	0.32%	0.17%
Au	0.34 - 9.88 g/t	0.20 - 9.88 g/t
Ag	7.00 g/t	7.00 g/t
As	40 ppm	31 ppm
Se	20 ppm	31 ppm
Te	19 ppm	32 ppm
Bi	4 ppm	26 ppm
Cl	370 ppm	310 ppm
Cu	19 ppm	38 ppm
Cr	19 ppm	5 ppm
Mn	27 ppm	17 ppm
Mo	45 ppm	23 ppm
Pb	7 ppm	5 ppm
Sn	50 ppm	400 ppm
V	<45 ppm	45 ppm
Zn	9 ppm	0.4 ppm

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Rutile appears as secondary mineralization in deposits associated with primary magma, such as copper porphyries and ferrous deposits. Czamanske and others (1981) indicate that rutile mimics the distribution of the original magmatic titanium minerals. Secondary rutile mineralization reaches its greatest abundance and grain size in the biotite potassium feldspar alteration zone of porphyries in the Western United States. In peripheral alteration zones, rutile abundance and grain size progressively diminish. In many porphyries, the distribution of rutile and of copper ore is about the same.

Table 3 below lists some titanium bearing deposits in Chile and elsewhere.

Table 3A.

<u>Deposit</u>	<u>Location</u>	<u>Type</u>	<u>% TiO₂</u>
San Manuel	Arizona	Copper Porphyry	0.75%
Tangse	Sumatra, Indonesia	Copper Porphyry	0.30%
Tacora	1st Region, Chile	Acid Volcanics	0.80%
Purico	1st Region, Chile	Acid Volcanics	0.20%
La Coipa	3rd Region, Chile	Acid Volcanics	1.00%
Volcan de Copiapo	" "	Acid Volcanics	1.30%
Algarrobo	3rd Region, Chile	Iron Deposit	1.20%
El Teniente	6th Region, Chile	Copper Porphyry	0.80 %
PIEDRA AMARILLA	3RD REGION	ACID VOLCANICS	3.20%

- Sulfur -

Deposition of elemental sulfur occurs as a result of hydrogen sulfide and sulfur dioxide gasses combining under pressure. The reaction precipitates the sulfur with water (steam) given off as a byproduct. Unlike the titanium mineralization, the sulfur is found as localized deposits, following the fissures and channel ways of the volcanic domes, creating veins that give the mineralization a style reminiscent of pegmatite or dike formations. Channel ways of nearly pure sulfur can be found, ranging from a few centimeters to over 4 meters in width. The mineralization generally penetrates into the porous areas of the caliche, with a corresponding decrease in grade as one moves away

from the main channel ways. Deposition of sulfur is among the last expressions of volcanic activity. The process may be multi-stage, occurring over several different gaseous periods. Some of the gaseous activity could have resulted during the last stages of the introduction of newer volcanics in the adjacent area.

Caliche deposits are thought to be generally parallel to sub-parallel to the surrounding terrain, having been formed in mushroom fashion with the roots extending into the volcanic conduit. The caliche deposits thin with distance away from the cones.

- Gold and Tellurium -

The principal source of the world's gold mineralization occurs as a result of hydrothermal deposition within intrusive and extrusive igneous rock formations. Most of the major gold reserves are found in sediments or placers or as disseminations adjacent to weathered igneous formation. Gold is common in distinctive types of volcanic rocks, often associated in tuffs with manganese or altered quartz minerals.

Primary gold mineralization in the Piedra Amarilla deposit is assumed to have taken place during the liquid phases of hydrothermal activity. Gold is found in the native state as well as a compound of tellurium abundant in the deposit. Native gold has been assayed at 0.2 - 0.34 grams per ton in the near surface areas.

Gold values associated with telluride mineralization have been tentatively identified at 4.50 grams per ton Au. The telluride mineralization is thought to consist of krennerite (Au,Ag)Te₂ with a minor proportion of calaverite (AuTe₂).

Tellurium is one of the rarest of all minerals, having an average crustal abundance of about 2 parts per billion. Only the radioactive series of elements and rhenium are less abundant. Telluride gold deposits are nearly as scarce. There are thought to be two main geological environments in which gold and silver tellurides occur.

The first is in pre-cambrian rocks, or metamorphosed volcanic lava, such as at Kalgoorlie, Australia. In this deposit, native tellurium is rare, native gold is abundant, and tellurides of Hg, Cu, and Bi are generally present. The stable form of AuTe₂ is clearly calaverite, not krennerite, and the telluride minerals lack crystalline form.

The second type of geological environment for gold and silver tellurides is similar to that found at Piedra Amarilla. Veins, fissures and breccia pipes formed in tertiary rocks, such as in Cripple Creek, Colorado; Emperor, Fiji; and the Carpathian

mountains of Russia are the present day models for this type of mineralization. In this type of occurrence, veins are vuggy and composed of quartz and carbonate minerals. Adjacent to the veins there is intense alteration of the host-rock due to the introduction of large amounts of hydrothermal fluids, carbon dioxide, and sulfur. Native tellurium is more abundant and native gold is rare. Tellurides of metals other than gold are present in lesser amounts. Krennerite is dominant over calaverite.

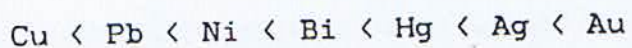
Tellurium content in the caliche has been determined at 19 part per million for lower grade caliche and 32 parts per million for medium grade caliche. The selenium content is almost identical, unlike other sulfide deposits where selenium is much more abundant. Though tellurium and selenium are closely related chemically, they behave differently in ore formation. In the subgroup of S-Se-Te, tellurium forms the largest and hence most polarized ions.

The size of a tellurium atom or ion and its polarization properties determine the formation of the tellurides of silver, gold, and other elements. Bond forming properties grow stronger from S-Se-Te. Thus, selenium forms independent minerals under special conditions only. For the most part, it enters the isomorphous lattice of the sulfides.

The chemical and metalloid properties of tellurium determine the structure of the tellurides. The solubility and volatility of the compounds of tellurium determine its paragenesis. This is expressed in the almost complete absence of selenium and tellurium in higher temperature sulfide deposits, and in the enrichment with tellurium of the lower temperature minerals found in epithermal deposits such as Piedra Amarilla.

Most of the tellurium crystallizes out during the epithermal stage. While sulfur passes easily into a hexavalent state and can form sulfates in the oxidized zone, tellurium becomes tetravalent with ease and forms tellurides.

It has been concluded from examination of tellurium minerals in deposits, that metal affinity to tellurium increases in the series :



Hence, gold tellurides occur in the presence of the metal sulfides of this series. Comparison shows that the above series agrees fairly closely with the series of decreasing polarization:



A number of problems have been encountered in utilizing standard atomic absorption and fire assay methods to quantify the exact amount of gold per ton contained in the tellurides. The compounds have a low volatilization point and the gold is not readily reduced using standard assay procedures.

Presence of the tellurides and their quantification was discovered using x-ray fluorescence procedures. Confirmation was attempted in part using standard leaching procedures. Leaching results of 1.75 grams per ton were obtained in one of the tests, corresponding to about 18% of the gold values assayed using x-ray fluorescence. Oxidation of the telluride mineral prior to leaching will enable a larger percentage recovery of gold content. A number of metallurgical steps will need to be accomplished during the feasibility studies.

- Silver -

Exact silver content of the Piedra Amarilla deposit is yet to be quantified, but is shown preliminarily as a minimum of 7 grams per ton, based on leach extraction tests. Microscopic tests revealed existence of some native silver. Precipitates of silver have been obtained in laboratory tests. When performing some gravitational tests, a compound similar to silver chloride was obtained in the concentrates. More diffraction work will have to be done to determine the exact makeup of the silver mineralization.

Some silver is undoubtedly associated with the telluride mineralization, in the form of hessite (Ag_2Te) and with the krennerite (Au,AgTe_2).

- Other Minerals -

Other minerals which can possibly be exploited include silica, kaolinite, and possibly alunite.

Extraction of sulfur and titanium will leave a fairly high purity silica tailings. Some of these can possibly be commercialized in the local economy. The refinery at Potrerillos, for example, must buy large quantities of low grade quartz ore from local gold miners to use as flux in their smelting operation.

The kaolinite can be separated from the gangue material either prior to or after flotation by simple decantation.

Some alunite appears to be present in parts of the deposit. If treated with sulfuric acid, a useful fertilizer of potassium sulfate could be manufactured and sold in the local agricultural market.

ORE RESERVES

Ore reserves have been quantified in three different mineralized sections within the core of the Piedra Amarilla property group. These cover portions of Piedra Amarilla 131, 161, 221, and 251. Reserves are quantified as both indicated and inferred, according to the Mineral Resource Classification System of the U.S. Bureau of Mines (see Table 6).

A reserve exploration program was outlined in April, 1988 in conjunction with the R.M. Parsons engineering company of Pasadena, California, who was commissioned to execute a pre-feasibility sulfur study on the mineral properties. Field exploration was carried out by Minexco in June and early July of 1988.

Ore reserves in Sections 2 and 3 were quantified by trenching predetermined areas associated with caliche mineralization. Section 1 was quantified by digging shallow pits for removal of bulk samples, and by sampling large outcrops which are exposed as hardened tuffs.

Trenched areas were sampled along 15 meter horizontal channels in both the mid-walls and floors. Some 15 tons of samples were removed from Sections 2 and 3, and some 5 tons from Section 1.

Sulfur reserves as quantified in the R.M. Parsons report were determined by using specified cut-off grades. In adding TiO₂ rutile and gold telluride mineralization, the reserves in this report use no cut-off grades because of poor grade correlation between minerals.

Trench and pit locations are shown in Table 7. Figures 14 and 15 show trench and pit schematics. Each of the properties was subdivided into 100 meter x 100 meter sections (one hectare) and sample results plotted and given a preponderance or area of influence of approximately 100 meters.

Depth dimensions of inferred and indicated ore reserves were conservatively determined using local geological characteristics such as outcrops and exposed caliche.

Although the sample base is sufficient for pre-feasibility purposes, it is inadequate for determination of measured reserves. Additional trenching and possibly some drilling will be required in conjunction with a detailed geological study during execution of the feasibility study.

A larger sample base and detailed geology will permit reserves to move into the "measured" category, and will undoubtedly enlarge the reserve base by uncovering additional reserves in the adjacent areas.

TABLE 6

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MINERAL RESOURCE CLASSIFICATION SYSTEMS

U.S. BUREAU OF MINES

- Measured - Reserves or resources for which tonnage is computed from dimensions revealed in outcrops, trenches, workings and drill holes and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurement are spaced so closely and the geologic character is so well defined that size, shape and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits which are stated, and no such limit is judged to be different from the computed tonnage ore grade by more than twenty percent.

- Indicated - Reserves or resources for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to permit the mineral bodies to be outlined completely or the grade established throughout.

- Inferred - Reserves or resources for which quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any samples or measurements. The estimates are based on an assumed continuity or repetition, of which there is geologic evidence; this evidence may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific geological evidence of their presence. Estimates of inferred reserves or resources should include a statement of the specific limits within which the inferred material may lie.

TABLE 7 - PIEDRA AMARILLA TRENCH LOCATIONS

<u>Trench</u>	<u>UTM Location</u>	<u>Trend</u>	<u>Length x Depth</u>
1	7,088,125 N 533,400 E	N x S	75m x 2m
2	7,088,125 N 533,350 E	E x W	60m x 1m
3	7,088,135 N 533,370 E	E x W	35m x 2m
4	7,088,050 N 533,250 E	NW x SE	50m x 1m
5	7,088,050 N 533,250 E	SW X NE	50m x 1m
6	7,088,500 N 532,875 E	N x S	50m x 2m
7	7,088,500 N 532,850 E	NW x SE	50m x 1m
8	7,088,250 N 532,600 E	NW x SE	50m x 2.5m
9	7,088,230 N 532,375 E	W x E	35m x 1m
10	7,088,400 N 533,100 E	N x S	35m x 1m
11	7,088,350 N 533,100 E	E x W	60m x 2m
12	7,088,625 N 533,150 E	E x W	40m x 1.5m
13	7,089,100 N 533,750 E	E x W	30m x 1.5m
14	7,087,400 N 532,400 E	N x S	100m x 2.5m
15	7,088,000 N 533,000 E	N x S	30m x 1m
16	7,088,000 N 533,100 E	N x S	30m x 1m
17	7,087,875 N 533,450 E	E x W	30m x 1.5m

TABLE 7 Continued
Location of Sampling Pits

<u>Pit #</u>	<u>UTM North</u>	<u>UTM East</u>
1	7,092.650	531.700
2	7,092.875	532.475
3	7,093.000	532.600
4	7,092.850	532.625
5	7,092.650	532.875
6	7,092.600	533.550
7	7,092.300	532.550
8	7,092.175	532.850
9	7,092.025	532.800
10	7,091.825	533.150
11	7,091.759	532.725
12	7,091.500	532.775
13	7,091.250	532.625
14	7,091.475	532.900
15	7,087.425	532.375

Bulk Samples.

More than twenty tons of samples were extracted from the pits, trenches, and outcrops for assay and metallurgical work. This included extraction of three groups of bulk samples containing 8, 6, and 4 tons respectively.

Sample extraction for assay work was supervised by Minexco senior geologist Carlos Ulriksen. Extraction of bulk samples was executed by a crew from Jorquera Limitada, a mine engineering firm specializing in topographic work from Copiapo.

Areas for extraction of bulk samples were identified by personnel from Intec-Chile during a site visit subsequent to the trenching program. Fourteen of the eighteen tons of bulk samples were sent to the Intec laboratories in Santiago, with the other four tons going to the Marambio laboratory in Copiapo.

Sulfur Reserves.

Blocked out sulfur reserves include indicated reserves of 21,120,000 metric tons grading 19.84% sulfur, and inferred reserves of 79,760,000 metric tons grading 10.66% sulfur (see figures 16-17, table 8). The grades correspond closely to the composite bulk samples assayed by Intec-Chile of medium grade and low grade caliche, containing 20.5% and 10.5% sulfur respectively (table 8).

In projecting sulfur grades into inferred mineralized areas, sulfur values were given as approximately 50% of the values contained in the adjacent indicated reserves.

Some of the trenches and pits failed to reach below the alluvial material and thus have low or no sulfur values. The Parsons pre-feasibility report originally discarded these samples because they fell below the predetermined cut-off grade. In adding titanium and gold values, however, these areas have been added back because of a poor correlation between the minerals.

For example, the highest gold values occur in the alluvial trenches, which should be expected because of artificial gravitational concentration through the erosional processes. In adding these additional reserves, the total reserves have been increased nearly four-fold from the original Parsons report to more than one hundred million tons. The downside is that the overall grade has dropped to 12.58% sulfur.

Because there are no major producers of native sulfur, Parsons, in designing the pre-feasibility study, had assumed that sulfur grades would probably have to be a minimum of 30% in order to make the project economically attractive. Based on blocked out reserves averaging 32% sulfur, a production price of \$35 per

metric ton F.O.B. was achieved, which would make the project very low cost compared to frasch producers in the U.S. and Mexico.

An extensive metallurgical study was undertaken by Intec-Chile and Corfo on the medium grade caliche (20% sulfur) and low grade caliche (10% sulfur) found at the Piedra Amarilla properties. Intec designed a new method to upgrade poor sulfur caliches through use of a flash flotation technique which permits rapid evacuation of a rougher concentrate in a large volume cylindrical flotation cell, which allows for processing of much larger ore tonnages in relatively short periods of time.

Because sulfur is naturally hydrophobic, one of the problems in designing a flotation circuit for higher grade sulfur ores is that the flotation kinetics don't permit a sufficiently rapid evacuation of the rougher concentrate. This allows some of the gangue material to continue on into the cleaner flotation circuit.

The flash flotation method developed by Intec allow grades of sulfur of 10% or even less to be processed for about the same cost of sulfur ores exceeding 30%. Since sulfur has now largely become a byproduct, the additional mining and crushing costs involved in mining lower grades are amortized among the various recovered mineral products.

Depth of Reserves.

In projecting the depth of the mineralized areas, a depth of 12 meters was chosen. This is based on the outcrops in area 1 which in some areas exceeded six meters, and on the depth of the trenches which was two meters in many areas. In studying exploration results of some of the volcanic sulfur deposits in Chile's Region 1, it was shown that many areas had layered depositions of native sulfur reaching as much as 50 meters in depth.

A drilling program on the Piedra Amarilla property group may show that sulfur beds lie beneath many of the alluvial areas. Trenching was hampered in part by ground ice which was prevalent in some areas. The trenching program was carried out in July, the coldest month in the Cordillera. The only snowfall occurred at the beginning of May, and the subsequent melting allowed for formation of ground ice during the cold months of June and July.

It should be noted that the sulfur reserves on the Piedra Amarilla 41 property have not been sampled nor blocked out, but based on visual evidence the aggregate reserves surpass all of the other Piedra Amarilla properties in both quantity and grade. The property lies an additional 1,000 meters in elevation, and pending road construction presently has very difficult access.

TABLE 8 - Sulfur Samples and Distribution

<u>Trench & Sample #</u>	<u>Sulfur %</u>	<u>As ppm</u>	<u>Se ppm</u>	<u>Te ppm</u>
1 - 001	9.6	1.6	8.2	1.8
1 - 002	26.9	0.8	65.0	30.0
1 - 003	24.7	1.0	37.0	20.0
2 - 004	3.6	3.1	4.9	0.5
2 - 005	2.8	2.5	2.9	0.2
2 - 006	42.2	0.9	52.0	8.4
3 - 007	24.5	2.8	48.0	6.4
3 - 008	22.0	3.8	34.0	5.5
3 - 009	17.7	1.0	3.4	1.1
4 - 010	3.6	3.1	4.9	0.5
5 - 011	2.8	2.5	2.9	0.2
6 - 012	15.1	0.6	2.7	0.9
6 - 013	14.2	2.3	3.3	1.0
6 - 014	29.8	-	-	-
7 - 015	17.6	0.8	3.5	1.2
7 - 016	4.5	2.7	2.3	0.8
8 - 017	32.3	0.8	0.3	0.2
8 - 018	26.2	-	-	-
9 - 019	4.2	4.2	25.0	0.4
10 - 020	0.0	1.2	0.1	0.1
11 - 021	0.2	2.1	0.2	<0.1
12 - 022	6.2	1.2	0.2	<0.1
13 - 023	0.0	1.9	0.1	<0.1
14 - 024	0.7	2.5	0.2	0.1
14 - 025	91.5	2.0	0.3	0.1
14 - 026	10.4	4.0	0.2	<0.1
14 - 027	21.0	4.0	0.4	<0.1
14 - 028	0.0	1.7	0.6	<0.1
14 - 029	0.0	1.5	0.3	<0.1
15 - 030	15.0	7.1	25.0	2.9
15 - 031	14.7	-	-	-

TABLE 8 Continued - Sulfur Samples and Distribution

<u>Trench & Sample #</u>	<u>Sulfur %</u>	<u>As ppm</u>	<u>Se ppm</u>	<u>Te ppm</u>
16 - 032	0.0	38.0	1.4	0.4
16 - 033	0.0	32.0	1.0	0.4
17 - 034	38.9	17.0	76.0	46.0
17 - 035	0.2	19.0	0.4	<0.1
17 - 036	0.3	17.0	1.7	<0.1

FIT SAMPLES

<u>Pit & Sample #</u>	<u>Sulfur %</u>	<u>As ppm</u>	<u>Se ppm</u>	<u>Te ppm</u>
1 - 037	0.4	25.0	0.2	<0.1
2 - 038	33.4	0.1	67.0	22.0
3 - 039	34.8	<0.1	67.0	11.0
4 - 040	12.6	-	-	-
5 - 041	26.8	-	-	-
6 - 042	2.2	23.0	0.4	<0.1
7 - 043	0.81	-	-	-
8 - 044	33.4	35.0	20.0	35.0
9 - 045	33.5	1.7	2.9	0.4
10 - 046	6.7	66.0	0.4	<0.1
11 - 047	47.4	-	-	-
12 - 048	36.5	3.0	4.1	0.5
13 - 049	47.5	-	-	-
14 - 050	37.1	2.6	3.3	0.3
15 - 051	31.01	-	-	-

TABLE 8 Continued - Sulfur Samples and Distribution

BULK SAMPLES

<u>Bulk Sample #</u>	<u>Sulfur %</u>	<u>As ppm</u>	<u>Se ppm</u>	<u>Te ppm</u>
INTEC - 1 6,000 kilos	10.5%	40.0	32.0	19.0
INTEC - 2 8,000 kilos	20.5%	31.0	31.0	32.0
Marambio - 1 4,000 kilos	31.0%	-	-	-

* samples with - signifies element not assayed in that sample

Figure 16

Sulfur Ore Reserves - Section 1

Legend.

0	-	5%	Green
5	-	10%	Violet
10	-	15%	Red
15	-	20%	Orange
>		20%	Yellow
.			Pit Sample Locations

✓ Indicated Reserves - Bounded Pink Line

Indicated Ore Tonnage - Section 1

	<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
1.	1.14%	1,440,000	16,360
2.	6.70%	480,000	32,160
3.	12.60%	480,000	60,480
4.	37.94%	3,840,000	1,456,800
	=====	=====	=====
	25.10% (Ave.)	6,240,000	1,565,800

Inferred Ore Tonnage - Section 1

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
12.55%	28,320,000	3,554,160

Total Inferred and Indicated Ore - Section 1

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
14.81%	34,560,000	5,119,960

7,093.500

500

555.000

500

7,093.000

P.A. 281

P.A. 251

7,092.000

7,091.500

7,091.000

P.A. 221

PA 19

Figure 17

Sulfur Ore Reserves - Sections 2 and 3

Legend.

0	-	5%	Green
5	-	10%	Violet
10	-	15%	Red
15	-	20%	Orange
>		20%	Yellow

Pit Sample Locations

/ Indicated Reserves - Bounded Pink Line

Indicated Ore Tonnage - Section 2

	<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
1.	0.73%	2,160,000	15,840
2.	6.20%	480,000	29,760
3.	14.00%	960,000	134,400
4.	18.35%	3,120,000	572,400
5.	29.30%	960,000	281,280
	=====	=====	=====
	13.46%	7,680,000	1,033,680

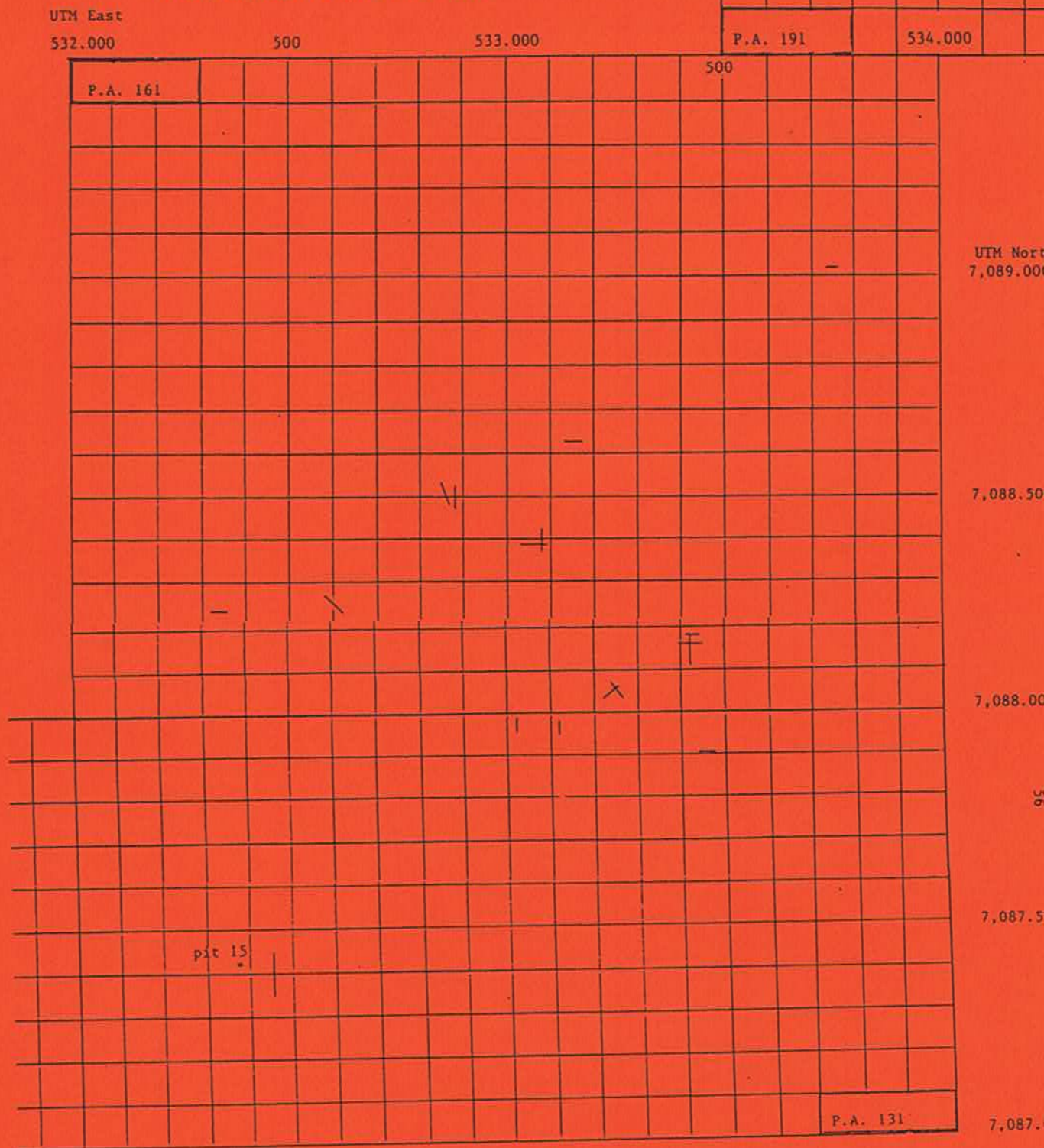
Inferred Ore Tonnage - Section 2

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
6.73%	17,040,000	1,146,792

Total Inferred and Indicated Ore - Section 2

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
8.82%	24,720,000	2,180,472

PIEDRA AMARILLA TRENCHING SCHEMATIC



Sulfur Ore Reserves - Sections 2 and 3 continued

Indicated Ore Tonnage - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
22.09%	7,200,000	1,590,480

Inferred Ore Tonnage - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
11.05%	34,400,000	3,801,200

Total Inferred and Indicated Ore - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
12.96%	41,600,000	5,391,680

Total Indicated Sulfur Reserves - Section 1, 2, and 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
19.84%	21,120,000	4,189,960

Total Inferred Sulfur Reserves - Sections 1, 2, and 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
10.66%	79,760,000	8,502,152

Total Indicated and Inferred Sulfur Reserves - Sections 1, 2, 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net Sulfur in Metric Tons</u>
12.58%	100,880,000	12,692,112

Sulfur Byproducts.

The field samples submitted for laboratory analysis were examined not only for sulfur but also for arsenic, selenium, and tellurium, which are elements that sometimes accompany native sulfur and need to be removed during the refining process.

Arsenic.

The average field sample had less than five parts per million (ppm) arsenic. This corresponded poorly with the analysis done on the large bulk samples, which assayed 31 ppm in the low grade caliche and 40 ppm in the medium grade caliche.

The arsenic content of the large bulk samples more closely agrees with the results of geochemical studies done by Anglo American in the Piedra Parada Basin, which showed that the area to the west of the Piedra Amarilla Properties had an average arsenic content of 90 ppm based on 100 samples. The area to the south of the main group of the Piedra Amarilla properties had an average arsenic content of 48 ppm based on 96 samples. And section three (see figure 17) of the Piedra Amarilla Properties had an average arsenic content of 27 ppm based on 12 samples.

Prior to the metallurgical studies, it was assumed that most of the arsenic mineralization was probably related to the sulfur mineralization, and that the arsenic would be recovered together with the sulfur during flotation. Laboratory analysis has since shown that the arsenic is closely related to the gangue material, and thus it is eliminated during the flotation steps and during autoclave refining.

Assay of final autoclave product prior to filtration shows an arsenic content of less than 1 part per one hundred million. Thus, arsenic is not recovered during any of the metallurgical steps used during the recovery of elemental sulfur.

Selenium.

Selenium content of the field samples varied from less than 1 ppm in section 3 to more than 20 ppm in section 2. The bulk samples show selenium content averages 30 ppm in the medium grade caliche and 20 ppm in the low grade caliche. Because of its extreme volatility selenium is difficult to assay using standard atomic absorption flame; thus the exact selenium content of the deposit may be somewhat higher.

Selenium is sulfated at low temperatures using sulfuric acid. Therefore most of the selenium minerals will be reduced to metallic selenium during autoclave refining of sulfur because of the slightly acidic nature of the autoclave charge. Analysis of the autoclave product prior to filtration shows that all of the

selenium contained in the head-ore is transferred into the final autoclave product, with a content of approximately 168 ppm. Considering that much of the selenium is probably lost in the gangue tailings and also to the autoclave agglomerate and bleed, the actual selenium content is probably much higher than is being assayed.

Based on assay of post-autoclave sulfur, the amount of metallic selenium contained in the sulfur ore reserves and available for recovery during final filtration of the molten sulfur product is 3,081.4 metric tons. Selenium which escapes final filtration is captured during sulfuric acid manufacturing in the dusts and sludges.

The selenium content in sulfuric acid sludges of different acid plants worldwide varies from 0.9% to more than 60%.

Tellurium.

Tellurium content of the field samples varied from less than 0.1 ppm in section 3 to more than 6 ppm in section 2. This corresponds poorly to the bulk samples which show tellurium content of 19 ppm in low grade caliche and 32 ppm in the medium grade caliche. As in the case of selenium, the extreme volatility of telluride compounds make it difficult to assay the exact tellurium content, and thus the actual content is probably higher than that shown in the bulk samples.

It is probable that the telluride compounds, like the selenium compounds, will partially sulfate during autoclaving. Analysis of the autoclave product prior to filtration shows that approximately 25% of the tellurium assayed in the head-ore passes into the final autoclave product, with a content of approximately 39 ppm.

Based on the analysis of post autoclave sulfur, the amount of tellurium contained in the sulfur ore reserves and available for recovery during final filtration of the molten sulfur product is 697 metric tons. Much of this tellurium was originally associated with gold and silver in telluride form, and thus the partial recovery of tellurium and associated precious metals will be done during this filtration step.

Kaolinite.

The sulfur caliche ore contains substantial amounts of kaolinite with some alunite. Based on very preliminary tests, it appears that these products may be largely separated prior to the initial rougher flotation, which is done at the relatively large size of -35 mesh, by simple decantation of the crushed ore in a slurry mixture with water. The kaolinite and alunite products tend to flocculate in solution, with the larger sulfur and quartz

particles quickly settling out of solution. The kaolinite and alunite can then be collected in large settling ponds. It is estimated that the total kaolinite contained in the caliche ore and available for recovery prior to primary flotation is approximately 2.2 million metric tons.

The total amount of alunite available for recovery is unknown, but would be substantially less. If the alunite can be separated, it could be treated with sulfuric acid to produce a high value potassium sulfate fertilizer.

Gold Values.

Field samples were checked for gold values using standard fire assay procedure, except that Minexco requested that a larger sample be prepared and submitted for assay. Geolab complied by preparing 100 gram samples instead of the usual 50 gram samples. Assay results were disappointing with only one assay registering more than 0.01 grams/ton, and 75% of the assays registering < 0.01 grams/ton. Results of fire assay for both gold and silver can be found in Table 9.

These fire assay results correspond poorly to fire assays done on the bulk samples, which registered 0.2 g/t, 0.2 g/t, and 0.315 g/t gold respectively. They also correspond poorly to the assays submitted to Geolab by Anglo American over the same areas, which varied from 0.04 to 0.29 g/t gold.

Composite samples were prepared and sent to the United States for analysis of gold and other metals using x-ray fluorescence (xrf). The xrf results showed unusually high values of both gold and silver. The gold values were largely contained in telluride minerals, with free gold accounting for only a fraction of the total gold values. Results of the xrf analysis can be found in Table 10. These values were plotted on the corresponding section found in Figures 18 and 19.

Total gold ore reserves encompass the same reserves as those blocked out for sulfur and also extend to include additional reserves in some of the eastern portions of section 1. Thus the gold reserves are slightly larger than the sulfur reserves, encompassing 19,200,000 metric tons of indicated reserves grading 4.74 g/t, and inferred reserves of 96,720,000 grading 4.46 g/t.

Total gold reserves are 115,920,000 metric tons averaging 4.5 g/t. In preliminary metallurgical tests conducted at the Marambio laboratory in Copiapo, extraction rates of 1.75 g/t were achieved. Additional testing needs to be done during feasibility studies concerning extraction of precious metals. Table 10 shows that 4 out of 5 samples tested for platinum and palladium show significant values of palladium and interesting amounts of platinum contained in the ore samples.

TABLE 9 - Gold Samples and Distribution
Fire Assay Results

<u>Trench & Sample #</u>	<u>Gold g/t</u>	<u>Silver g/t</u>
1 - 001	.01	0.2
1 - 002	<.01	0.1
1 - 003	.01	8.2
2 - 004	<.01	0.2
2 - 005	<.01	0.2
2 - 006	<.01	<0.1
3 - 007	<.01	<0.1
3 - 008	.01	0.1
3 - 009	-	<0.1
4 - 010	<.01	0.2
5 - 011	<.01	0.2
6 - 012	<.01	0.2
6 - 013	<.01	0.1
6 - 014	-	-
7 - 015	<.01	0.2
7 - 016	<.01	0.3
8 - 017	-	-
8 - 018	-	-
9 - 019	.01	0.1
10 - 020	<.01	0.1
11 - 021	.01	0.1
12 - 022	<.01	0.1
13 - 023	.01	0.1
14 - 024	<.01	0.2
14 - 025	<.01	22.0
14 - 026	<.01	0.2
14 - 027	<.01	0.2
14 - 028	<.01	0.3
14 - 029	.01	0.2
15 - 030	<.01	0.3
15 - 031	-	-

7,093.500

532.000

500

533.000

500

7,093.000

.A. 281

P.A. 251

7,092.000

7,091.500

7,091.000

P.A. 221

PA 191

Gold Ore Reserves, X-ray fluorescence - Sections 2 and 3
continued

Indicated Ore Tonnage - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net gold in grams</u>
0.93 g/t	5,600,000	5,208,000
5.46 g/t	1,600,000	8,736,000
=====	=====	=====
1.94 g/t	7,200,000	13,944,000

Inferred Ore Tonnage - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net gold in grams</u>
1.94 g/t	43,200,000	83,808,000

Total Inferred and Indicated Ore - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net gold in grams</u>
1.94 g/t	50,400,000	97,752,000

Total Indicated Gold Reserves - Section 1, 2, and 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net gold in grams</u>
4.74 g/t	19,200,000	91,085,760

Total Inferred Gold Reserves - Sections 1, 2, and 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net gold in grams</u>
4.46 g/t	96,720,000	431,031,600

Total Indicated and Inferred Gold Reserves - Sections 1, 2, 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net gold in grams</u>
4.50 g/t	115,920,000	522,117,360

P.A. 131

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A breakdown of the total mineral reserves given in this section is summarized in Table 12.

Total Reserves.

After extraction of the economic minerals, certain portions of the discharged tailings will contain relatively pure grades of silica quartz. Some of the local copper refineries may be interested in using the tailing as flux for their operations. Other industries which require high purity silica such as the glass and ceramics industries may be targeted if an economical transportation system can be devised. Total silica reserves are in excess of 90,000,000 metric tons.

Silica.

The average content of celestine, SrSO₄, would be approximately 2.1 times that of the strontium content alone, or roughly 0.63%. Total reserves of celestine, SrSO₄, in the blocked out titanium reserves of Piedra Amarilla are estimated at 710,136 metric tons.

A number of other elements were analyzed in some of the assays. Strontium appears to be the most consistent byproduct, averaging 0.3%. The large and rich strontium reserves found in the Salar de Piedra Parada were originally carried in solution from surrounding areas such as the Piedra Amarilla properties. Strontium mineralization in Piedra Amarilla is probably dominated by the mineral celestine, SrSO₄, while the dominant mineral in the Salar is likely strontianite, SrCO₃.

Titanium Byproducts.

Total titanium reserves encompass the same reserve areas as those blocked out for sulfur and gold, with some minor adjustments to reflect localized sampling.

Most of the individual field samples were combined locally to form composite samples. Individual samples were assayed at Geolab in Santiago and Rogers in the United States.

Blocked out titanium reserves in the form of rutile (TiO₂) include indicated reserves of 18,960,000 metric tons grading 3.63% TiO₂, and inferred reserves of 93,760,000 metric tons grading 3.11% TiO₂ (see figures 20-21, table 11). Combined indicated and inferred reserves total 112,720,000 metric tons grading 3.20% TiO₂, which corresponds closely to the average of the bulk samples assayed by Intec and SGS (3.04% TiO₂).

Titanium Reserves.

Figure 19

Gold Ore Reserves, by X-ray Fluorescence - Sections 2 and 3

Legend.

0	-	2 grams/ton	Green
2	-	4 grams/ton	Violet
4	-	6 grams/ton	Red
6	-	8 grams/ton	Orange
> 8	grams/ton	Yellow	

Pit Sample Locations

Indicated Reserves - Bounded Pink Line

Indicated Ore Tonnage - Section 2

Grade	Metric Tons	Net gold in grams
1.31 g/t	720,000	943,200
4.42 g/t	720,000	3,182,400
6.55 g/t	4,080,000	26,707,200
22.14 g/t	720,000	15,940,800
=====	=====	=====
7.50 g/t (ave)	6,240,000	46,773,600

Inferred Ore Tonnage - Section 2

Grade	Metric Tons	Net gold in grams
7.50 g/t	15,600,000	117,000,000
Total Inferred and Indicated Ore - Section 1		
7.50 g/t	21,840,000	163,773,600

TABLE 11 - Rutile Samples and Distribution

Composite Samples	Trench & Pit #	TiO ₂ %	Sr %
trenches	1,2,3 - 01	1.62	-
trenches	1,2,3 - 02	2.37	-
trenches	4,5	1.37	-
trenches	6,7	1.70	-
trench sample	6 - 014	0.60	-
trenches	8,9	1.73	-
trench sample	8 - 018	4.59	-
trenches	10,11	1.17	0.6
trench	14	7.03	-
trench	15,16,17	1.00	0.2
trench sample	15 - 031	1.60	-
pits	2,3,4	2.18	-
pit sample	4 - 040	0.65	-
pits	5,6,7	5.13	0.3
pit sample	5 - 041	0.92	-
pit sample	7 - 043	1.00	-
pits	8,9	1.87	-

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TABLE 11 Continued - Rutile Samples and Distribution

Composite Samples Trench & Pit #	TiO ₂ %	Sr %
pits 11, 12, 14	1.77	-
pit sample 11 - 047	0.35	-
pit 13	0.50	0.1
pit sample 13 - 049	0.40	-
pit 15	3.78	0.3
BULK SAMPLES		
Bulk Sample #	TiO ₂ %	
INTEC - 1 6,000 kilos	2.64	
INTEC - 2 8,000 KILOS	2.74	
SGS - 1 composite of INTEC 1 & 2	3.73	

Figure 20

Rutile (TiO₂) Ore Reserves - Section 1

Legend.

< 1.0%	Green
1.0 - 1.5%	Violet
1.5 - 2.0%	Red
2.0 - 2.5%	Orange
> 2.5%	Yellow

Pit Sample Locations

Indicated Reserves - Bounded Pink Line

Indicated Ore Tonnage - Section 1

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
0.50%	480,000	2,400
1.81%	2,400,000	43,440
2.18%	1,440,000	31,392
5.13%	1,440,000	73,872
=====	=====	=====
2.62%	5,760,000	151,104

Inferred Ore Tonnage - Section 1

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
2.62%	35,520,000	1,081,536

Total Inferred and Indicated Ore - Section 1

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
2.62%	41,280,000	1,232,640

7,093.500

500

533.000

500

7,093.000

P.A. 281

•

•

•

•

•

•

•

•

P.A. 251

•

7,092.000

•

•

7,091.500

•

•

7,091.000

•

P.A. 221

PA 191

Figure 21

Rutile (TiO₂) Ore Reserves - Sections 2 and 3

Legend.

< 1.0%	Green
1.0 - 1.5%	Violet
1.5 - 2.0%	Red
2.0 - 2.5%	Orange
> 2.5%	Yellow

Pit Sample Locations

Indicated Reserves - Bounded Pink Line

Indicated Ore Tonnage - Section 2

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
1.12%	3,120,000	35,088
1.68%	2,880,000	48,384
=====	=====	=====
1.39%	6,000,000	83,472

Inferred Ore Tonnage - Section 2

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
1.39%	15,840,000	220,176

Total Inferred and Indicated Ore - Section 2

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
1.39%	21,840,000	303,648

PIEDRA AMARILLA TRENCHING SCHEMATIC

UTM East

532.000

500

533.000

P.A. 191

534.000

P.A. 161

500

UTM North
7,089.000

7,088.500

76

7,088.000

7,087.500

7,087.000

pit 15

P.A. 131

Scale 1:10,000

72

Rutile (TiO₂) Ore Reserves - Sections 2 and 3 continued

Indicated Ore Tonnage - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
6.30%	7,200,000	454,160

Inferred Ore Tonnage - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
3.8%	42,400,000	1,611,000

Total Inferred and Indicated Ore - Section 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
4.16%	49,600,000	2,065,360

Total Indicated Rutile Reserves - Section 1, 2, and 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
3.63%	18,960,000	688,736

Total Inferred Rutile Reserves - Sections 1, 2, and 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
3.11%	93,760,000	2,912,912

Total Indicated and Inferred Rutile Reserves - Sections 1, 2, 3

<u>Grade</u>	<u>Metric Tons</u>	<u>Net TiO₂ in Metric Tons</u>
3.20%	112,720,000	3,601,648

Table 12

=====

Mineral Reserve Summary

<u>Mineral</u>	<u>Indicated Reserves 000 M tons</u>	<u>Inferred Reserves 000 M tons</u>	<u>Total Reserves 000 M tons</u>	<u>Net mineral Reserve 000</u>
Sulfur - grade %	21,120 19.84%	79,760 10.66%	100,880 12.58%	12,692 mt
Selenium - grade g/t	21,120 30.54 g/t	79,760 30.54 g/t	100,880 30.54 g/t	3.081 mt
Tellurium - grade g/t	21,120 6.91 g/t	79,760 6.91 g/t	100,880 6.91 g/t	0.697 mt
Kaolinite - grade %	21,120 2.2%	79,760 2.2%	100,880 2.2%	2,219 mt
Gold - grade g/t	19,200 4.74 g/t	96,720 4.46 g/t	115,920 4.50 g/t	522,117 g
Rutile TiO ₂ - grade %	18,960 3.63%	93,760 3.11%	112,720 3.20 %	3,602 mt
Celestine - grade %	18,960 0.63%	93,760 0.63%	112,720 0.63%	710 mt

Mining Plan

The conceptual mining plan calls for extraction of 25,000 metric tons per day (tpd) of ore to feed the mill at a rate of 22,500 tpd. Mill design will be 25,000 tpd with an actual operations at 90% of design capacity. Thus the mill will stockpile ore at the rate of 2,500 tpd. The mine is designed to operate 330 days per year (dpy), with a planned shutdown of 35 days during the June-July period for scheduled maintenance of ore extraction and transport machinery. During the planned shutdown of the mine, the mill will utilize the stockpiled ore.

Actual extraction of ore may be subcontracted out to reputable Chilean mining contractors, which would save substantial money in initial capital outlays for ore extraction equipment.

Parsons identified two pit areas to be developed into open pit mines for ore extraction. Identified as the Parsons North and South Pits, these encompass sections 1 and 3 of the mining reserves identified in the Piedra Amarilla properties (see Figure 22). Parsons used a sulfur cut-off grade of 20% for determining initial pit parameters. With the addition of titanium and gold minerals, no cut-off grade will be used in the new mining plan.

Parsons original plan called for equal extraction from each of the two pits and blending of the ore prior to beneficiation in order to achieve grade uniformity. A third pit should probably be developed in section 2. Each of the pits could contribute a minimum of 8,333 tpd in order to achieve the 25,000 tpd target. Blending of ore would probably best be suited following initial ore crushing and milling and after flash flotation for primary removal of sulfur concentrates.

Ore reduction will probably take place at or near the pits. Primary jaw crushers and secondary rock-to-rock crushers appear to be the most economic choice. Ore reduction equipment at each pit will have oversized capacities so that ore reduction equipment at two of the pits can compensate should maintenance be required at the third pit, requiring shutdown of the mining extraction and crushing activities at that pit.

Pre-stripping and stripping requirements are of a minimal nature. In the pre-feasibility study, Parsons estimated an overall stripping ratio of waste-rock to ore of 1:7. Even this modest requirement can probably be eliminated since the alluvial material contains important amounts of gold and titanium. Only the larger volcanic blocks will have to be stripped prior to ore extraction. Mining activities will begin at the top of the volcanic domes, eventually working downward and outward.

The equipment take-off list used by Parsons in the pre-feasibility study will form the basis for an expanded equipment

list to reflect the increase in ore extraction from 8,250 tpd to 25,000 tpd.

Elevation curves showing altitude changes are plotted on Figure 22, along with the different reserve areas, which will form the nucleus of the proposed pit areas. Elevation changes in the Piedra Amarilla group are generally less pronounced from north to south than they are from east to west.

The development of the mining pits will be preceded by road building activities from the pits to the mill area, as outlined in the pre-feasibility report.

Mine Life.

The indicated and inferred reserves are sufficient to support ore extraction at the rate of 25,000 tons per day, and 330 days per year, for a period of about 14 years.

Additional reserves are available in adjacent areas of the Piedra Amarilla properties, and will significantly extend the anticipated mine life. For purposes of reserve calculations, an in place ore weight or specific gravity of 2.0 was used. Actual laboratory measurements show that the caliche averages more like 2.1 g/cm³, and Parsons has estimated a minimum in place compacted weight of 2.2 g/cm³, which by itself would add additional mining reserves of 10%, and extend mine life an additional 16 months.

Hydrology Study

As part of the pre-feasibility study, Minexco executed a basic hydrological study of the Piedra Parada Basin and adjacent areas. Although Minexco's field personnel are not considered experts in hydrology, the basic data obtained from the study serves as a basis from which to begin a more in-depth and extended exploration of the area water resources during the feasibility study.

There are two basic types of surface water resources in this area of the Andes. The largest resource consists of saline or brine lakes and salars. These saline lakes receive most of their inflow in the form of underground streams and springs, usually in the form of thermal waters. Runoff from snow melt contributes a smaller portion of total water inflow into the saline lakes.

The second water resource consists of small streams which usually owe their source to thermal springs. Most of these small streams run for only a short distance before disappearing beneath the alluvial cover. Most of them surface at the margins of the lakes or salars.

Surface water resources which were within a 25 km circumference of the main group of the Piedra Amarilla properties were plotted on a 1:250,000 scale diagram shown in Figure 23. Directional flows of surface waters and watershed boundaries are shown in Figures 24 and 25.

Each of the salars and saline lakes vary in salinity and mineral composition. A sampling program was executed on the larger saline lakes. The Rio de la Cueva was sampled because it represents fresh water inflow before reaching an enclosed basin. The Rio Negro, Juncalito, and Leoncito systems were not sampled because almost all of this resource is used by Codelco at their El Salvador - Potrerillos copper complex.

Substantial subterranean waters are undoubtedly available in select areas. A subterranean study is beyond the purview of this pre-feasibility study.

Climate.

The northern portion of Chile is dominated by the Atacama desert from the coast to the pre-cordillera. Considered the world's driest desert, it owes the lack of precipitation to two basic climatic features.

The first of these is the prevailing trade winds, which in this portion of Chile blow from east to west. Thus, the moisture picked up from the trade winds is dropped on the Argentine Pampa and the eastern slopes of the Andes.

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from Piedra Parada. The Volcan de Copiapo area, which lies about 100 kilometers south of Piedra Parada, receives about twice as much annual precipitation. The cordillera east of Vallenar, about 240 kilometers south of Piedra Parada, receives about 3 times as much annual precipitation. These increases southward in precipitation are due to the stronger influence of the traditional Pacific winter storms. Because of the increased precipitation, these areas have substantially more fresh water resources available than does Piedra Parada.

The trenching program at the Piedra Amarilla properties was executed in June and July of 1988. The only winter snowfall that year occurred in early May, dropping some 10 - 12 inches of snow throughout the basin area. Most of the snow had melted by July except in the shady areas and areas above 16,000 feet.

Wind in the basin created many snowdrifts along the access road. These snowdrifts were typically about 2 feet deep. The bulldozer was used to clear the access road. This indicates that during winter months a snowplow will probably be needed to keep the roads open, especially after a heavy snowfall.

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Table 13

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Chemical Analysis of Surface Lakes

Mineral Content (in ppm)	<u>Laguna Brava</u>	<u>Laguna Jilgero A</u>	<u>Laguna Jilgero B</u>	<u>Laguna Bayo</u>
Chlorine	57,414	9,209	6,505	1,179
Sodium	21,218	5,060	3,991	966
Sulfates	7,700	3,900	4,200	1,700
Magnesium	1,661	254	173	55
Calcium	1,519	1,115	1,030	340
Hardness Mg as CaCO ₃	6,809	1,043	708	224
Hardness Ca as CaCO ₃	3,798	2,788	2,576	1,000
Potassium	1,137	523	327	111
Carbonates	1,211	211	220	143
Arsenic	8.8	2	2	0.5
Solids in Suspension	4,755	325	550	410
Dissolved Solids	115,690	23,110	17,545	4,810
Total Particulates	120,445	23,435	18,095	5,220
pH	8.0	8.25	8.25	8.5
Langelier Index	3.80	3.10	3.15	2.80

Rio de la Cueva.

Water samples were taken from Rio de la Cueva, near its source. The Rio de la Cueva begins as a thermal spring near the foot of the Panteon de Aliste volcano. The temperature of Rio de la Cueva is about 75o F. The stream runs for only a short distance before disappearing under the alluvial cover. The waters then flow underground and feed the La Laguna lake 15 kilometers distance from the thermal source.

The Rio de la Cueva is thought to be typical of the Piedra Parada Basin area's year round streams, which generally surface as thermal waters. The chemical salt composition is shown in Table 14. The stream's rate of flow is difficult to calculate because of a lack of expertise, but is thought to average better than 10 liters per second on a year round basis. The relatively low salt content of the Rio de la Cueva makes this an ideal water source for ore processing purposes.

Other sources of water which are low in salt content are the Rio Agua de Morales and the Pozo de Aliste thermal springs. The Rio Agua de Morales appears to have a higher rate of flow in liters per second than does the Rio de la Cueva.

The thermal waters entering the Jilgero A lake are much warmer than those of Rio de la Cueva. Likewise, the thermal springs at Rio Negro exceed 100o F.

For comparison purposes, water samples were extracted from two streams some 100 kilometers to the south near the Volcan de Copiapo area. The first of these streams drains the Paipote Canyon from Vega la Junta to La Puerta. Three samples were taken at different points, representing a total of some 50 kilometers distance. The second stream is a tributary of the Copiapo River, and lies about 15 kilometers south of Vega la Junta.

The results of these chemical analysis can be found in Table 15. Some interesting comparisons are found when studying Tables 14 and 15. The most striking are the difference in chlorine content, and predominance of Na over Ca in the Rio de la Cueva. The relatively high carbonate content of Rio de la Cueva suggests the thermal waters may be in partial contact with a marine sedimentary basement formation.

The chloride-charged thermal waters of the Piedra Parada basin make an excellent medium for leaching of ferrous and non ferrous metals. This may help explain the fairly high metal content of the Piedra Parada Salar dune formations, and the presence of precious metals therein. The precious metals would precipitate out of solution from the thermal waters with a significant change in pH, such as that found in a salar or brine lake. In the Paipote stream, arsenic values decline with distance due to probable ion precipitation.

The second climatic feature is the Humboldt ocean current, a cold current flowing north from the Antarctic to Peru. The evaporating ocean water forms fog along the coast but does not condense into rain further inland because of the temperature difference.

Evaporation of ocean water creates late night and early morning fog which is prevalent along the coast during most of the year. The fog reaches inland some 20 miles or so, burning off during the early morning hours in summer, and later in the day during the winter months.

The area from Chanaral to Piedra Parada roughly forms the southern boundary of the dry Atacama desert area. Rainfall throughout the Atacama cordillera areas occurs generally from rare storms formed offshore in the Pacific during the advent of the El Nino weather phenomena, and from storms which blow over from the eastern Andes during the February - April period, known in South America as the Bolivian winter.

Because the Piedra Parada area is far enough to the south, it is impacted by some of the stronger conventional storms from the Pacific winter weather pattern, which sometimes wander a little further to the north. These storms provide the bulk of the annual precipitation to the basin area, contributing from 10 to 15 inches of snow in a typical year. Storms from El Nino usually occur only every 6 or 8 years, but can drop as much as two feet of snow. El nino occurred in 1987 and again in 1991. Snow from the Bolivian winter is generally modest, averaging 1 or 2 inches per snowfall, and occurring once or twice a year.

Annual precipitation increases dramatically as one travels south from Piedra Parada. The Volcan de Copiapo area, which lies about 100 kilometers south of Piedra Parada, receives about twice as much annual precipitation. The cordillera east of Vallenar, about 240 kilometers south of Piedra Parada, receives about 3 times as much annual precipitation. These increases southward in precipitation are due to the stronger influence of the traditional Pacific winter storms. Because of the increased precipitation, these areas have substantially more fresh water resources available than does Piedra Parada.

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Table 14

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Chemical Analysis of Rio de la Cueva

<u>Mineral Content</u>	<u>in ppm</u>
Chlorine	1,027
Sodium	715
Sulfates	550
Magnesium	61
Calcium	303
Hardness Mg as CaCO ₃	248
Hardness Ca as CaCO ₃	757
Potassium	64
Carbonates	489
Arsenic	1
Solids in Suspension	0
Dissolved Solids	3,250
Total Particulates	3,250
pH	6.8
Langelier Index	1.50

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Lake Analysis.

A chemical analysis of four of the saline lakes is shown in Table 13. Laguna Brava represents by far the largest surface water resource in the immediate area. It also contains by far the highest salt content of any of the waters tested. The total particulate content of 120,445 parts per million translates into a salt content of 12%, more than three times saltier than ocean water.

The entry point of inflow into the Laguna Brava has not been pinpointed, but is believed to be related to a small lake area just north of the main body of water. Intercepting the inflow before it reaches the main body of water would give the project a fairly large amount of "fresh" water for processing purposes.

Because Laguna Brava represents the worst case scenario as far as salt content in local surface waters, it was decided to employ the water from Laguna Brava in metallurgical tests to determine any negative effects during processing. Samples were taken from three different areas along the eastern shore of Laguna Brava and sealed in plastic barrels, and transported to Intec in Santiago.

Several pilot tests were run using the salt water during ball milling and primary flotation for extraction of sulfur. Recovery rates were marginally higher using the Laguna Brava salt water, as opposed to the regular tap water in Santiago. The salt water did cause noticeable corrosion of exposed iron in the milling and flotation circuit.

The Langelier index, which measures corrosion and deposition of salt in salt water is a relatively high 3.8 for the Laguna Brava waters. Thus, water employed from this source will have to be done with processing equipment in which all exposed iron surfaces have been treated or sealed to protect against corrosion.

Water samples taken from the Jilgero lakes show salt contents on a smaller magnitude. These waters are somewhat limited as to quantity.

The Laguna Bayo is the "freshest" of the lake waters, with a salt content approaching that of the Rio de la Cueva (see Table 14). Because of the relatively small quantity of water contained in the lake, Minexco feels that the lake must have an underground outlet which allows it to maintain a fairly low salt content. The water inflow into Laguna Bayo comes from the nearby Sierra Nevada, which has several peaks approaching 20,000 feet in elevation, and which has some year round snowpack. It is possible that some of the water flows northward into Laguna Brava, as both lakes are at about the same elevation. The water from Laguna Bayo is an ideal source for processing purposes.

Table 15

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Chemical Analysis of Paipote River System and Paton Tributary
of the Copiapo River

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<u>Mineral Content (in ppm)</u>	<u>Paipote 1 La Junta</u>	<u>Paipote 2 P.Grandes</u>	<u>Paipote 3 La Puerta</u>	<u>Vega Paton</u>
Chlorine	35.4	60.5	129	31.4
Sodium	67.9	126.0	278	69.6
Sulfates	277	560	1,125	338
Magnesium	34.0	17.9	64	14.6
Calcium	88.4	179.0	290	105
Carbonates	<0.5	<0.5	<0.5	<0.5
Bicarbonates	243.8	134.9	252.8	160.9
Total Alkalinity	200	110.7	207.4	132
Total Hardness	360.9	521.2	988.4	322.6
Arsenic	8.1	2.7	0.8	370
Selenium	.0003	.0005	.0004	<.0002
Tellurium	<.01	.01	<.01	<.01
Solids in Suspension	630	1,010	2,000	660
pH	7.37	6.68	7.93	6.53

Temperature.

In spite of the high altitudes, winter temperatures are tempered by the relative proximity of the Pacific Ocean, which lies 120 miles to the west, and the prevailing surface winds, which come from the northwest. Temperature readings as recorded by Minexco personnel are as follows:

Summer Maximum Average	65o F
Summer Minimum Average	25o F
Winter Maximum Average	40o F
Winter Minimum Average	10o F
Warmest Recorded Temp.	80o F
Coldest Recorded Temp.	-22o F

Temperatures were recorded at 14,000 feet to 15,000 feet above sea level, near the main body of the Piedra Amarilla Properties. Winter temperatures and climate are sufficiently mild to permit year round mining activities.

Wind.

Wind velocity is an important factor in certain areas of the Cordillera. Afternoon temperatures on the desert floor cause large air masses to rise, forcing adjoining air masses to move through the mountain passes as afternoon winds. Wind velocity can vary greatly from one area to the next. The mountain ranges and volcanic peaks form natural wind barriers, creating wind blocks, forcing the winds to change paths. The western slope of the Piedra Amarilla 131 and 161 properties receives substantial afternoon winds, with an average mean velocity of 35 kilometers per hour during windy periods. Winds die down at night. Plant and mining structures will have to be built to withstand maximum sustained winds of 100 kilometers per hour (60 miles per hour).

Seismology.

No seismographic information is available for the Piedra Parada Basin area. Most of the major earthquake activity in this area of Chile has its epicenter offshore in the Atacama Trench.

There are several major geologic fault systems identified in the pre-cordillera area. Geologists at the Coipa deposit 75 kilometers to the southwest of Piedra Parada have determined seismographic activity as follows:

Maximum 10 Year S Wave	6.8 Richter
Maximum 10 Year P Wave	6.3 Richter

Further seismographic information is available at the Chilean National Mining Service.

Preliminary Flow-sheets for Metallurgical Recovery of Sulfur and Gold.

DEF-501316

METALLURGICAL STUDIES

A number of metallurgical studies have been performed on bulk ore samples extracted from the pits and trenches of the Piedra Amarilla properties. These studies have largely been executed by Intec-Chile, and are mostly targeted at the recovery of the elemental sulfur reserves present on the properties.

Prior to the pre-feasibility study executed by the Parsons Company, only two metallurgical studies had been completed. The pre-feasibility study addressed the weaknesses of the metallurgical work done to that time and set the tone and direction to be undertaken in the subsequent metallurgical work.

Table 16 gives a list of the completed studies published to date, totalling more than 1,100 pages. Not included on this list are certain marketing and economic studies previously compiled by C.S.I., nor an economic study largely finished by Intec and which covers production of sulfur and titanium. This latter study is currently published in preliminary draft form only and is in the possession of both Intec and Corfo. C.S.I. is currently searching for a joint venture partner to help in finishing this economic study and in other metallurgical work required at the level of a feasibility study.

The parameters for the metallurgical work outlined hereafter are influenced to some degree by the ability to market the finished products. Titanium pigment, for example, is produced instead of marketing rutile concentrates, because of the difficulty and expense in producing 95%+ TiO_2 grade concentrates.

The metallurgical flowsheet for sulfur recovery as outlined herein has been extensively researched and will suffer few if any modifications in subsequent studies.

The flowsheets for both rutile and telluride gold recovery are based on fewer study criteria, and include such factors as process of elimination and supposition. Substantial data was borrowed from metallurgical flowsheets developed by other companies on ore samples of varying characteristics.

Thus, during future metallurgical studies, the flowsheets for both gold and rutile may undergo major changes from those presented here. For example, it might be shown in later studies that the production of titanium pigment is more economical using a chlorination method rather than the sulfation method.

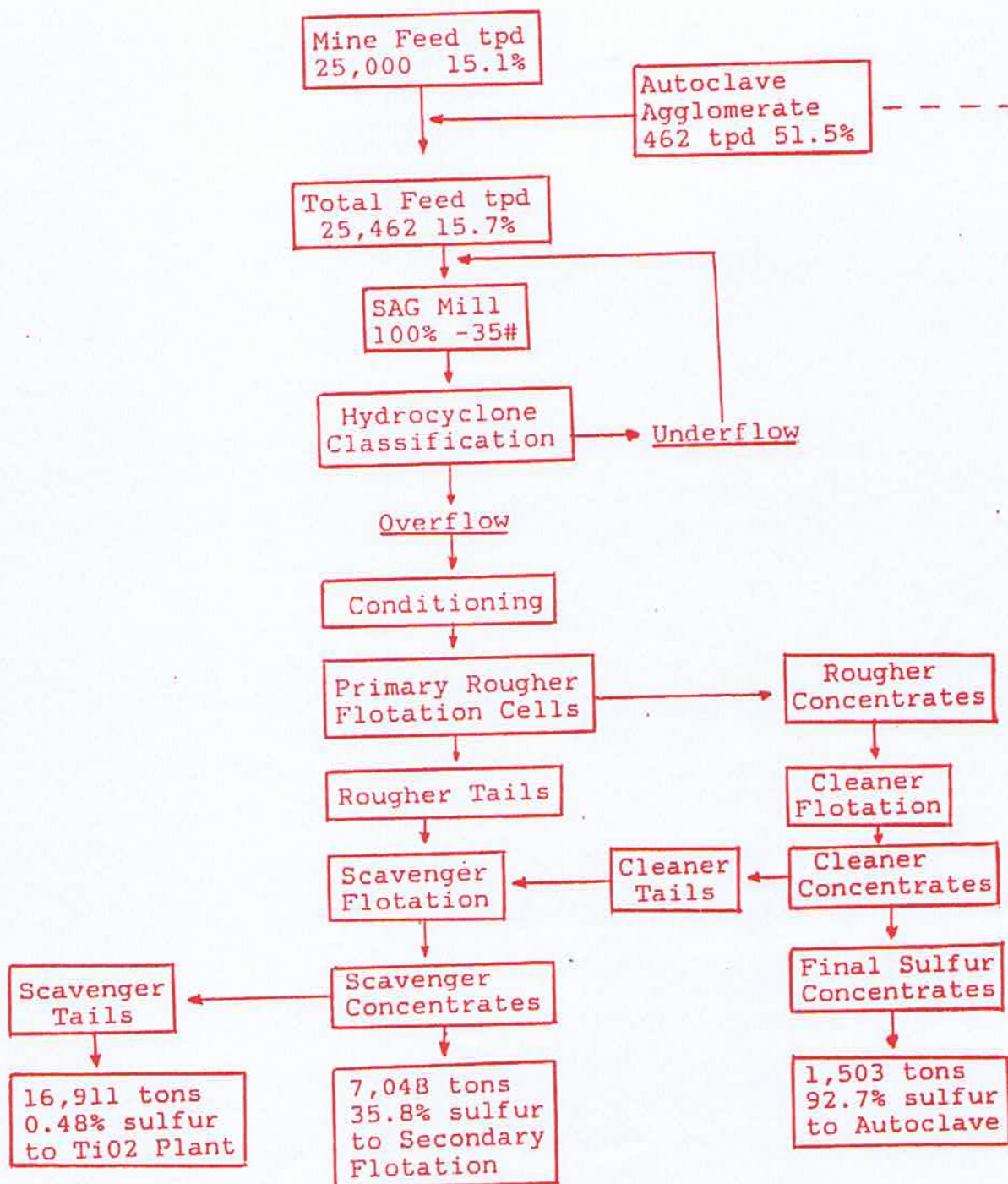
Sulfur Flowsheet.

The principle steps in recovery of elemental sulfur are outlined in the flowsheets found in figures 26, 27, and 28. Sulfur concentrates are recovered during various stages of flotation, with final concentrates feeding a continuous autoclave melter.

METALLURGICAL STUDIES
TABLE 16

<u>Date</u>	<u>Title of Study</u>		<u>Lab/Sponsor</u>
July 1987	Concentration of Sulfur	21pp	Intec-Chile
Oct. 1988	Flotability of Sulfur Caliche	19pp	Intec-Chile
Nov. 1988	Chile Sulfur Project Prefeasibility Study	157pp	RM Parsons Pasadena, CA
Jan. 1989	Flotability of Sulfur Caliche and Control of Impurities	21pp	Intec-Chile
May 1989	Flotation of Sulfur Caliche	10pp	Intec-Chile
Oct. 1989	Technology for Production of Sulfur on a Large Scale	12pp	Intec-Chile
Oct. 1989	Technology for Production of Sulfur on a Large Scale	28pp	Intec-Chile
Nov. 1989	Titanium and Gold Telluride Distrib. in Sulfur Flotation	10pp	Rogers Research Salt Lake City
Nov. 1989	Titanium and Gold Telluride Distrib. in Caliche Samples	5pp	Rogers Research Salt Lake City
Dec. 1989	Preliminary Cyanidation Exper- ience using Caliche Samples	4pp	Marambio Lab Copiapo
Dec. 1989	Flotation of Sulfur Caliche - 3rd Region	290pp	Intec-Chile
Aug. 1990	Minerological Characterization of Titanium Bearing Species	7pp	CIMM Santiago
Aug. 1990	Recovery of Industrial Minerals from Sulfur Caliche	30pp	Intec-Chile
Sep. 1990	Preliminary Analysis of a Bulk Caliche Sample	1pp	Marambio Lab Copiapo
Dec. 1990	Chemical and Microscopic Character- ization of Ti Bearing Caliche	6pp	Intec-Chile
Jan. 1991	Pilot Flotation of Caliche Sulfur and Melting of Concentrates in Continuous Autoclave	490pp	Intec-Chile
July 1991	TiO ₂ Concentration Study Proposal	5pp	SGS Santiago

FIGURE 26
SULFUR METALLURGICAL FLOWSHEET
PRIMARY FLOTATION



Melted sulfur is formed using standard prill technology.

Primary Flotation.

DEF-501322

Primary flotation material consists of -35 mesh feed which has been reduced using primary and secondary jaw crushers, and tertiary reduction using back-to-back centrifugal rock against rock mills. No tests have been performed in the rock against rock mills, but were selected by Intec because of the friability of the gangue material, which is composed of semi-compacted volcanic ash and loose tuffs.

It should be noted that in some areas the tuffs are more hardened, thus tertiary reduction using rock against rock mills may not be adequate. Oversize material from the rock against rock mills feed either a ball mill or a SAG type mill. Hardened quartz tuffs from the Carolina properties would provide grinding media for a SAG mill. Elimination of as much tramp iron as possible should be a priority in selecting reduction equipment for the ore.

Mine feed as shown in Figure 26 averages 15.1% sulfur, which is higher than the overall project average of 12.58%. It is presumed that ore extraction will begin near the tops of the various volcanic domes where the sulfur content is higher, thus during the first years of the project sulfur extraction will be higher than in the latter years.

Mine feed is mixed with a sulfur rich autoclave byproduct called agglomerate, which will probably have to be re-crushed and ground in the SAG circuit. Tests by Intec have shown that this agglomerate byproduct can be recovered when recycled prior to primary flotation. Thus total mine feed is projected during the first years of operation to average 15.7% sulfur.

Prior to conditioning and primary rougher flotation, the ground ore will be routed to a gravity circuit for recovery of telluride gold and silver values (see gold flowsheet). Gold tailings are then conditioned and sent to primary rougher flotation of sulfur.

Intec successfully experimented with a novel flotation device and developed a "flash" flotation technique which should be applied as a pre-concentration step prior to the primary rougher flotation step. Since sulfur is naturally hydrophobic, conventional flotation cells have difficulty evacuating the sulfur concentrate with sufficient speed, causing a backup of sulfur product within the cells, and a partial up-welling of quartz gangue material, which is evacuated along with the sulfur.

The flash flotation device consists of using large volume cylindrical tanks, operated in the same manner as a conventional flotation cell. Sulfur concentrate is evacuated from the tank over a spillway by turbulent air bubbles forced into the slurry.

Flash flotation can be used to upgrade primary flotation sulfur feed, eliminating substantial amounts of gangue material. Intec has also discovered that due to the flotation kinetics of sulfur, low grade sulfur ore of say, 10% S, can be concentrated in primary rougher flotation using the same amount of conventional flotation cells as ore containing 30% S. This makes virtually all of the lower grade sulfur reserves economically recoverable.

Primary rougher concentrates are sent to a cleaning stage, with final concentrates reporting to the autoclave. Primary rougher tails are mixed with the cleaner tails, and sent to scavenger flotation. Scavenger concentrates report to a secondary grinding system, where they are prepared for secondary flotation. Primary scavenger tails report to the rutile recovery plant.

Secondary flotation.

Primary scavenger concentrates are classified using hydrocyclones, with underflow reporting to a secondary SAG mill, where they are ground to 70% -100#. This step helps to separate the finer sulfur particles from the gangue material, and will allow the primary scavenger concentrates to become substantially upgraded during secondary flotation.

The steps for secondary flotation are similar to those of primary flotation, with production of a final cleaner concentrate which reports to the autoclave, and a scavenger concentrate which reports to a tertiary grinding circuit in preparation for final flotation.

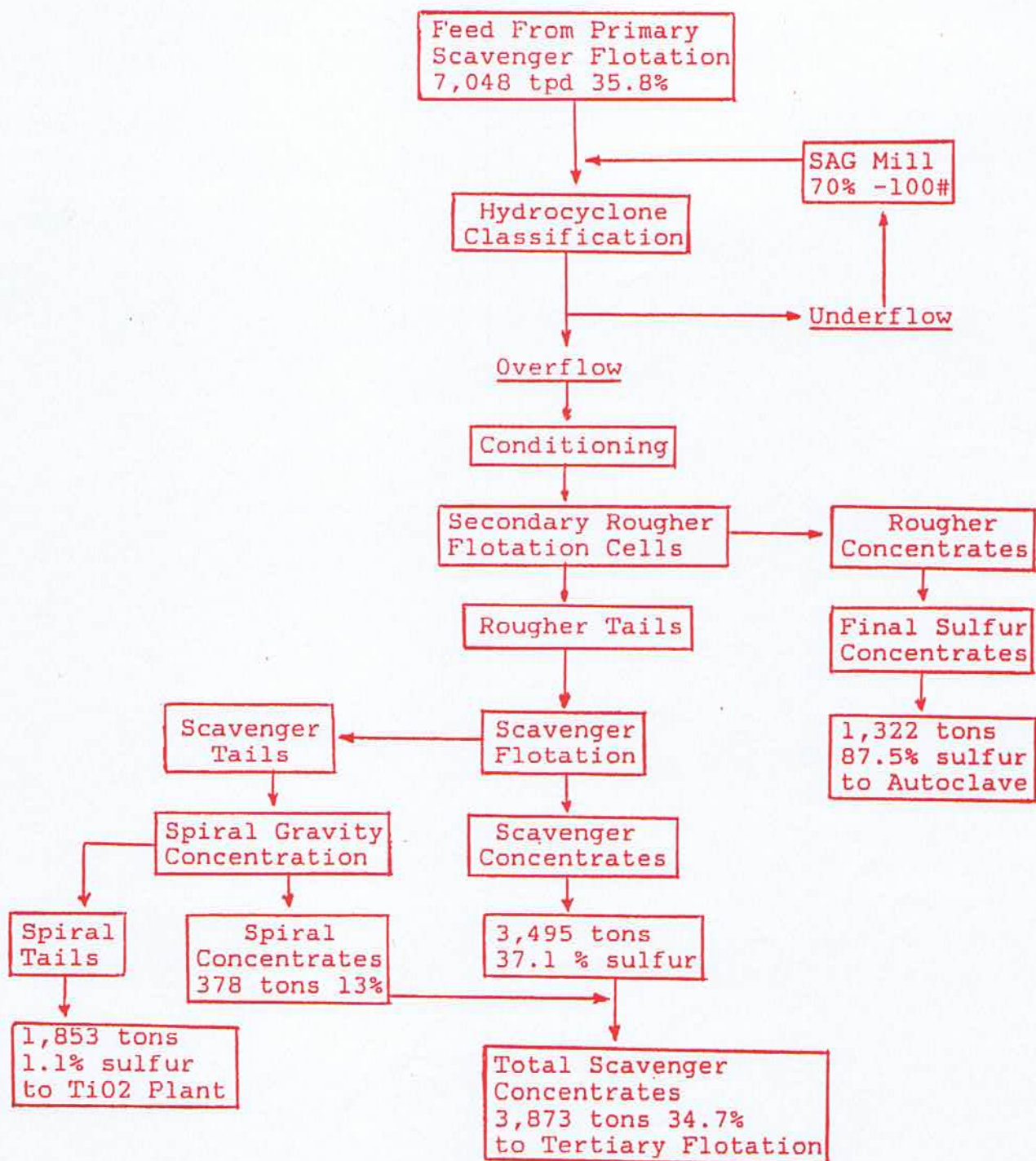
The scavenger tails from secondary flotation can be upgraded by using large volume spiral concentrators. In this case, the sulfur, with a specific gravity of 2.0, is concentrated in the tailings portion of the spiral product. The flotation frothers still present in the slurry will also tend to make the sulfur float as it works its way down the spiral. Since the total volume to be treated is less than 10% of the original primary feed, this step can be accomplished for a very modest capital investment, and may add as much as 40 mtpd of sulfur to the secondary scavenger concentrates.

Final scavenger tails will report to the rutile recovery plant.

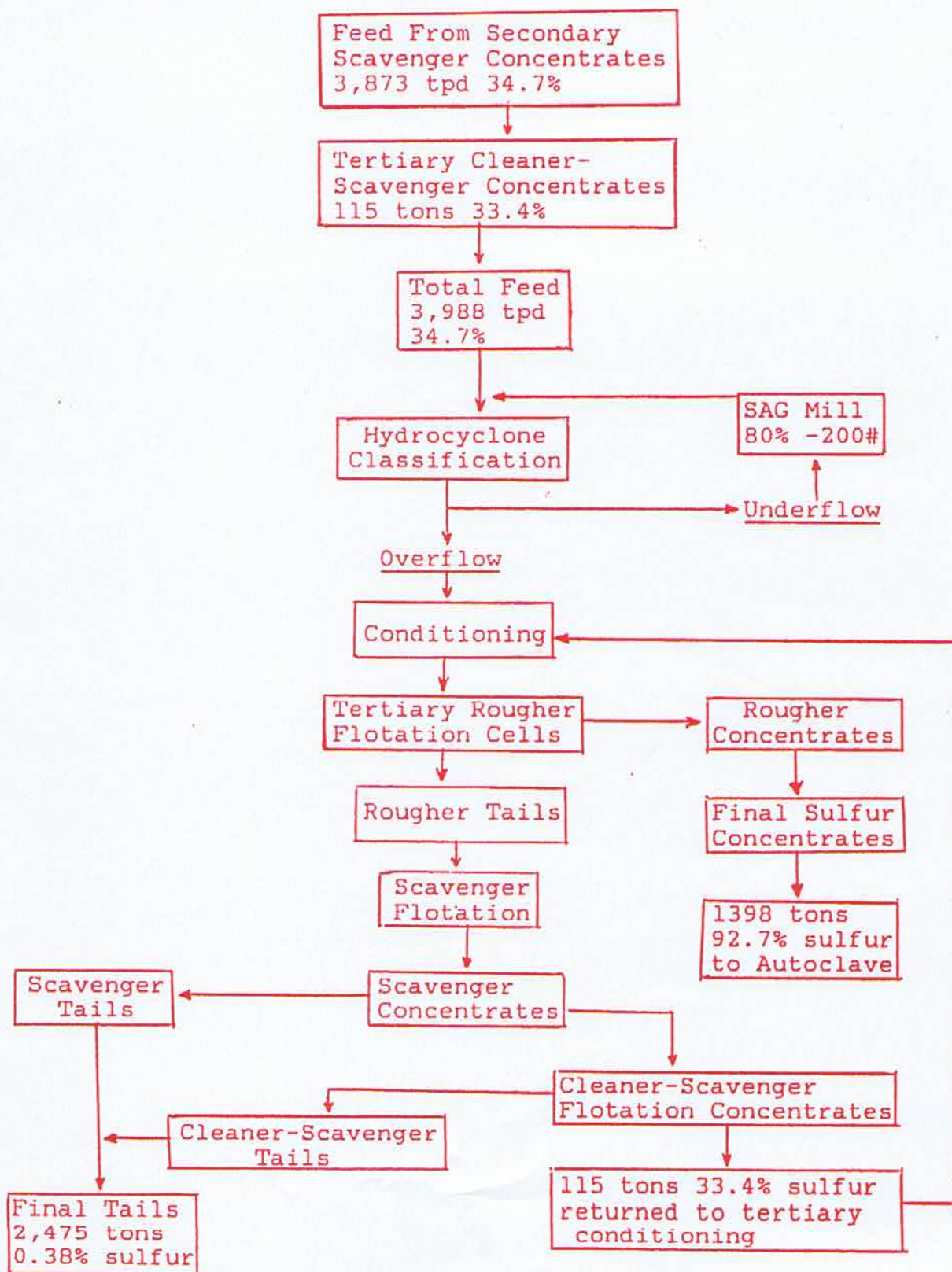
Tertiary Flotation.

Secondary scavenger concentrates are classified, with underflow feeding a SAG mill where they are ground to 80% -200#. Again, this differential grinding allows separation of sulfur from the gangue mineral, and upgrading of the concentrates in a tertiary flotation step.

FIGURE 27
SECONDARY FLOTATION



Tertiary Flotation



Tertiary flotation is similar to the secondary flotation, except that scavenger concentrates are recycled to the tertiary conditioning tank, and scavenger tails are not treated with gravitational spirals. Cleaner concentrates are routed to the autoclave, and final tailings are routed to the rutile recovery plant.

Autoclave.

Intec developed a method which allows continuous autoclaving of sulfur concentrates. The technology was developed under a partial subsidy from Corfo, and under terms of the agreement previous to the study, any new technology developed would be jointly owned by Corfo and Minexco. The continuous operation has obvious advantages over autoclaves which are confined to batch operation. The general flowsheet for the melting and prilling operations is shown in Figure 29.

Concentrates entering the autoclave are melted at a pre-determined temperature (± 1300 C). and pressure. The molten sulfur bath separates into three distinct layers according to specific gravity, each of which is tapped and bled.

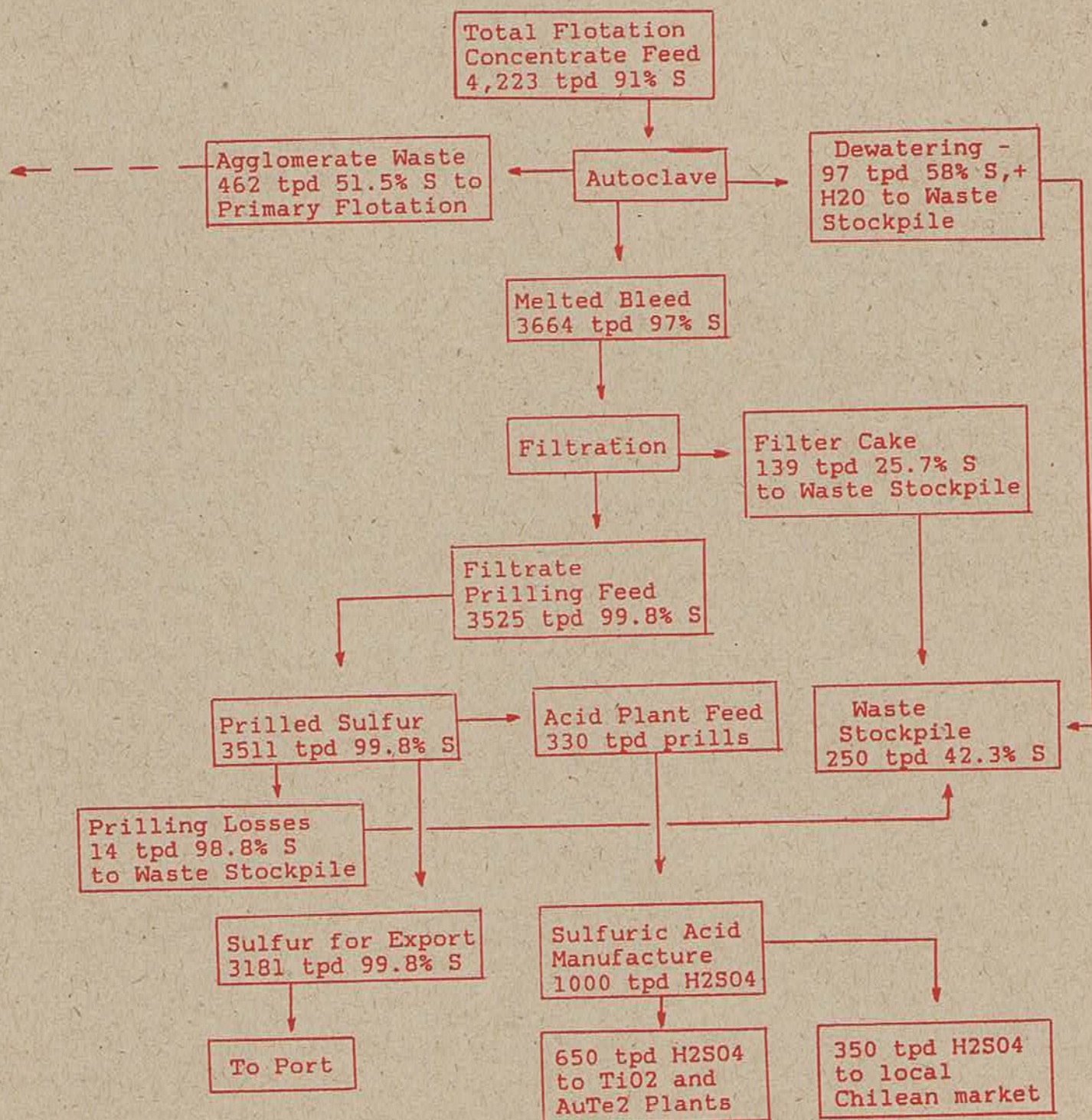
The top layer consists largely of super-heated water, which is the melting medium. The bleed from the first layer forms the de-watering waste referred to in figure 29.

The bottom layer receives the unmelted solid particles, consisting largely of silicates, which because of their heavier specific gravity decant. The bleed from this bottom layer forms the agglomerate waste. The molten sulfur which is bled along with the silicate particles will cool and solidify, forming a hard silicate-sulfur agglomerate. This agglomerate will be rerouted to the primary flotation circuit for sulfur recovery, but will need to be reduced in the crushing and grinding circuit first.

The bleed from the middle layer contains liquid sulfur which averages 97%+ purity. In practice, Intec was able to consistently bleed sulfur with a purity exceeding 99.6%. Impurities consist largely of fine ash silicates. The molten bleed is routed to pressure filtration, with the molten filtrate feeding the prilling plant. A final sulfur prill of 99.8% purity is produced. A portion of the final sulfur product, about 10%, will be routed to the sulfuric acid plant.

Wastes from the de-watering bleed, filter cake, and prilling losses will be stockpiled. The filter cake should contain significant gold tellurides and will need to be treated. No test work has been done on recovering sulfur from the various stockpile wastes, but conceivably an additional 70 - 100 tons per day of additional sulfur could be recovered from these wastes.

FIGURE 29
SULFUR MELTING AND PRILLING



Sulfuric Acid Manufacture.

It is contemplated that about 10% of the molten sulfur filtrate will be fed directly into a sulfuric acid plant. Since the acid plant furnace requires molten sulfur, there is no need to prill this material. A number of different types of acid plant designs are available and can be installed on a turn-key basis. Capital cost for installation of a 1,000 tpd sulfuric acid plant has been estimated at \$92,000,000 by Intec-Chile. This is by far the largest cost item of the projected fixed capital costs.

Having an acid plant will ensure an abundant supply of H_2SO_4 at a cost of about \$20 per metric ton, which is largely the cost of plant amortization and interest. The acid plant will also be an important net provider of energy.

Local supplies of acid in Chile are uncertain at best. A projected abundance of acid from Chiquicamata has yet to materialize. Oxide leaching of copper waste rock has consumed most of the available acid supply. Even if substantial amounts of acid become available in the future, the cost of shipping from the Ventanas smelter or from Chiquicamata must be factored in to the purchase of smelter acid.

Supplies at Paipote are likewise uncertain. A planned expansion of the existing acid plant to 100,000 mtpy will make about 50,000 mtpy of acid available to the local market in the Atacama region. A large portion of this acid will probably be consumed by existing copper operations, including the new Candelaria project. Spot market prices in the region for small users is currently in excess of \$100 per metric ton.

Daily Acid Requirements.

The projected TiO_2 pigment recovery plant requires about two tons of acid for each ton of pigment produced. This translates into about 600 metric tons per day of 90%+ acid. The sulfation of the gold telluride concentrates will require perhaps an additional 50 metric tons per day of acid, for a total daily consumption of 650 metric tons. Excess plant acid can be readily sold in the local markets at a steep discount from prevailing prices.

A portion of the acid used in the pigment plant will be consumed in reactions creating sulfates of iron, calcium, potassium, in neutralizing carbonates, etc. The balance will be discharged from the pigment plant as a slurry stream with water. Industry practice has been to neutralize this slurry stream with lime, which adds about \$30 cost for each ton of pigment produced.

If the pigment plant is located close to the rutile flotation plant, then most this acid slurry can be recycled and used to lower the pH of the flotation water, which must remain at a pH of 2 - 3. This requires from 6 to 12 lbs. of acid per metric ton of

ore, a total daily requirement of 68 to 136 metric tons of sulfuric acid. If an excess of un-neutralized acid still remains in the slurry stream after pH adjustment of the rutile flotation water, it can be used in processing any of the large mineral beds found in the Piedra Parada Salar or perhaps in the manufacture of potassium sulfate from local sources of alunite.

Native Sulfur Impurities.

International sulfur buyers require minimum sulfur purities of 99.5%. In the contact process used in sulfuric acid manufacture, a catalyst is used to speed up the reaction of converting SO_2 into SO_3 . This catalyst usually is vanadium pentoxide, although platinum is sometimes used.

Impurities such as arsenious oxide (As_2O_3) produced during the burning of impure sulfur or the roasting of metal sulfides can contaminate the catalyst and stop the reaction from proceeding. Thus, in most plants the hot sulfur dioxide produced in the furnace is then purified by passage through a dust collector in which dust is electrostatically precipitated, and then through a scrubbing tower in which gases pass through a trickling of concentrated sulfuric acid. This treatment removes most of the impurities in the gas.

Table 17 shows typical impurities found in the molten ore bled from the autoclave prior to filtration. Because Intec-Chile lacks pressure filtration equipment, no real filtration tests were performed. Pressure filtration was attempted using two 70 mesh screens placed at a 45 degree angle. The effective filtration size was only about 100 mesh, but was sufficient to eliminate about 50% of the silicate impurities.

Table 17
Sulfur Product Before and After 70 mesh Filtration.

	Molten Sulfur	After 70 Mesh Filtration
- Sulfur	99.60 %	99.86 %
- Silicates	0.41 %	0.22 %
- Arsenic	0.009 ppm	0.002 ppm
- Bismuth	0.08 ppm	0.06 ppm
- Tellurium	38.80 ppm	38.10 ppm
- Selenium	156 ppm	181 ppm

Table 17 shows that the sulfur can be upgraded to 99.95% + by elimination of the silicates during filtration. Arsenic and bismuth are far below maximum accepted contaminant levels. This indicates that As and Bi are strongly associated with the quartz gangue and not with the sulfur.

Additional removal of tellurium and selenium contained in the molten sulfur is important to avoid marketing penalties. These penalties might be avoided in any event because the overall purity of the final sulfur prill is far above the average sulfur product sold in the international market. Pressure leaf filtration may remove the bulk of the Te and Se. If not, Parsons indicates that a change in pH by adding lime might precipitate the impurities.

Removal of the tellurium is desirable because of the concentration of gold tellurides in the molten sulfur product. The Te levels in the molten sulfur represent about 25% of the known Te values contained in the head ore. This indicates a moderate association with sulfur, which is probably due more to co-flotation of the tellurides during sulfur flotation than any chemical association. This also indicates that removal of tellurides during filtration or using pH adjustment is a high probability.

Se values in the molten sulfur are 100% of known values contained in the head ore, showing a strong affinity with the native sulfur and indicating that removal through filtration may be difficult. During the manufacture of sulfuric acid, selenium and tellurium are precipitated along with other impurities into the plant sludges. Table 18 shows the typical content of sludges in varying acid plants.

Table 18
Elemental Content of Sulfuric Acid Sludges

- Selenium	0.9 - 63.7%	- Lead	4.82 - 30.4%
- Tellurium	0.2 - 15.0%	- Iron	8.0 %
- Arsenic	0.3 - 55.3%	- Water	0.2 - 13.55%
- Sulfur	5.0 - 22.0%		

Variance in sludge content is due to direct smelting of pyritic ores in some plants. Even using the lower contaminant percentages found in the acid sludges, Table 18 would seem to indicate that typical sulfur feed for acid plants does contain significant amounts of selenium and tellurium contaminants. Thus, there would appear to be plenty of negotiating room for international marketing of a sulfur product which is very low in total impurities but not entirely free of selenium and tellurium contaminants.

PRECIOUS METALS RECOVERY

Precious metals content of the Piedra Amarilla ore body is largely related to mineralization in the form of gold and silver tellurides. Platinum group metals, especially palladium, are also found, probably as both sulfides and tellurides. Metallurgical work relating to the precious metals has been hampered by a lack of company resources and the general lack of existing technical expertise in Chile.

The inability to accurately assess precious minerals content using standard fire assay procedures also casts a cloud of doubt and uncertainty as to the accuracy of the subsequent values found through the secondary XRF analysis completed in the United States.

The absence of any XRF equipment in Chile made it imperative to develop an alternative method of verifying precious metals content. It was with this in mind that a modest cyanide leaching program was undertaken at Marambio's laboratory in Copiapo. Based on our limited knowledge of telluride extraction, the preliminary results seem to indicate that the ore does indeed possess a relatively important gold content, vindicating the XRF analysis by Rogers.

A representative ore sample taken from a four ton bulk sample from the excavated pits was cyanided for 24 hours using a solution of NaCN. Leach solutions were read directly by A.A. for gold leached into solution. After seven hours, a total of 0.8 g/ton had leached into solution, and after 24 hours a total of 1.25 g/t had leached into solution.

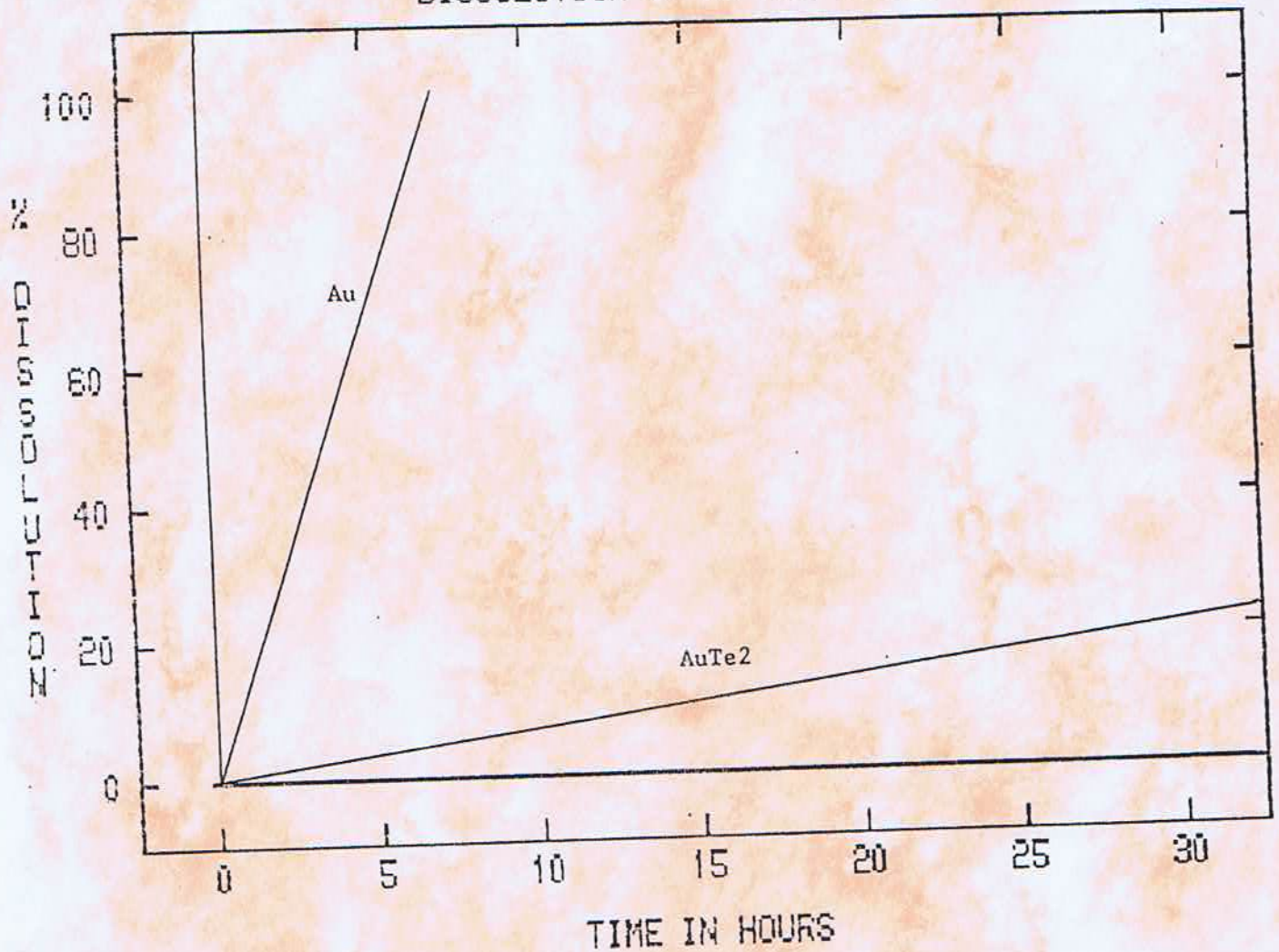
The pregnant solution was treated to precipitate the gold out of solution. Only 28% of the gold values originally leached into the solution, or 0.35 g/t, precipitated and was recovered. This indicates that the gold telluride must be oxidized prior to leaching in order to liberate the leach solution.

Figure 30 indicates the leaching efficiency of gold tellurides using cyanide solution. As the graph indicates, after 30 hours only a small fraction of the telluride values contained in concentrates is solubilized.

The leaching experiments were successful in determining a minimum level of gold content in the Piedra Amarilla ore. The experiments would seem to limit direct leaching of the tellurides as a secondary consideration due to the problems in dropping the gold from solution.

A number of oxidation procedures are available which will permit chemical reduction of the telluride and recovery of elemental gold. First, a satisfactory procedure must be developed for recovering the telluride values into concentrate form.

FIGURE 30
DISSOLUTION OF Au AND AuTe₂



Comparison of leaching efficiency rates of gold vs. gold-telluride in solution of 0.25% KCN at ambient temperature.

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Telluride Characteristics.

Precious metal tellurides are characterized by their softness and density. Most of the gold and silver tellurides have densities between 8.0 and 9.0, making them heavier than all but a few minerals. Most of them also have a hardness of between 1.5 and 2.5 on the Mohs scale.

During normal mining operations, any telluride minerals present in the ore are continually pulverized because of their softness. Drilling, blasting, loading, trucking, unloading, crushing, and milling of ore all contribute to pulverization of tellurides. The Kalgoorlie telluride mine in Australia took advantage of the tendency of the tellurides to concentrate in the fines by sorting out the coarser mesh ore after primary milling.

Methods of Concentration.

The high density of the tellurides offers the strong possibility that an important portion can be recovered using gravitational devices, unless the ore is too finely ground. A fine grind would probably pulverize the majority of the tellurides to a point at which gravitational recovery becomes economically less attractive.

Flotation can also be applied to many types of telluride minerals. The Emperor mine at Fiji employs telluride flotation ahead of sulfide flotation, producing a rich concentrate. The gold tellurides are floated selectively at a pH of 9, to depress the iron sulfides. During flotation, a frother is used, but no collector is employed.

If a flotation method were to be used on the Piedra Amarilla ore, it would most likely have to take place after removal of the sulfur, but before titanium flotation. Attempts to float the tellurides ahead of sulfur flotation would be futile since sulfur is naturally hydrophobic, and a large portion would float off, complicating the subsequent sulfation steps.

Flotation of sulfur ahead of telluride flotation does have the disadvantage of co-flooding a portion of the tellurides. Much of this telluride fraction may, however, be recovered in the autoclave agglomerate and filter cake. It is also probable that the filter cakes from rutile sulfation may contain additional amounts of gold precipitated from any tellurides which are present in the rutile concentrate and subsequently sulfated along with the rutile.

For purposes of this pre-feasibility study, we have chosen a Knelson centrifugal gravity concentrator as the most promising method for inexpensively recovering the majority of telluride minerals. The Knelson is capable of recovering a significant percentage of the high density minerals. The Knelson is

effective even at the smaller mesh sizes, and in the case of tellurides can recover mineral particles down to about 30 microns. Preliminary ore milling could be done as contemplated at -35 mesh which would effect liberation of the telluride minerals, hopefully with minimal pulverization. The ore would then be classified at -200 mesh, with underflow bypassing the primary gravitational circuit. The larger mesh fractions will contain only a very small percentage of the total telluride mineralization.

A 5 minute grind is sufficient to reduce the ore to -35 mesh, with 52% of the charge passing -200 mesh. To prevent over-grinding of the telluride fraction, it may be necessary to allow only 2 or 3 minutes residence time in the milling circuit. Mill feed from the secondary crushing system has 23.3% passing -200 mesh. At three minutes, 41.8% of the ore passes -200 mesh.

Using -35 mesh feed would be ideal as it would permit gold circuit tailings to report directly to the primary sulfur circuit. It would also allow the Knelson concentrators to remove the majority of the tramp iron from the feed caused by wearing of the crushing and grinding media prior to primary sulfur flotation. Considering an iron loss of 0.25 lbs per metric ton of ore during crushing and primary milling, and a daily processing rate of 25,000 tons, total tramp iron entering the circuit is 6,250 lbs per day.

Placement of Knelson concentrators will also follow the secondary and tertiary sulfur grinding circuits. The secondary grinding circuit will mill about 7,000 mtpd to 70% -100 mesh. Some additional telluride mineralization will be recovered at this step, as will tramp iron from the milling. The tertiary grinding circuit will mill about 3,800 mtpd to 80% -200 mesh. Very little telluride mineralization is expected to be recovered at this step with the Knelson concentrators. The tramp iron will, however, be removed from the circuit.

Tramp iron has a tendency to react with the slightly acidic sulfur during flotation, especially if saline or brine water is used. This produces a discoloration of the final sulfur product and may result in either penalties or difficulties in the marketing phase. Since only the -200 mesh fraction of each of the grinding circuits will be processed by the Knelson, the daily total to be processed at each step is shown in Table 19 below.

<u>Grinding Step</u>	<u>Total Milled Ore</u>	<u>+200 Mesh Underflow</u>	<u>-200 Mesh to Knelson</u>
Primary	25,462 mt	12,196 mt	13,266 mt
Secondary	7,048 mt	2,678 mt	4,370 mt
Tertiary	3,873 mt	775 mt	3,098 mt
			=====

Total mtpd processed by Knelson:

20,734 mt

Primary Gravity Concentrates.

The largest Knelson concentrator will process 36 mtph. The concentrators must be shut down for 2 minutes at determined cycle times to allow the captured concentrates to be discharged from the centrifuge. Each Knelson will capture approximately 70 lbs of concentrate during each cycle. For purposes of this pre-feasibility study, we have assumed a one hour cycle time.

A total of 24 concentrators will be in continuous operation, 15 of which process feed from the primary grinding circuit, 5 at the secondary grinding circuit, and 4 at the tertiary grinding circuit. The concentrators can be arranged in batteries of 2 or more, with feed being distributed through a manifold.

At each of the grinding stations, it will be necessary to have additional concentrators on standby. These standby units will receive feed as each of the other units or batteries is shut down at the end of each cycle period for discharge of concentrates. A minimum of 6 standby units would be required, 4 at the primary circuit and one each at the secondary and tertiary circuits. This will permit as much as 15 minutes on a rotating basis for shutdown and discharge of concentrates.

Daily primary gravity concentrates will total 18.3 mtpd. Of this amount, approximately 15% or 2.75 mt is expected to be tramp iron, which can subsequently be removed using a wet magnetic separator. Excluding the tramp iron, the concentrates total 15.6 mt.

Primary Concentrate grade.

Of the 20,734 tpd processed by the concentrators, most of the tertiary mill feed and approximately 50% of the secondary mill feed will have been processed by the Knelson at least twice. The actual tonnage per day processed by the concentrators is approximately 15,500 tpd, which is the -200 mesh fraction classified at each of the milling stations. The adjusted ratio of concentration, then, is about 1,000 : 1.

Initial feed to the grinding circuit averages 4.5 g/t of gold, mostly in the telluride form. By eliminating the +200 mesh fraction after grinding, 38% of the total feed is eliminated prior to gravity concentration, but with only a 10% loss of the total gold values. Recycling the autoclave agglomerate product will introduce additional gold into the feed, and thus the final gravity feed after classification is upgraded to 8.42 g/t of gold. The Knelson concentrators will recover about 70% of the primary gold values into a rougher concentrate. The rougher concentrates will then be processed to remove the tramp iron.

After the primary gold concentrates have been processed with a wet magnetic separator, they will be routed to an additional gravity step for additional upgrading.

The primary concentrates will be fed to a separate Knelson concentrator, producing a cleaner concentrate. Gold recoveries in the cleaner concentrate are expected to average 85%, contained in a final concentrate of one to two metric tons. Cleaner tailings, which will still contain appreciable amounts of gold, along with the residues recovered from sulfur and rutile filter cakes, will be sent to a leaching circuit for further processing.

Cleaner gravity tailings, totalling 13 to 14 tons per day, will contain an average of 1,034 g/ton of gold. The filter cake obtained from the molten sulfur filtration is 139 mtpd. This by-product is projected to contain a minimum of 5% of the total gold content of the feed ore, and could be as high as 10%. Efficient removal of the tellurides following primary milling will keep the gold content of the filter cake at a lower percentage, and help in reducing the selenium and tellurium content of the final sulfur prill product.

The filter cakes from rutile sulfation may contain from 5 to 10% of total gold content of the feed ore. The rutile filter cakes may produce several hundred tons of filtered material each day, averaging from 20 to 30 g/ton of gold.

Gold Processing of Filter Cakes and Cleaner Tailings.

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The sulfur filter cake will average 40 g/ton gold and 25.7% sulfur. Sulfur recovery from the filter cake may involve grinding the filter cake to -325 mesh, followed by flotation at an extreme pH to try and depress the telluride fraction. Since the tellurides were previously recovered with the sulfur during flotation, a different method may have to be employed.

Experiments by Minexco showed that when native sulfur is cooked in boiling sulfuric acid, the sulfur forms a gelatinous mass which tends to agglomerate in a middle layer, and solid particles composed of gangue material decant to the bottom. This method could be used to separate the tellurides from the native sulfur while sulfating them at the same time.

The native sulfur could be bled out as a nearly purified product. The sulfated gangue material would then be filtered and water leached for removal of sulfates and tellurium. The free gold particles could then be leached using a cyanide solution and recovered in a Merrill Crowe unit for subsequent refining.

The rutile filter cakes will have been already sulfated during rutile sulfation. The filtered material needs only to be water leached for removal of sulfates, and then treated with cyanide for dissolution of the free gold. Gold is recovered in a Merrill Crowe unit and sent to the refinery.

Gold recoveries from both the sulfur and rutile filter cakes has been estimated at about 70% for purposes of this study.

Gravity cleaner tailings will need to be sulfated prior to cyanidation. It is presumed that almost all of the gravity cleaner tailings at this point will pass -200 mesh, so that further grinding will not be necessary. Sulfation of tellurides at sizes larger than 200 mesh is only partially successful. Sulfation of gold tellurides below 200 mesh in concentrated sulfuric acid is completed in about 30 minutes at temperatures of 200o C.

Once the cleaner tailings have been sulfated, they will be water leached for removal of sulfates. The tailings can then be leached with a cyanide solution for gold recovery.

Treatment of Cleaner Concentrates.

Because of the high gold content, cleaner concentrates will undergo a slightly different recovery process than the cleaner tailings. The concentrates are oxidized with sulfuric acid at a moderate temperature. The sulfated concentrates are then roasted at a moderate heat in a muffle like furnace. The roasted concentrates are then quenched with water while hot and water leached for removal of sulfates. The leached concentrates are next sent to a dore furnace, for direct smelting. Two slag products are obtained during the smelting process. The first slag, is skimmed after the charge is completely molten, and contains most of the lead, arsenic, iron, and so on.

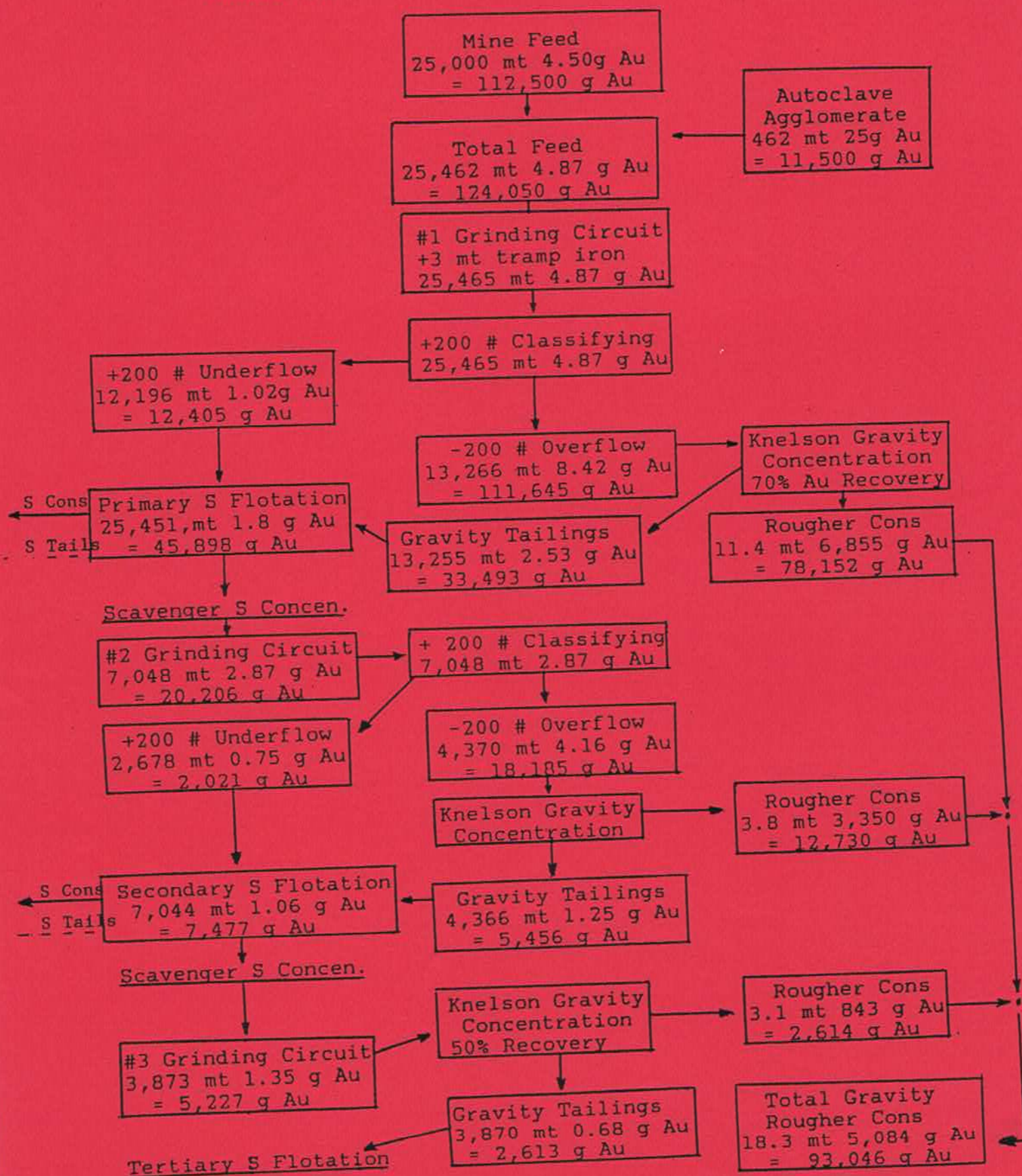
The dore charge then receives an addition of iron or is blown with air, with fused soda ash being used as a flux. The soda slag which forms contains most of the selenium and tellurium remaining in the charge. Prior to casting, the dore metal is cleaned up with Portland cement. The slag from this cleaning is recharged to the furnace with a subsequent charge.

The slag furnace products are ground up and subjected to an alkaline chlorination leach, using sodium carbonate, sodium hydroxide, and calcium hypochlorite as reagents. The resulting slurry is leached with cyanide to dissolve remaining gold and silver. In this stage, any gold and silver dissolved in the chlorination step are converted to the more stable cyanide complexes. The leached slurry is filtered and the solution passes to the Merrill Crowe recovery circuit.

Platinum Group Metals.

Metallurgy for recovery of platinum group metals (PGM) involves roasting the sulfated concentrates at 1,050o C. to convert the PGM to their elemental states, and leaching the roasted concentrates with hydrochloric acid. The pregnant solution is stripped of PGM through cementation with nickel. This process will have to be compatibilized with the gold recovery process. No recoveries of PGM are included in this pre-feasibility study.

FIGURE 31
GRAVITY CIRCUIT FOR RECOVERY OF TELLURIDE GOLD



31 CONTINUED
GOLD RECOVERY FROM GRAVITY CONCENTRATES

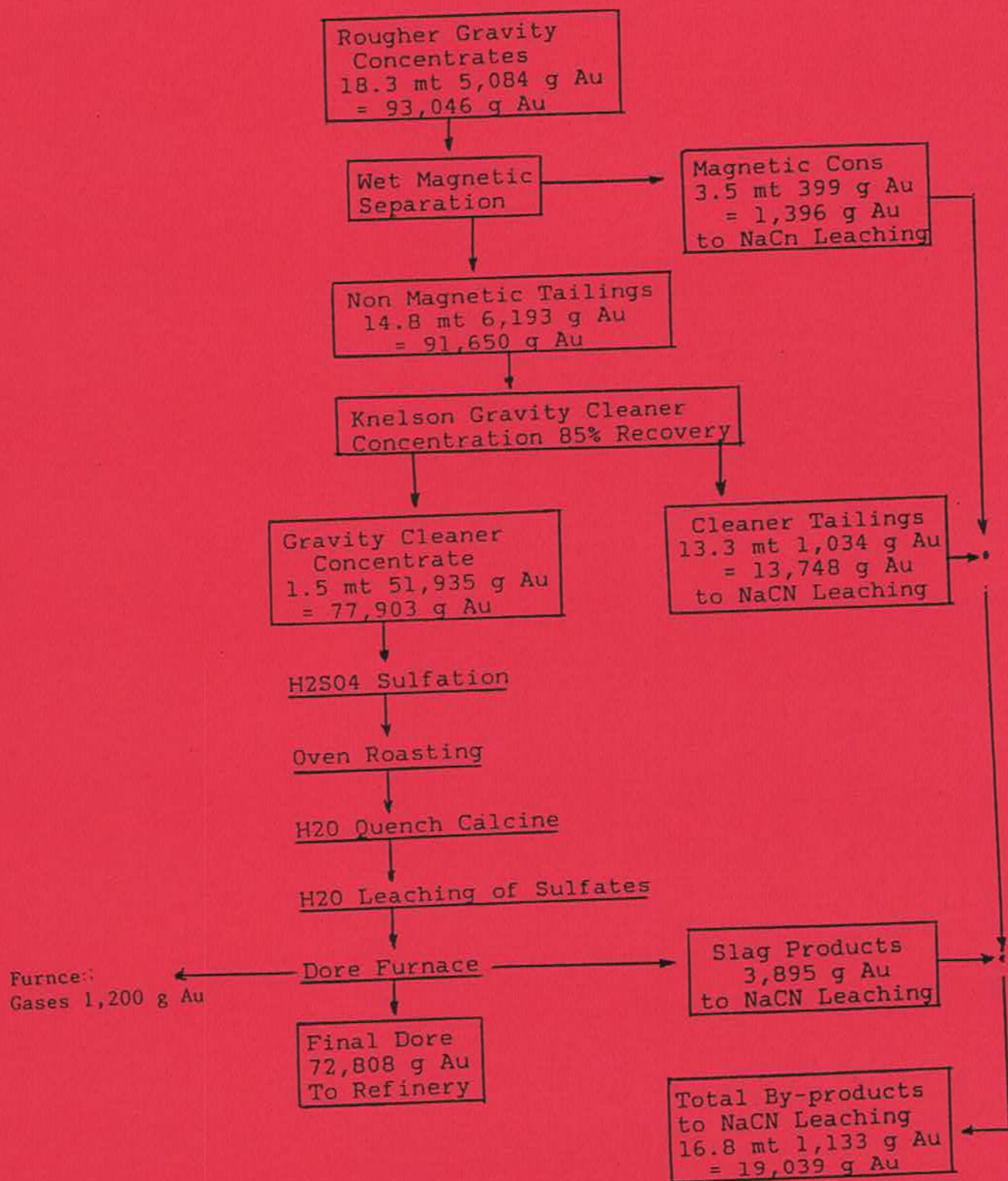
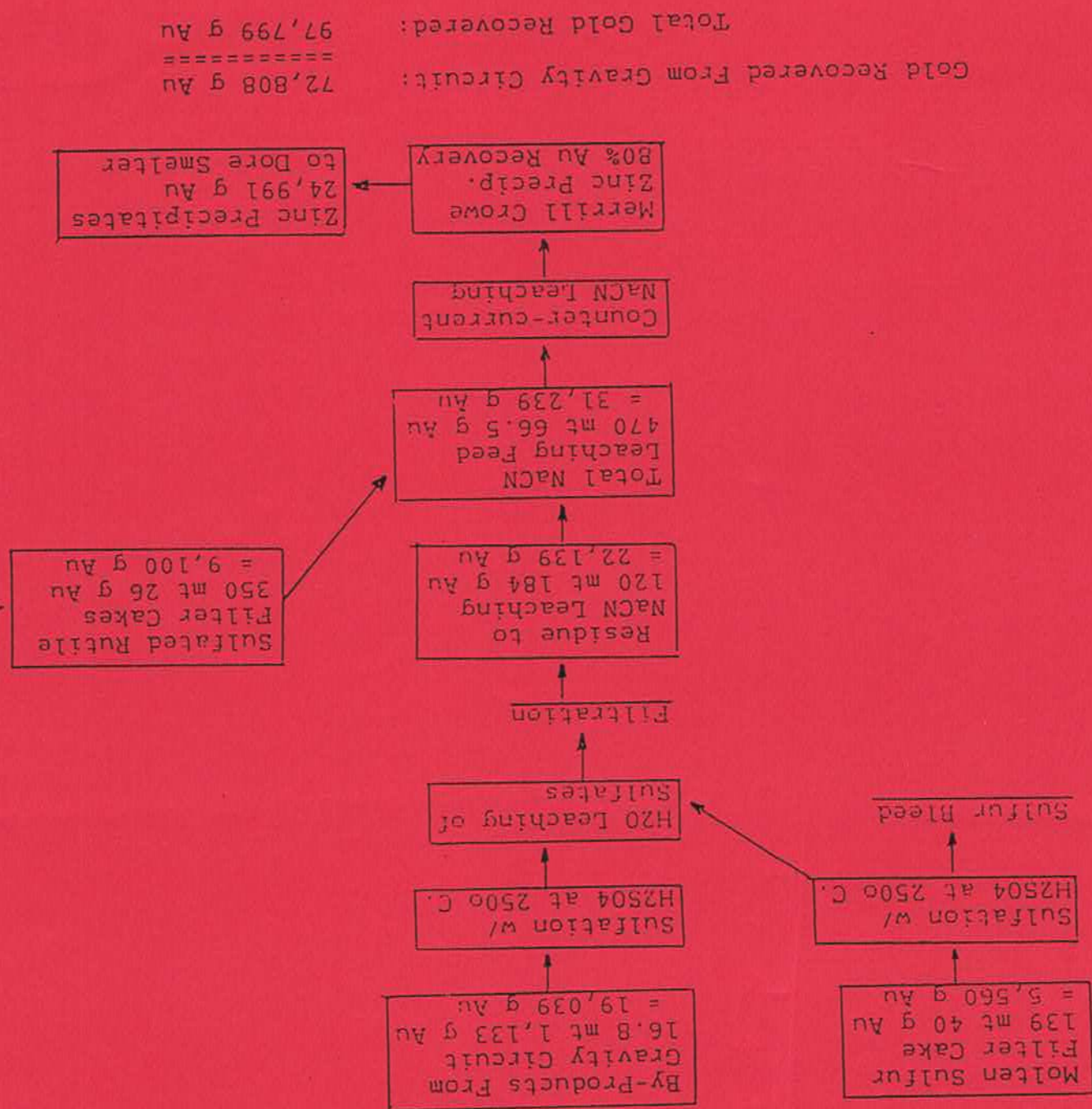


TABLE 32
GOLD LEACHING CIRCUIT



Gold Recovered From Gravity Circuit: 72,808 g Au
 Total Gold Recovered: 97,799 g Au
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