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. Therefor 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

Trench & Sample #	Gold g/t	Silver g/t
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	-	-

TABLE 9 Continued - Gold Samples and Distribution

Fire Assay Results

Trench & Sample #	Gold g/t	Silver g/t
16 - 032	<.01	0.3
16 - 033	<.01	0.4
17 - 034	<.01	0.2
17 - 035	<.01	-
17 - 036	<.01	-

PIT SAMPLES - Fire Assay Results

Pit & Sample #	Gold g/t	Silver g/t
1 - 037	<.01	-
2 - 038	<.01	0.1
3 - 039	.01	0.1
6 - 042	<.01	-
8 - 044	.11	0.3
9 - 045	<.01	0.1
10 - 046	<.01	-
12 - 048	<.01	0.1
14 - 050	.01	0.1

BULK SAMPLES - Fire Assay Results

Bulk Sample #	Gold g/t	Silver g/t
Intec 2 8,000 kilos	0.20	-
SGS 1 400 kilos	0.20	-
Marambio 1 4,000 kilos	0.315	7.0

TABLE 10 - Gold Samples and Distribution

X-ray Fluorescence Results

Composite Samples Trench & Pit #	Au g/t	Ag g/t	Pd g/t	Pt g/t
trenches 1,2,3 - 01	6.28	207.7	-	-
trenches 1,2,3 - 02	1.31	283.6	-	-
trenches 4,5	4.42	689.8	-	-
trenches 6,7	7.28	252.8	-	-
trenches 8,9	6.62	218.3	-	-
trenches 10,11	22.14	93.6	2.55	0.7
trench 14	0.93	43.5	-	-
trenches 15,16,17	6.16	32.0	1.56	0.34
pits 2,3,4	5.22	106.1	-	-
pits 5,6,7	4.64	336.5	2.16	0.04
pits 8,9	4.79	97.3	-	-
pits 11,12,14	8.02	632.0	-	-
pit 13	0.47	327.0	trace	no
pit 15	5.46	551.8	1.84	0.14
Marambio 1 composite leaching	1.75	-	-	-

Figure 18

Gold Ore Reserves, by X-ray Fluorescence - Section 1

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 1

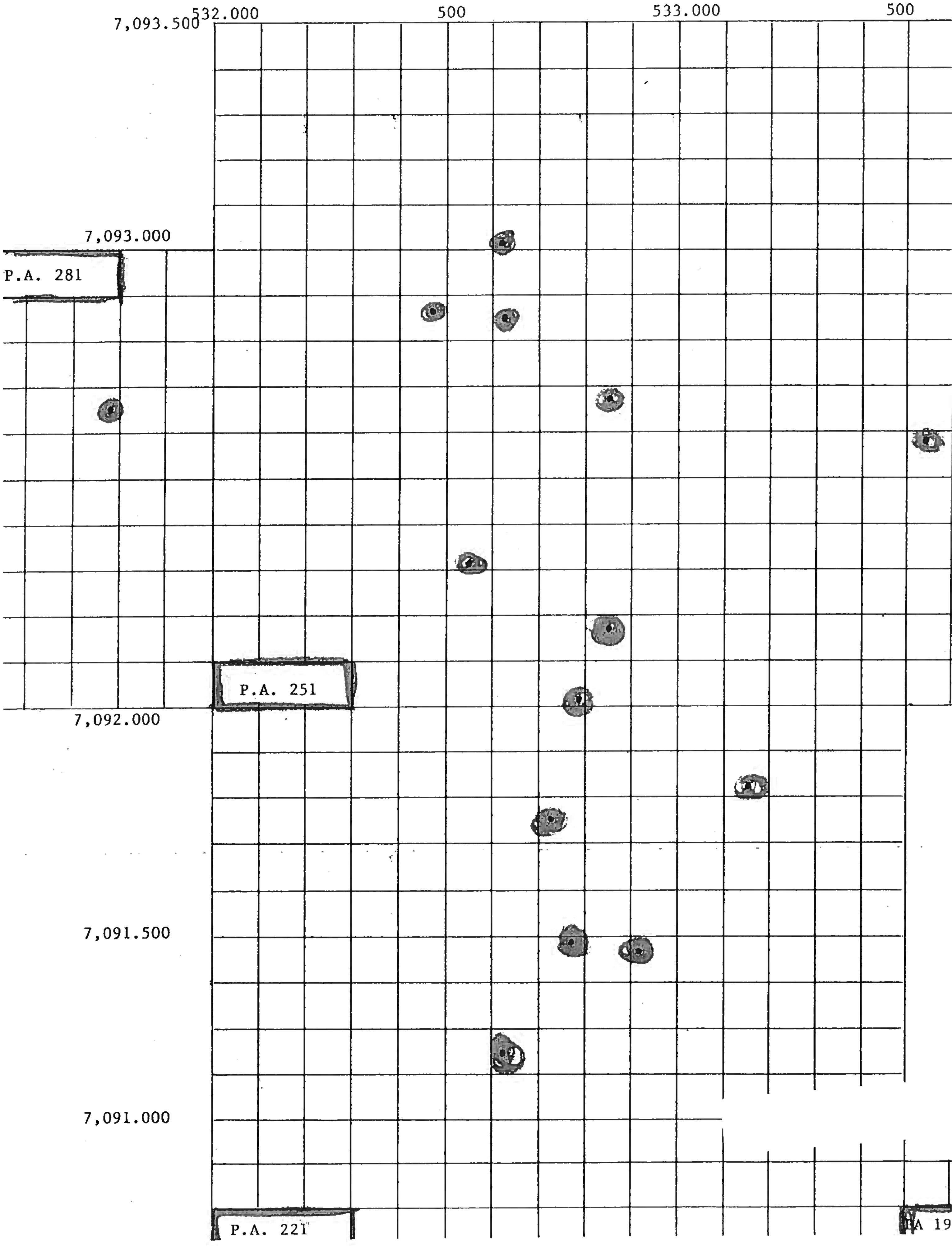
Grade	Metric Tons	Net gold in grams
0.47 g/t	480,000	22,560
4.36 g/t	4,320,000	18,819,360
8.02 g/t	1,440,000	11,548,800
5.27 g/t (ave)	5,760,000	30,368,160

Inferred Ore Tonnage - Section 1

Grade	Metric Tons	Net gold in grams
5.27 g/t	37,920,000	230,193,600

Total Inferred and Indicated Ore - Section 1

Grade	Metric Tons	Net gold in grams
5.27 g/t	43,680,000	260,561,760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533.000
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7,092.000
P.A. 251
7,091.500
7,091.000
P.A. 221
A 19

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

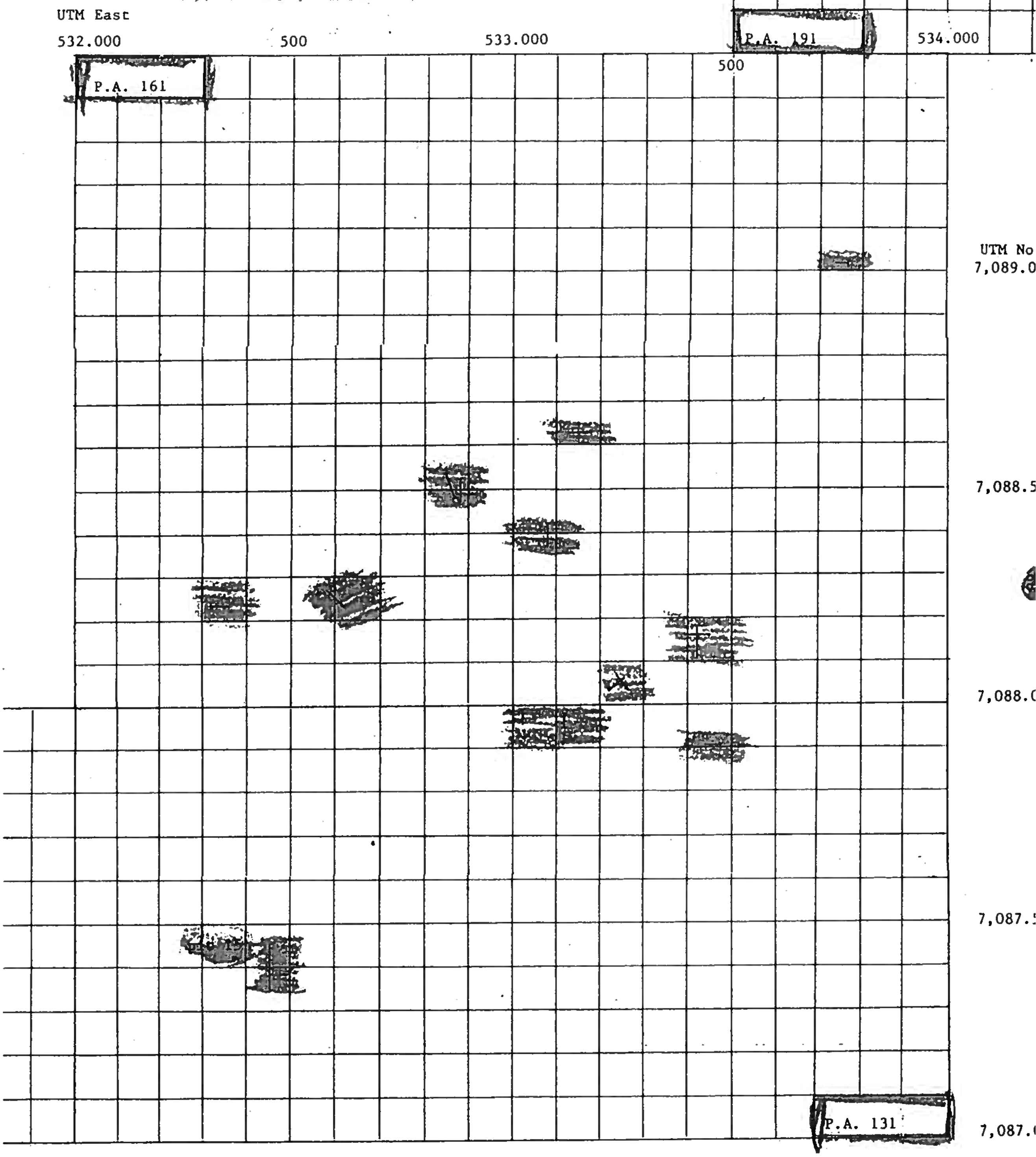
Inferred Ore Tonnage - Section 2

Grade	Metric Tons	Net gold in grams
7.50 g/t	15,600,000	117,000,000

Total Inferred and Indicated Ore - Section 1

Grade	Metric Tons	Net gold in grams
7.50 g/t	21,840,000	163,773,600

PIEDRA AMARILLA TRENCHING SCHEMATIC



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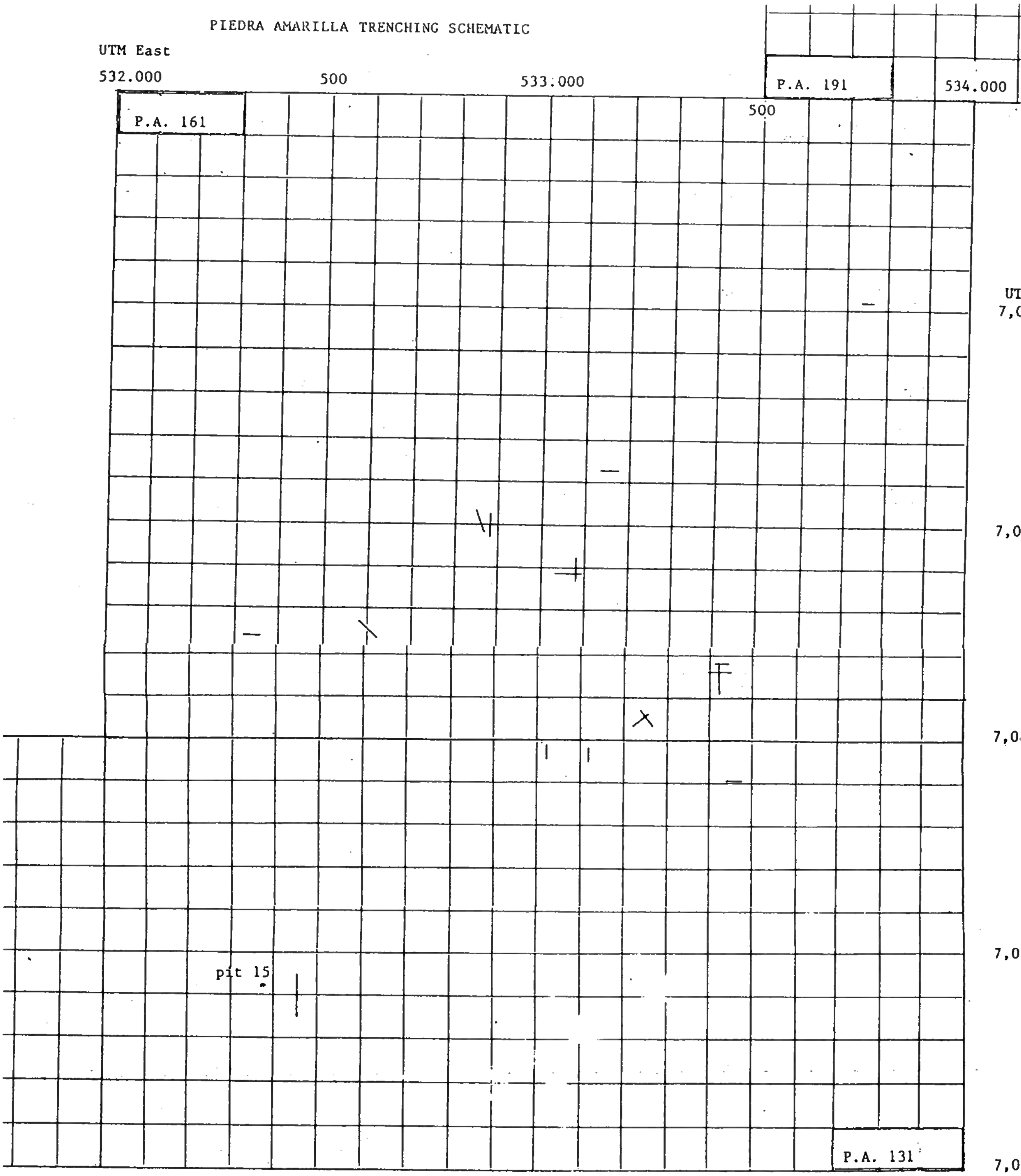

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Gold Ore Reserves, X-ray fluorescence - Sections 2 and 3 continued

Indicated Ore Tonnage - Section 3

Grade	Metric Tons	Net gold in grams
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

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

Total Inferred and Indicated Ore - Section 3

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

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

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

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

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

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

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

Titanium Reserves.

Blocked out titanium reserves in the form of rutile (TiO_2) include indicated reserves of 18,960,000 metric tons grading 3.63% TiO_2, and inferred reserves of 93,760,000 metric tons grading 3.11% TiO_2 (see figures 20-21, table 11). Combined indicated and inferred reserves total 112,720,000 metric tons grading 3.20% TiO_2, which corresponds closely to the average of the bulk samples assayed by Intec and SGS (3.04% TiO_2).

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.

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

Titanium Byproducts.

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, $SrSo_4$, while the dominant mineral in the Salar is likely strontianite, $SrCO_3$.

The average content of celestine, $SrSO_4$, would be approximately 2.1 times that of the strontium content alone, or roughly 0.63%. Total reserves of celestine, $SrSO_4$, in the blocked out titanium reserves of Piedra Amarilla are estimated at 710,136 metric tons.

Silica.

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.

Total Reserves.

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

TABLE 11 - Rutile Samples and Distribution

Composite Samples Trench & Pit #	TiO2 %	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	-

TABLE 11 Continued - Rutile Samples and Distribution

Composite Samples Trench & Pit #	TiO2 %	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 #	TiO2 %
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

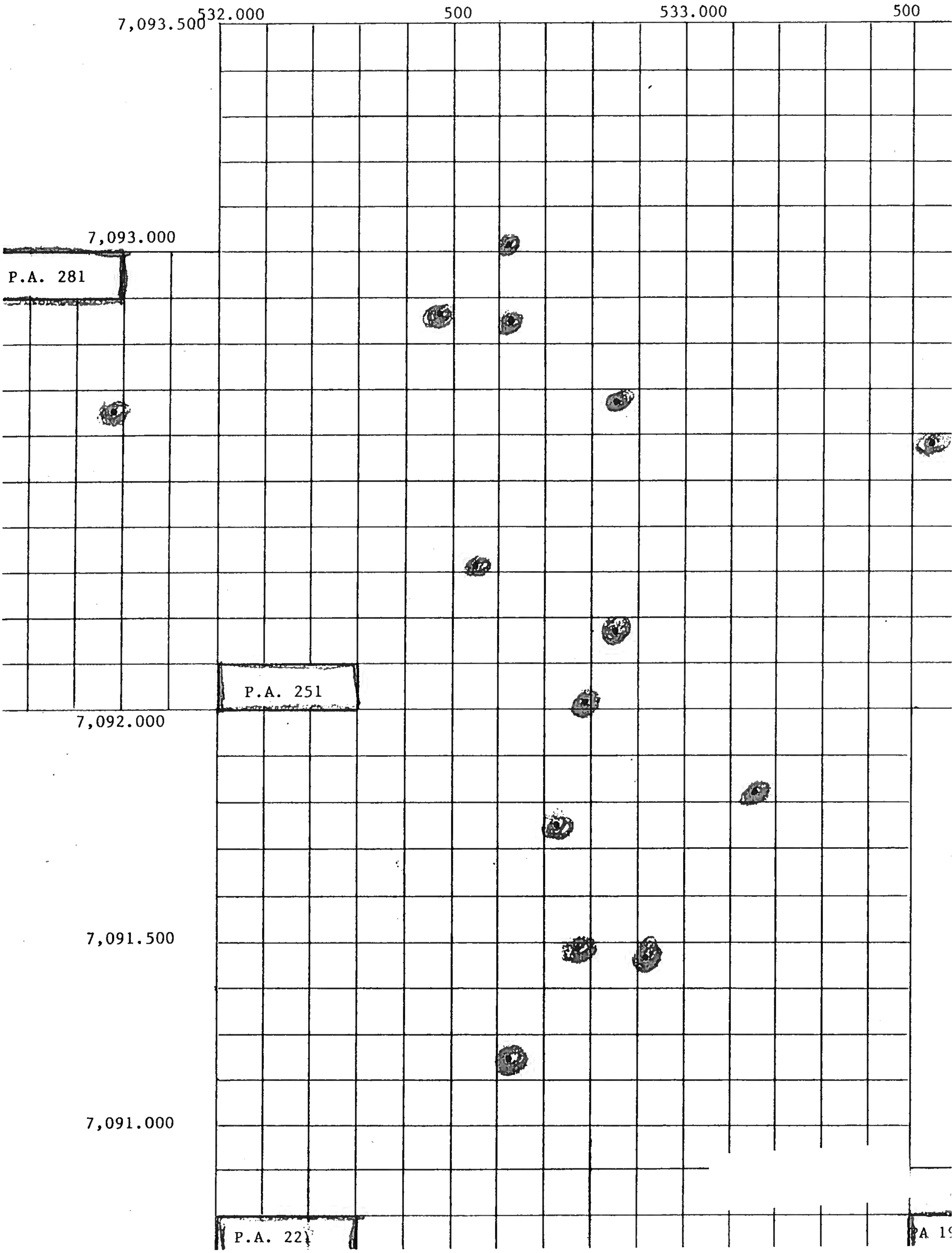
Grade	Metric Tons	Net TiO2 in Metric Tons
0.50%	480,000	2,400
1.81%	2,400,000	43,440
2.18%	1,440,000	31,392
5.13%	1,440,000	73,872
======	==========	=========
2.62%	5,760,000	151,104

Inferred Ore Tonnage - Section 1

Grade	Metric Tons	Net TiO2 in Metric Tons
2.62%	35,520,000	1,081,536

Total Inferred and Indicated Ore - Section 1

Grade	Metric Tons	Net TiO2 in Metric Tons
2.62%	41,280,000	1,232,640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P.A. 221
PA 19

Figure 21

Rutile (TiO_2) 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

Grade	Metric Tons	Net TiO_2 in Metric Tons
1.12%	3,120,000	35,088
1.68%	2,880,000	48,384
======	==========	=========
1.39%	6,000,000	83,472

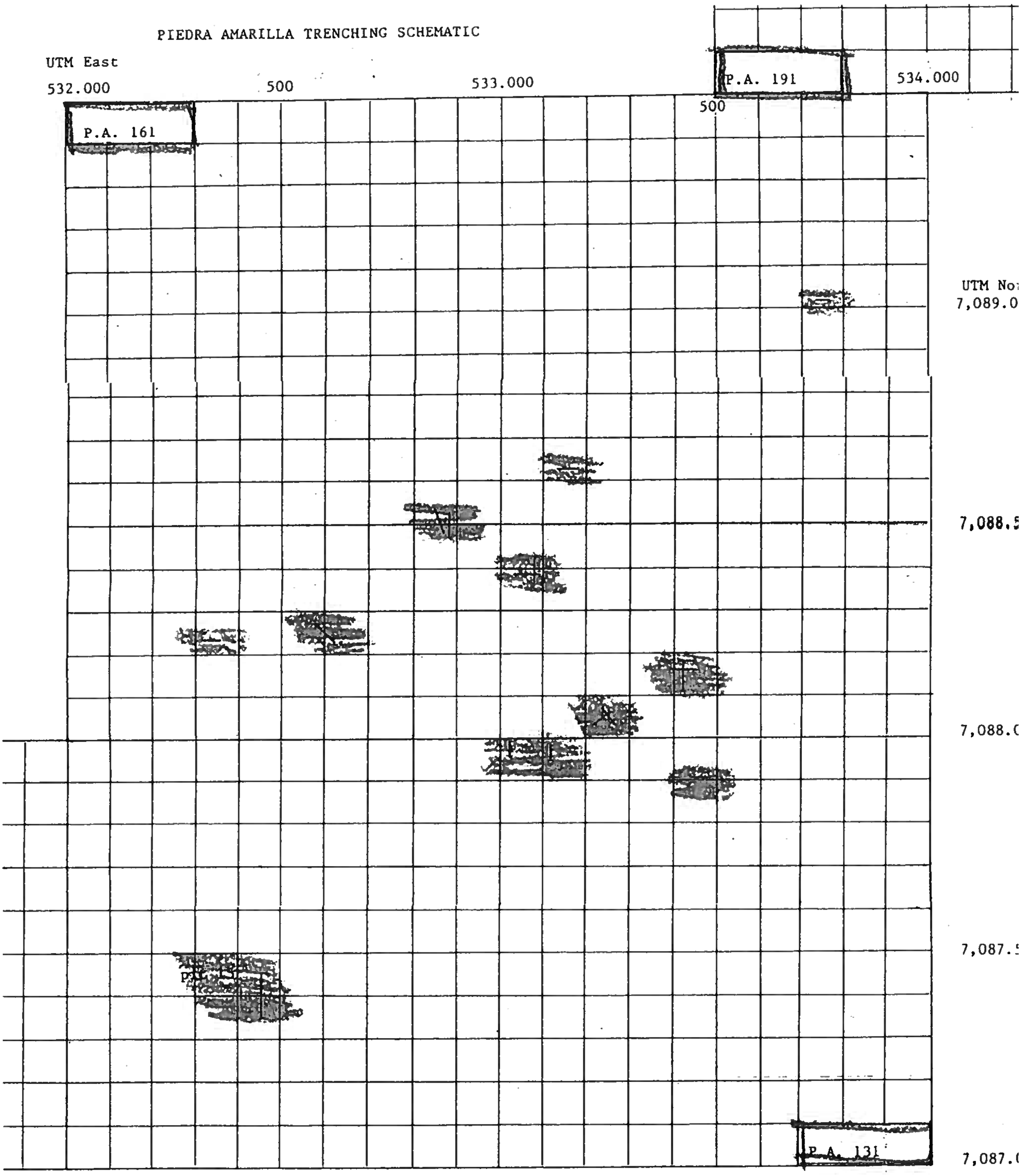
Inferred Ore Tonnage - Section 2

Grade	Metric Tons	Net TiO_2 in Metric Tons
1.39%	15,840,000	220,176

Total Inferred and Indicated Ore - Section 2

Grade	Metric Tons	Net TiO_2 in Metric Tons
1.39%	21,840,000	303,648

PIEDRA AMARILLA TRENCHING SCHEMATIC



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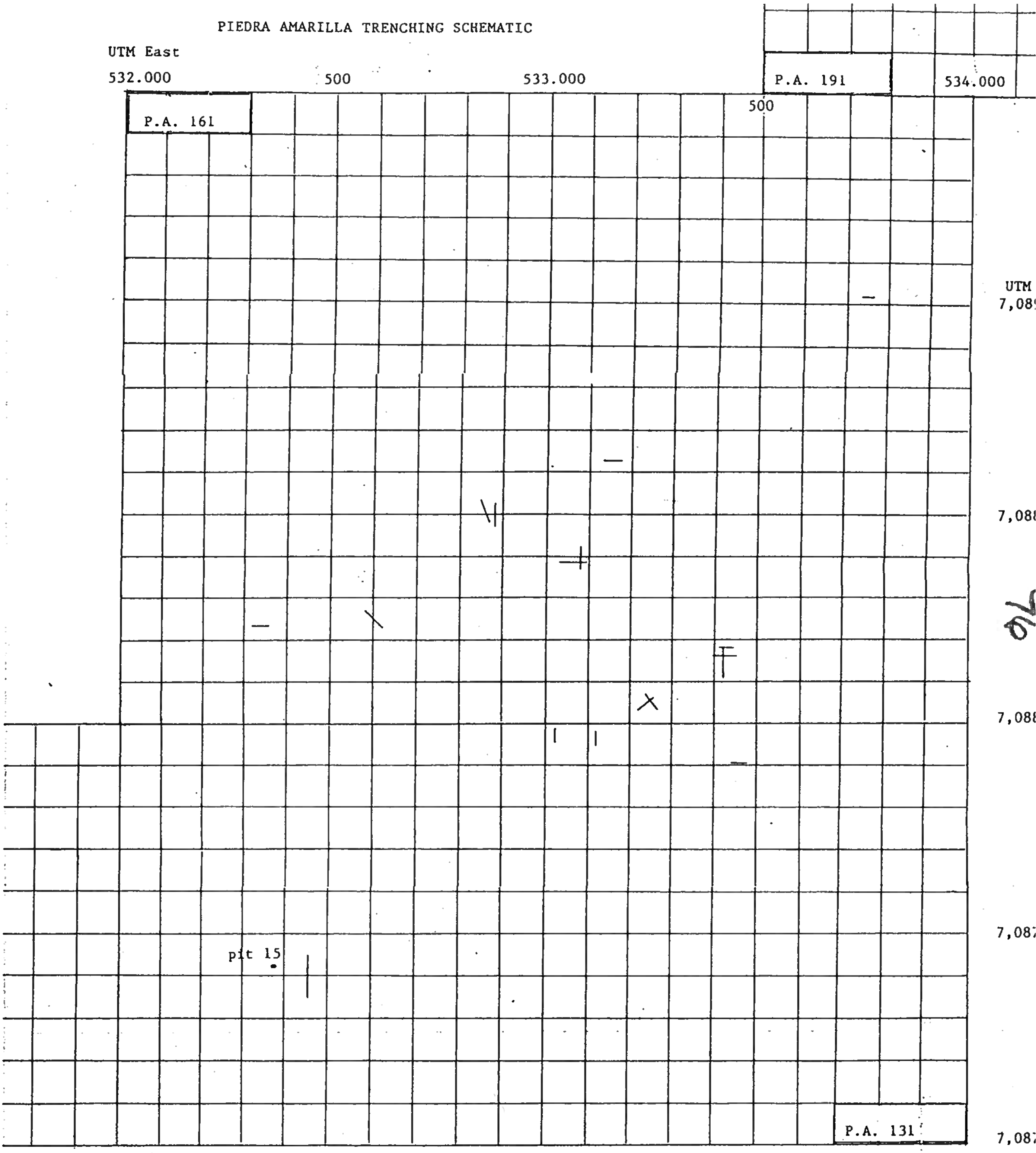

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Rutile (Ti02) Ore Reserves - Sections 2 and 3 continued

Indicated Ore Tonnage - Section 3

Grade	Metric Tons	Net Ti02 in Metric Tons
6.30%	7,200,000	454,160

Inferred Ore Tonnage - Section 3

Grade	Metric Tons	Net Ti02 in Metric Tons
3.8%	42,400,000	1,611,000

Total Inferred and Indicated Ore - Section 3

Grade	Metric Tons	Net Ti02 in Metric Tons
4.16%	49,600,000	2,065,360

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

Grade	Metric Tons	Net Ti02 in Metric Tons
3.63%	18,960,000	688,736

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

Grade	Metric Tons	Net Ti02 in Metric Tons
3.11%	93,760,000	2,912,912

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

Grade	Metric Tons	Net Ti02 in Metric Tons
3.20%	112,720,000	3,601,648

Table 12

Mineral Reserve Summary

Mineral	Indicated Reserves 000 M tons	Inferred Reserves 000 M tons	Total Reserves 000 M tons	Net mineral Reserve 000
Sulfur	21,120	79,760	100,880	12,692 mt
- grade %	19.84%	10.66%	12.58%	
Selenium	21,120	79,760	100,880	3.081 mt
- grade g/t	30.54 g/t	30.54 g/t	30.54 g/t	
Tellurium	21,120	79,760	100,880	0.697 mt
- grade g/t	6.91 g/t	6.91 g/t	6.91 g/t	
Kaolinite	21,120	79,760	100,880	2,219 mt
- grade %	2.2%	2.2%	2.2%	
Gold	19,200	96,720	115,920	522,117 g
- grade g/t	4.74 g/t	4.46 g/t	4.50 g/t	
Rutile TiO2	18,960	93,760	112,720	3,602 mt
- grade %	3.63%	3.11%	3.20 %	
Celestine	18,960	93,760	112,720	710 mt
- grade %	0.63%	0.63%	0.63%	

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 overdesigned 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.

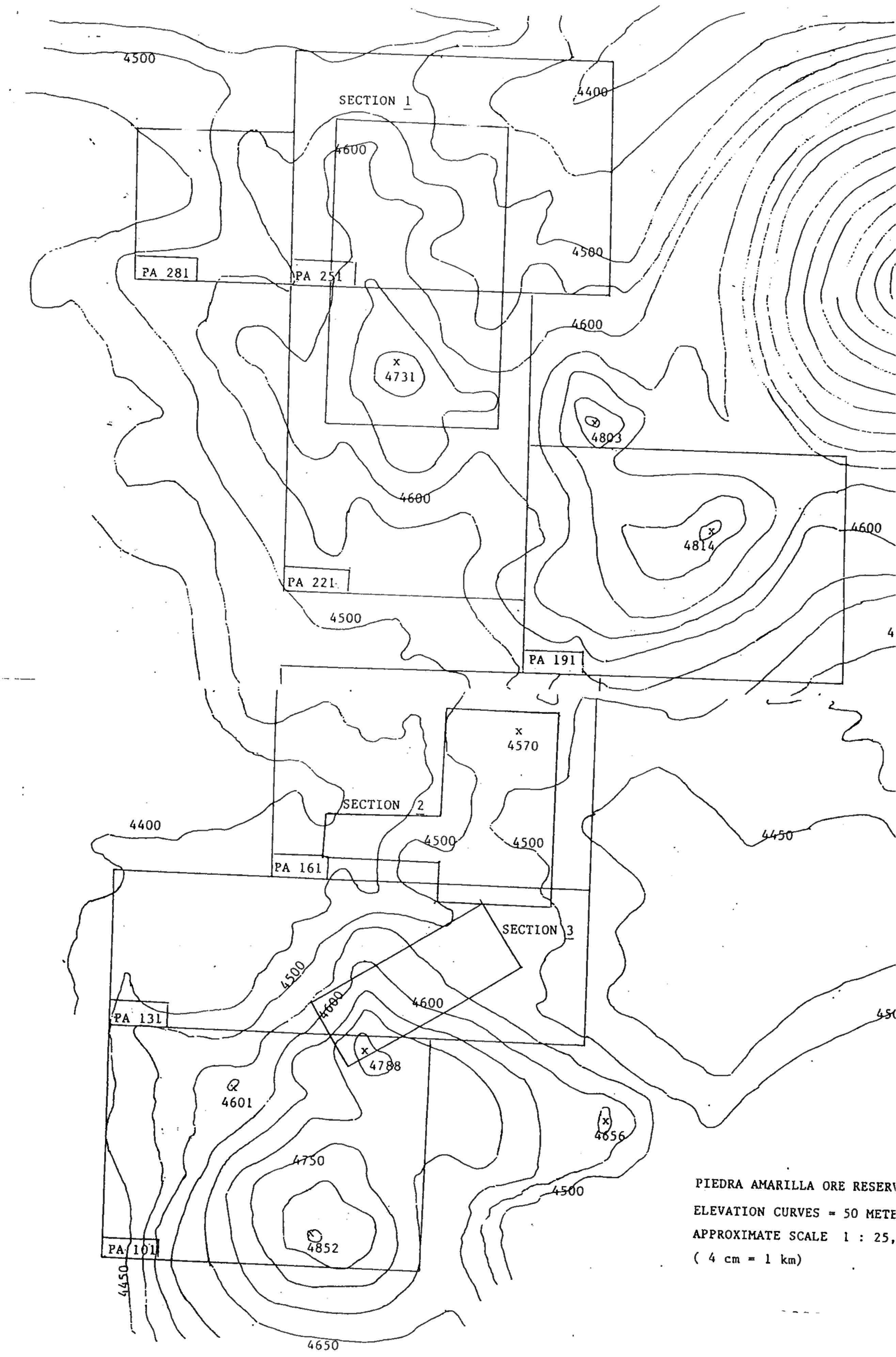
Elevation curves showing altitude changes are plotted on Figure 22, along with the different reserve areas, which will form the nucleus of the proposed pit areas. Elevation changes in the Piedra Amarilla group are generally less pronounced from north to south than they are from east to west.

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.



PIEDRA AMARILLA ORE RESERV

ELEVATION CURVES = 50 METE

APPROXIMATE SCALE 1 : 25,

(4 cm = 1 km)

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.

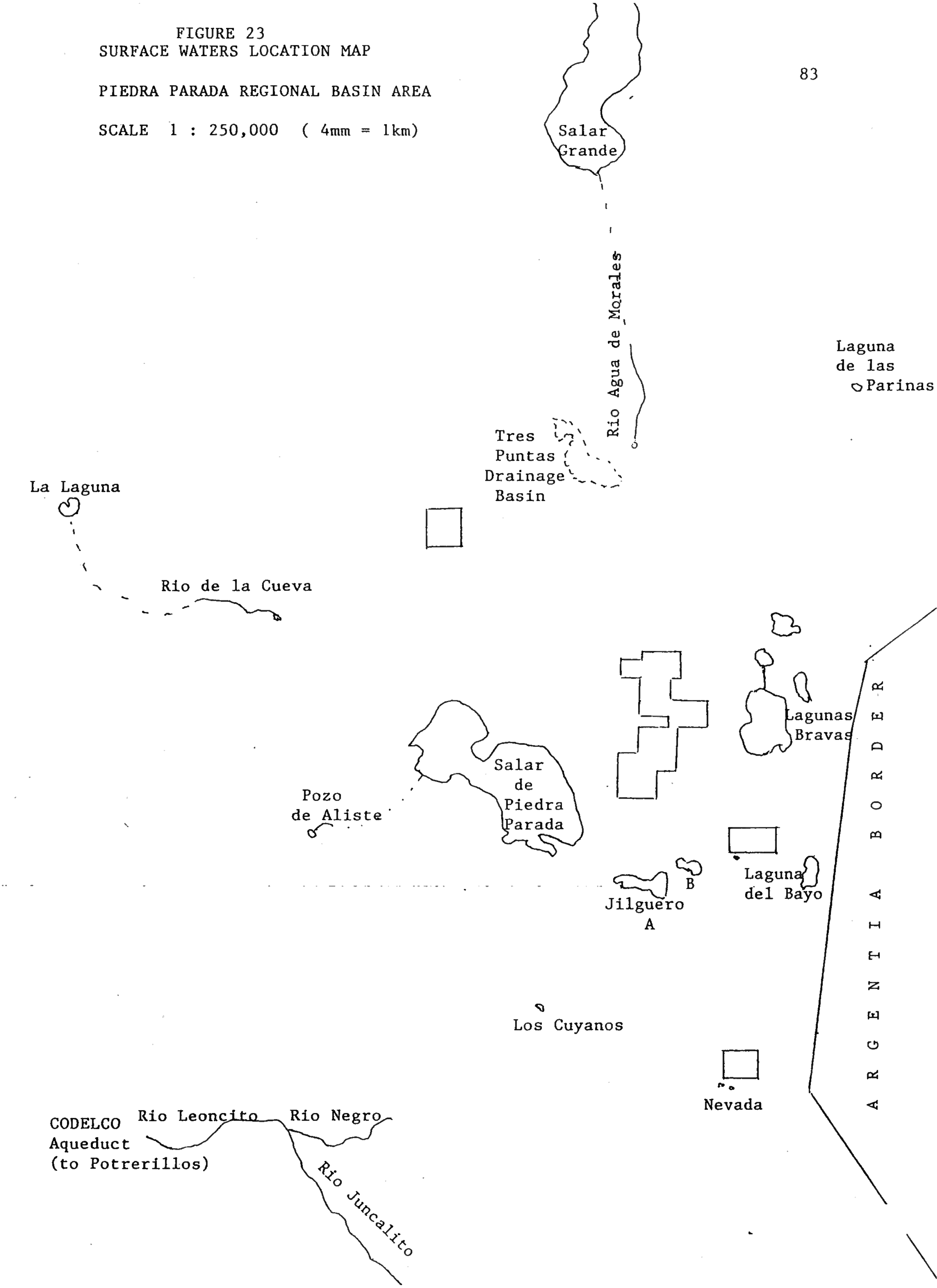
The northern portion of Chile is dominated by the Atacama desert from the coast to the pre-cordillera. Considered the world's driest desert, it owes the lack of precipitation to two basic climatic features.

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.

FIGURE 23
SURFACE WATERS LOCATION MAP

PIEDRA PARADA REGIONAL BASIN AREA

SCALE 1 : 250,000 (4mm = 1km)



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 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.

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.

FIGURE 24
SURFACE WATERS LOCATION MAP

PIEDRA PARADA REGIONAL BASIN AREA

SCALE 1 : 250,000 (4mm = 1km)

ARROWS INDICATE SURFACE FLOW

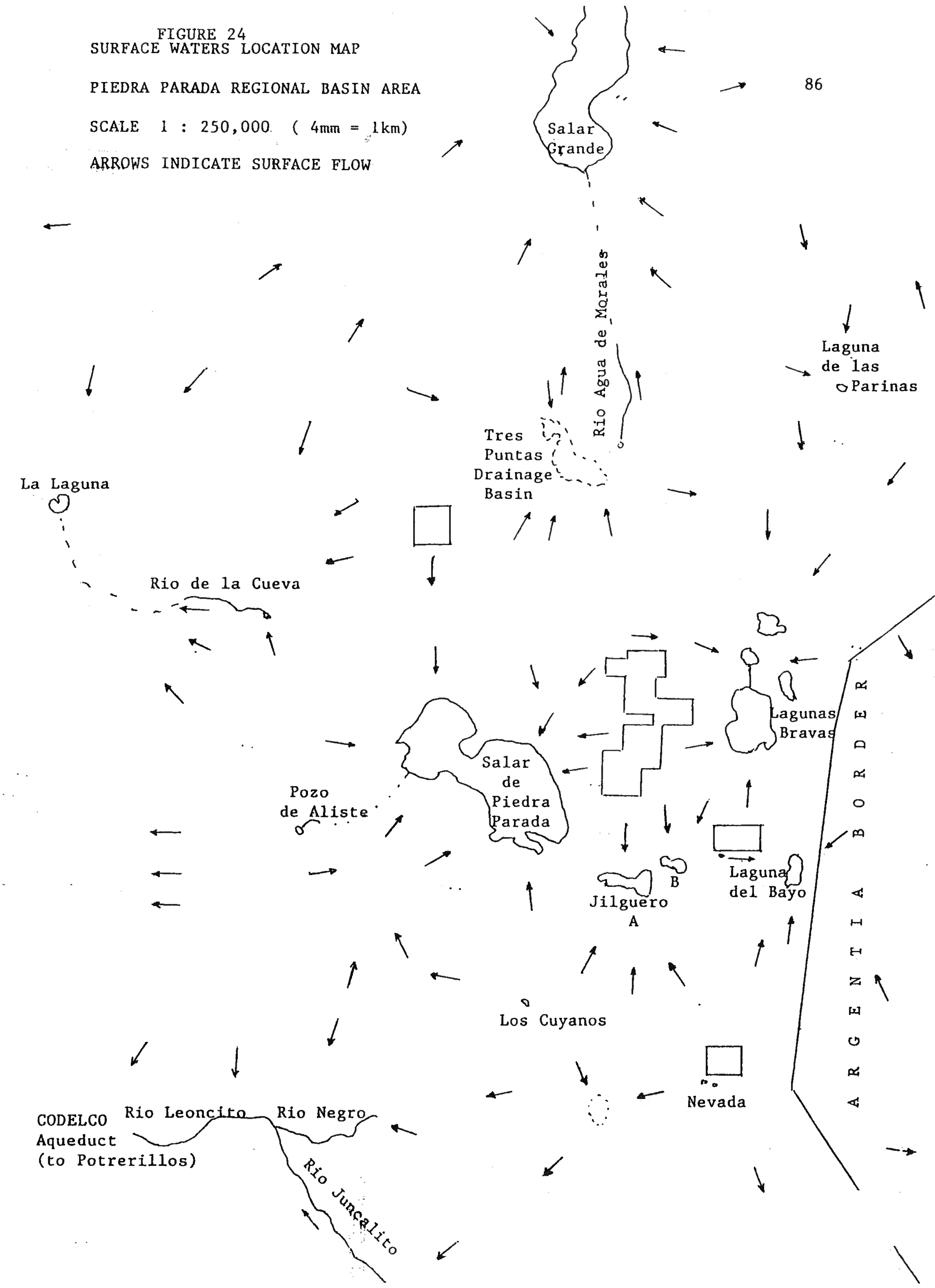
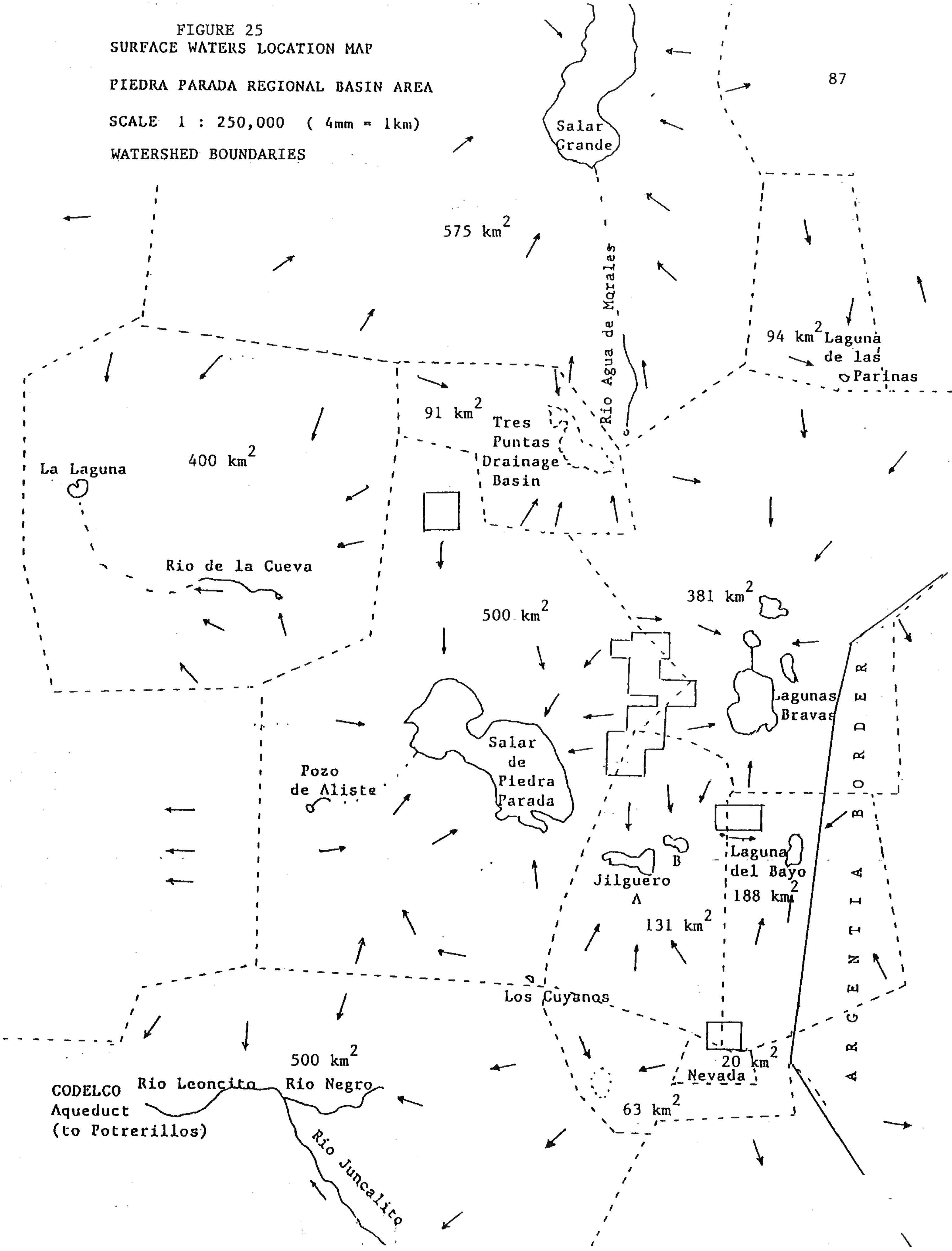


FIGURE 25
SURFACE WATERS LOCATION MAP

PIEDRA PARADA REGIONAL BASIN AREA

SCALE 1 : 250,000 (4mm = 1km)

WATERSHED BOUNDARIES



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 13

Chemical Analysis of Surface Lakes

Mineral Content (in ppm)	Laguna Brava	Laguna Jilgero A	Laguna Jilgero B	Laguna Bayo
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 CaCO3	6,809	1,043	708	224
Hardness Ca as CaCO3	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.

Table 14

Chemical Analysis of Rio de la Cueva

Mineral Content	in ppm
Chlorine	1,027
Sodium	715
Sulfates	550
Magnesium	61
Calcium	303
Hardness Mg as CaC03	248
Hardness Ca as CaC03	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

Table 15

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

Mineral Content (in ppm)	Paipote 1 La Junta	Paipote 2 P.Grandes	Paipote 3 La Puerta	Vega Paton
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

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%+ $TiO2$ 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.

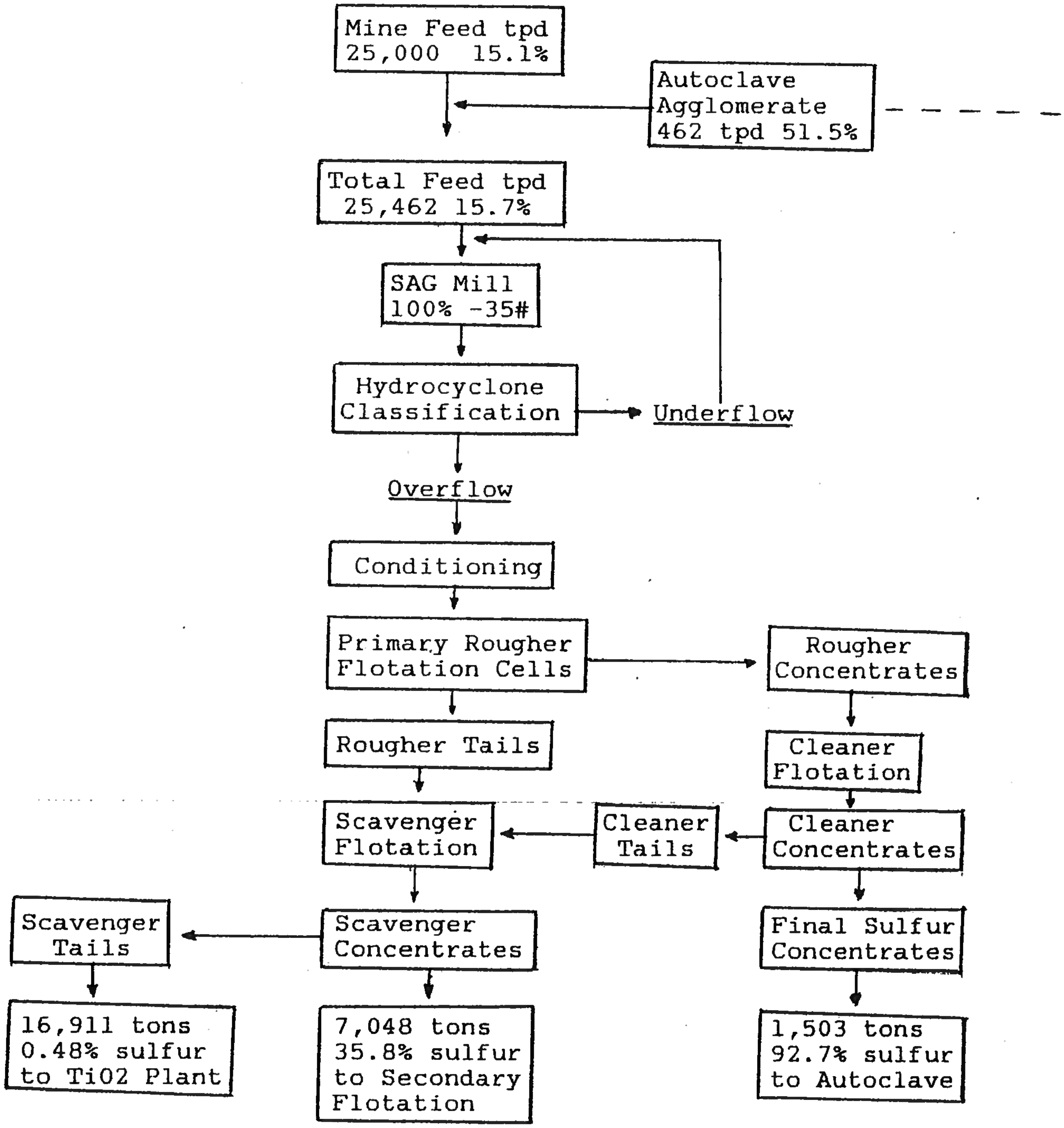
Sulfur Flowsheet.

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.

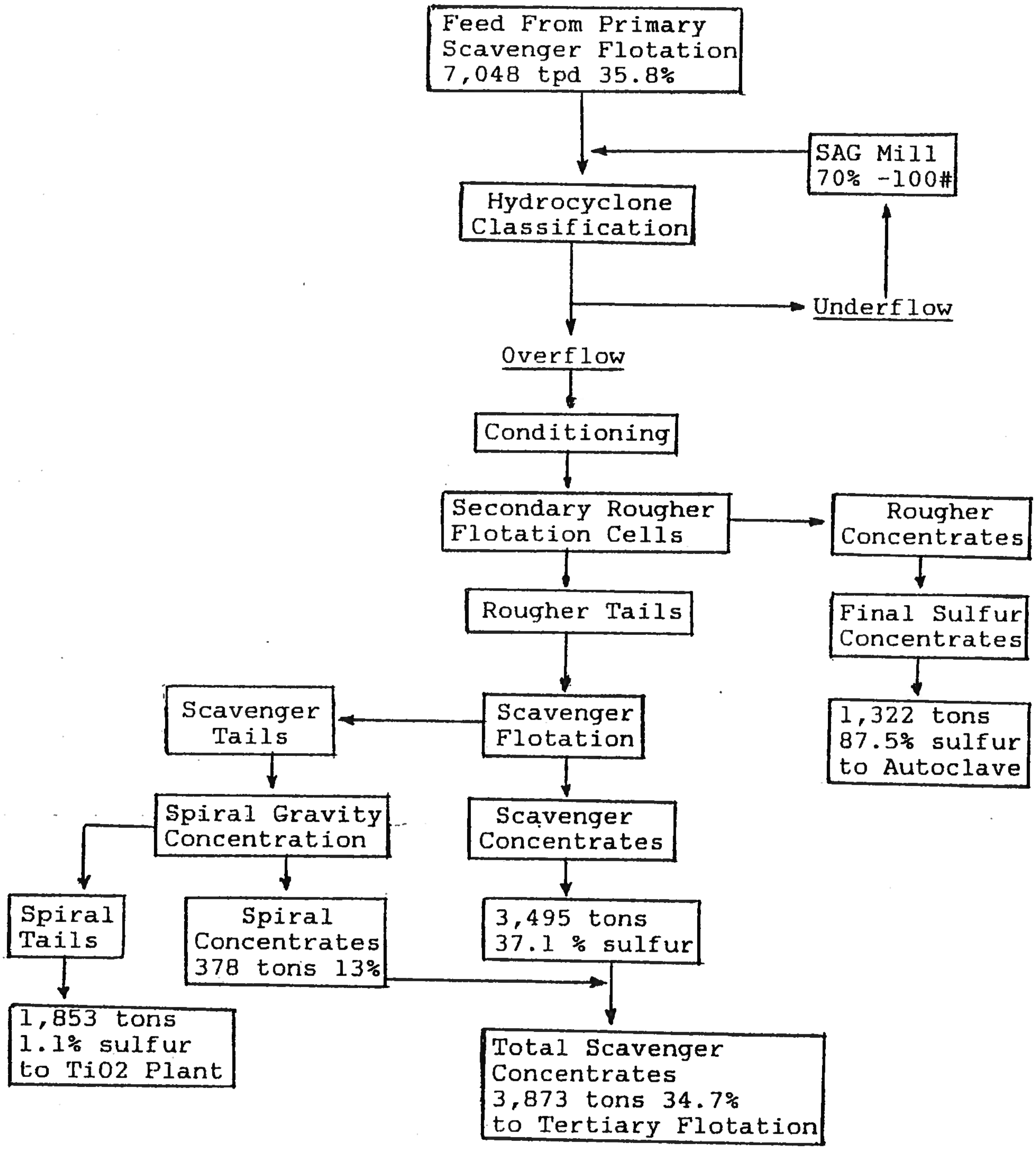
METALLURGICAL STUDIES
TABLE 16

Date	Title of Study		Lab/Sponsor
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 Experience 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 Characterization 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	Ti02 Concentration Study Proposal	5pp	SGS Santiago

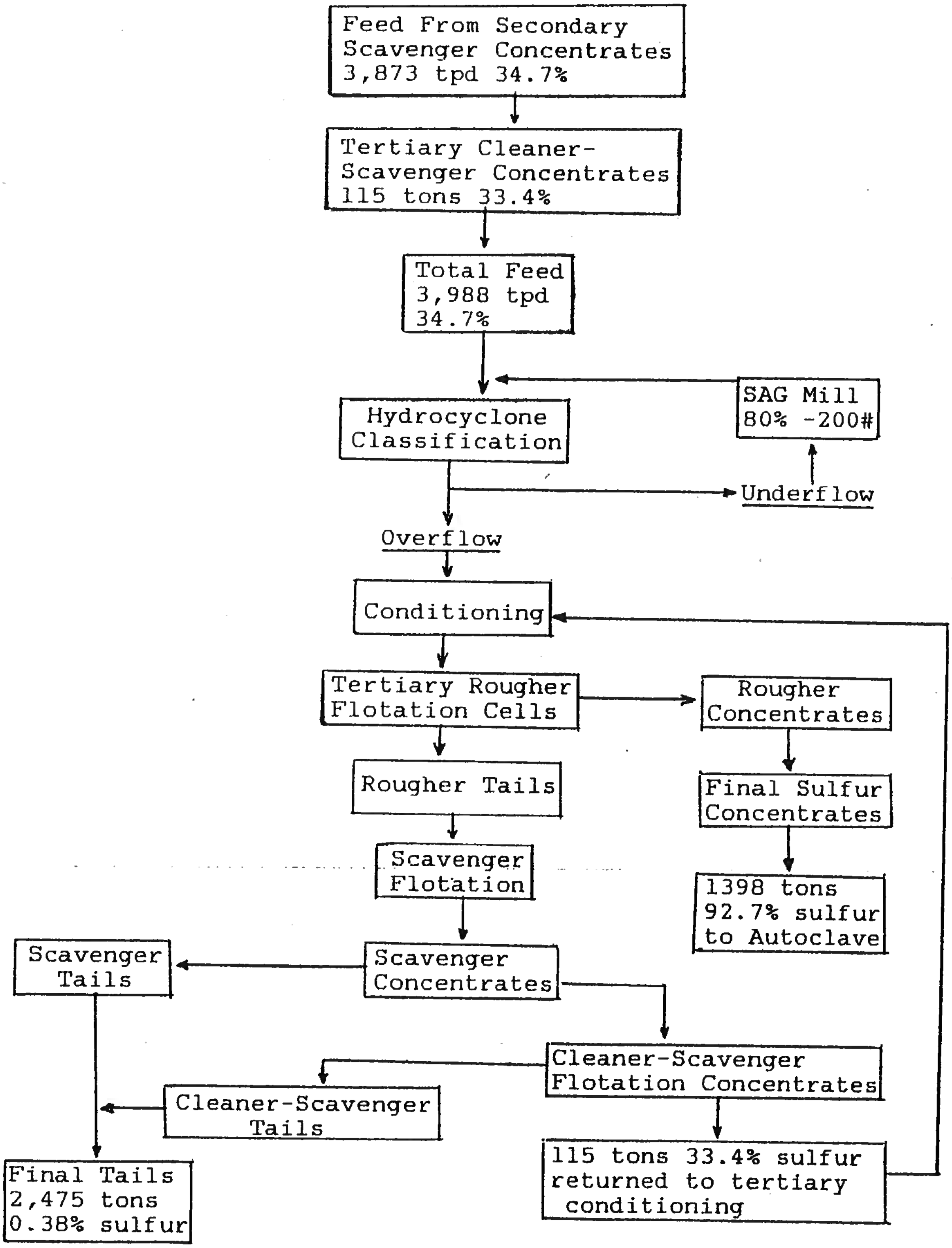
SULFUR METALLURGICAL FLOWSHEET
PRIMARY FLOTATION



SECONDARY FLOTATION



Tertiary Flotation



Melted sulfur is formed using standard prill technology.

Primary Flotation.

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.

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 predetermined temperature (+/- 130o 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.

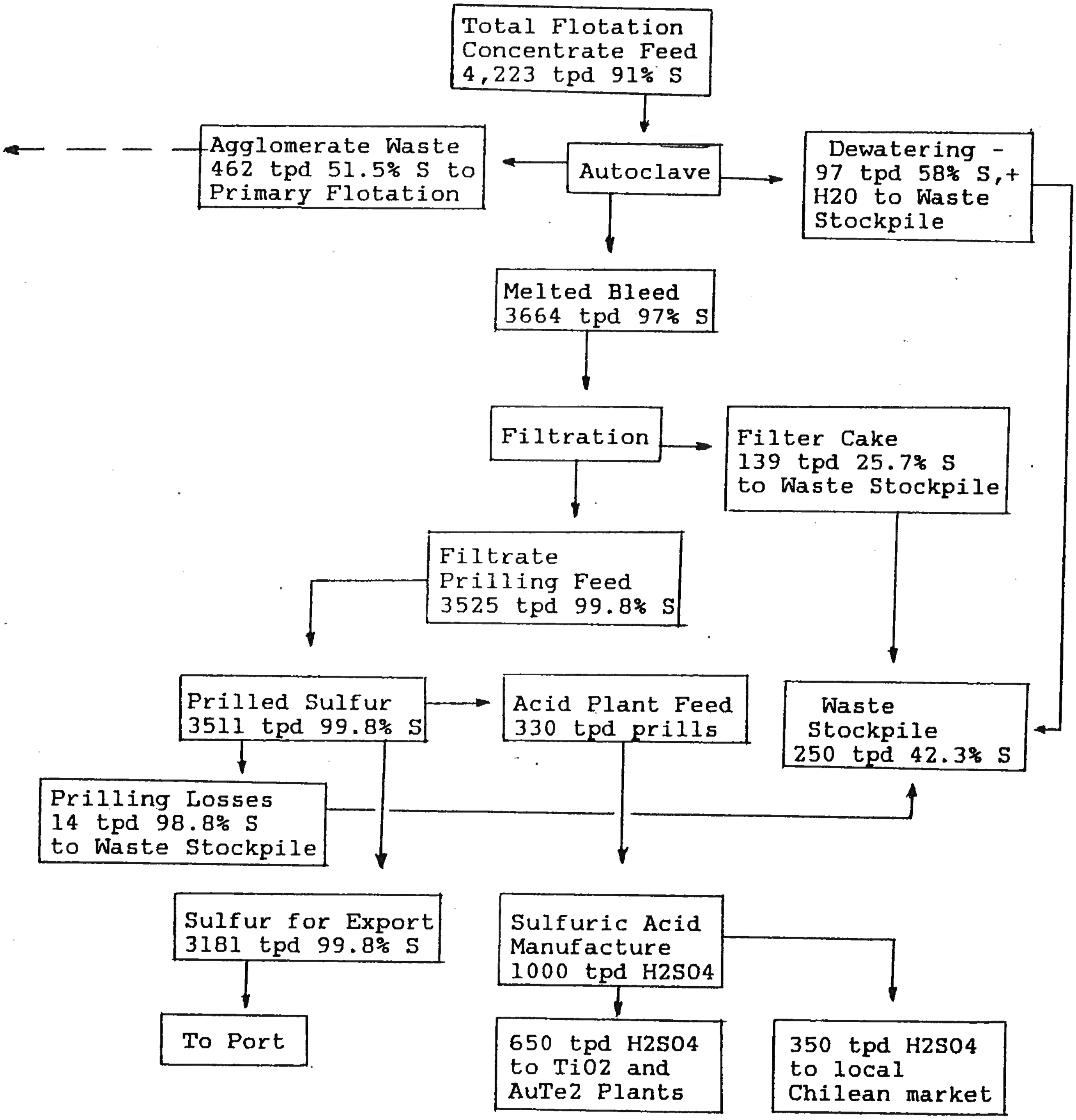
The bottom layer receives the unmelted solid particles, consisting largely of silicates, which because of their heavier specific gravity decant. The bleed from this bottom layer forms the agglomerate waste. The molten sulfur which is bled along with the silicate particles will cool and solidify, forming 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 $H2SO4$ 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 $TiO2$ 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 S02 into S03. This catalyst usually is vanadium pentoxide, although platinum is sometimes used.

Impurities such as arsenious oxide (As203) 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.

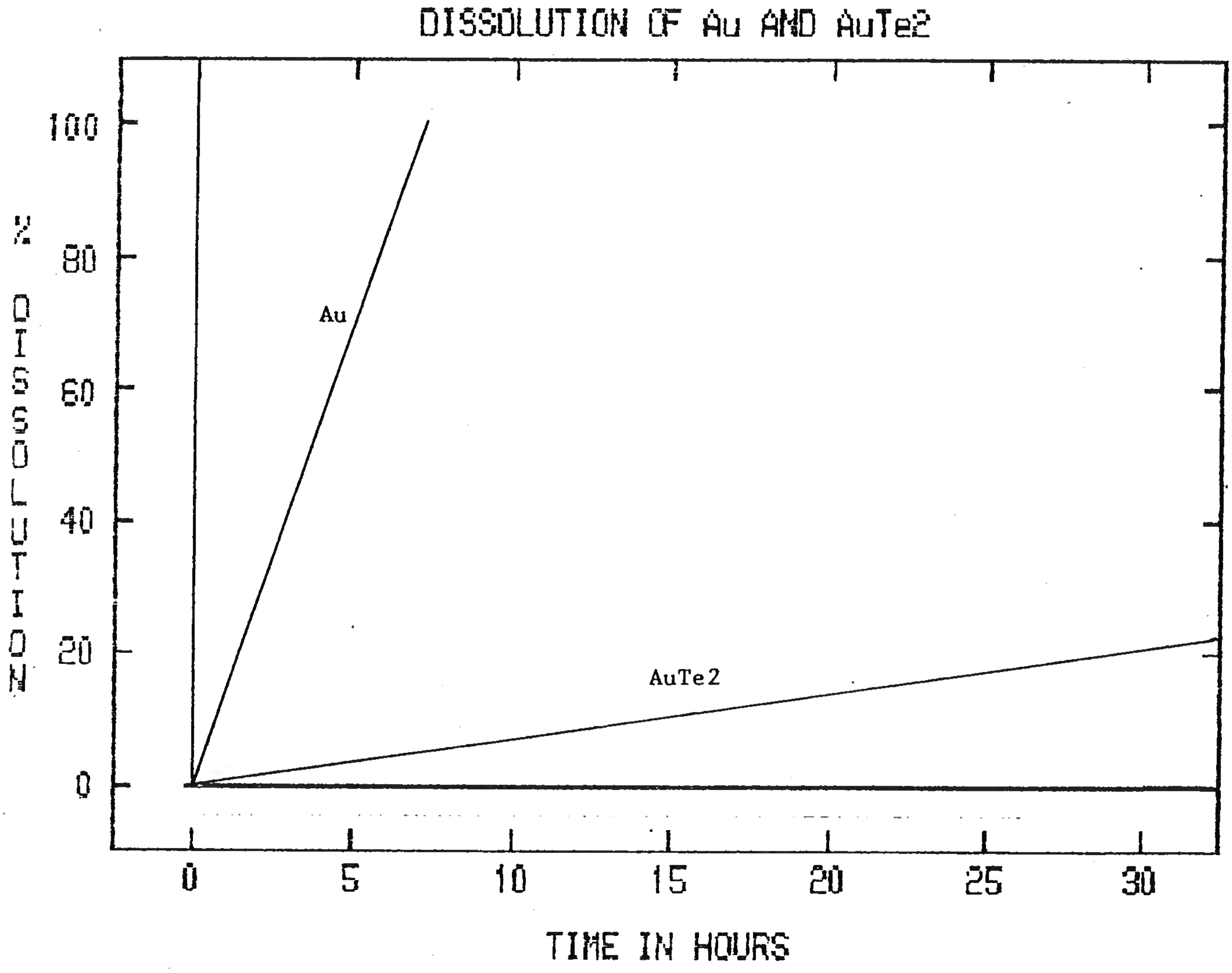
The pregnant solution was treated to precipitate the gold out of solution. Only 28% of the gold values originally leached into the solution, or 0.35 g/t, precipitated and was recovered. This indicates that the gold telluride must be oxidized prior to leaching in order to liberate the leach solution.

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



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

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-floating 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.

Grinding Step	Total Milled Ore	+200 Mesh Underflow	-200 Mesh to Knelson
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.

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.

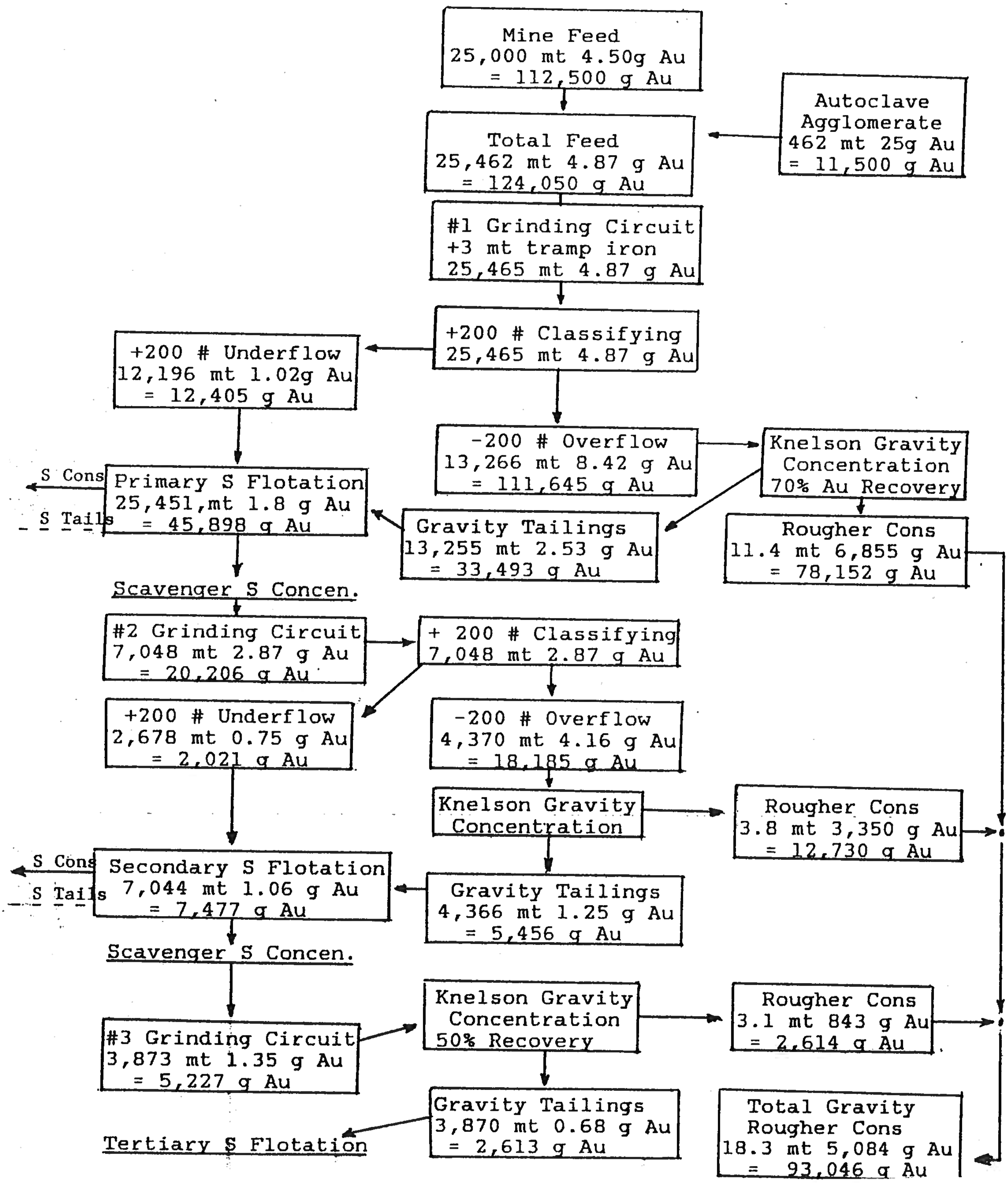
The dore charge then receives an addition of iron or is blown with air , with fused soda ash being used as a flux. The soda slag which forms contains most of the selenium and tellurium remaining in the charge. Prior to casting, the dore metal is cleaned up with Portland cement. The slag from this cleaning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

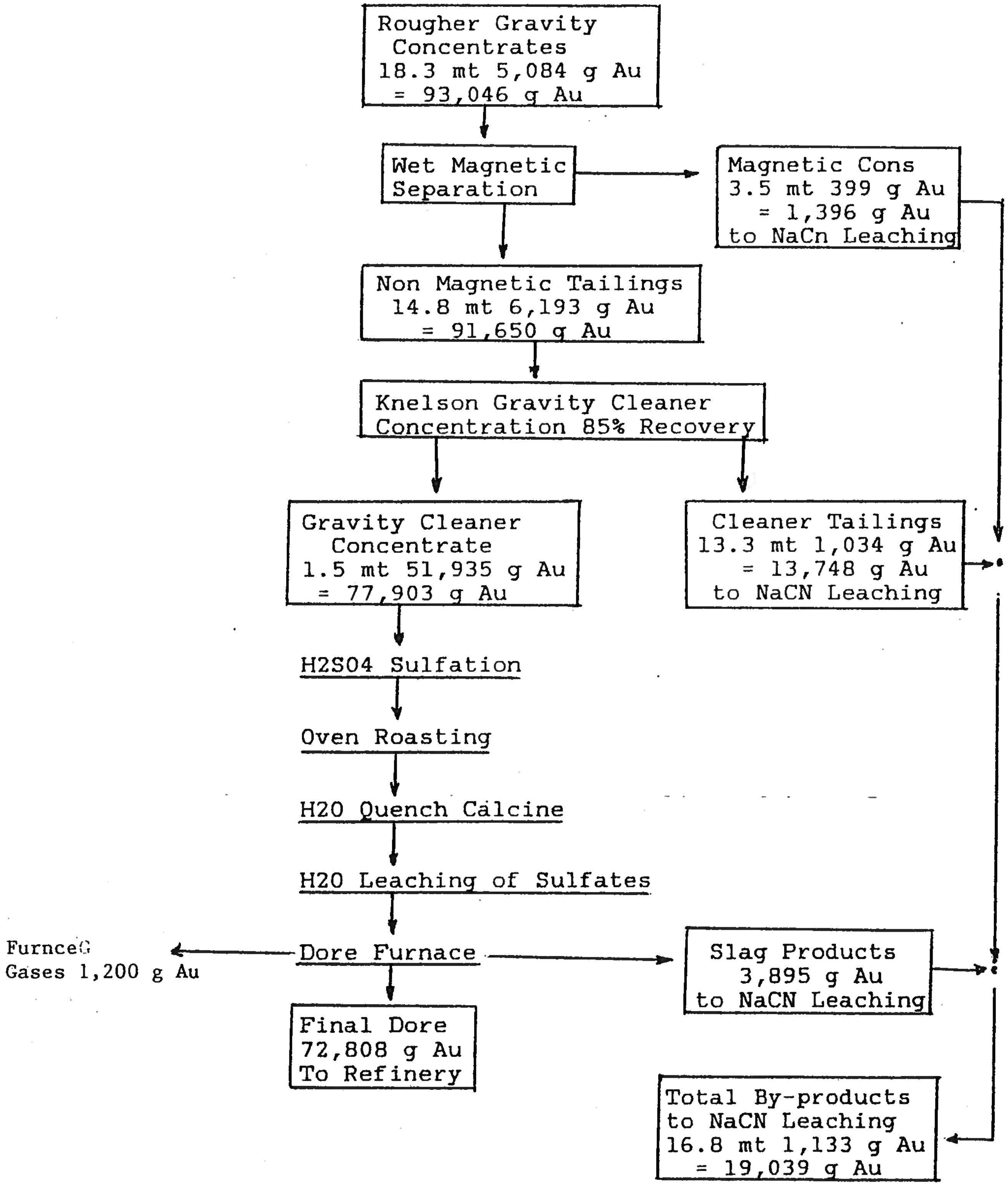
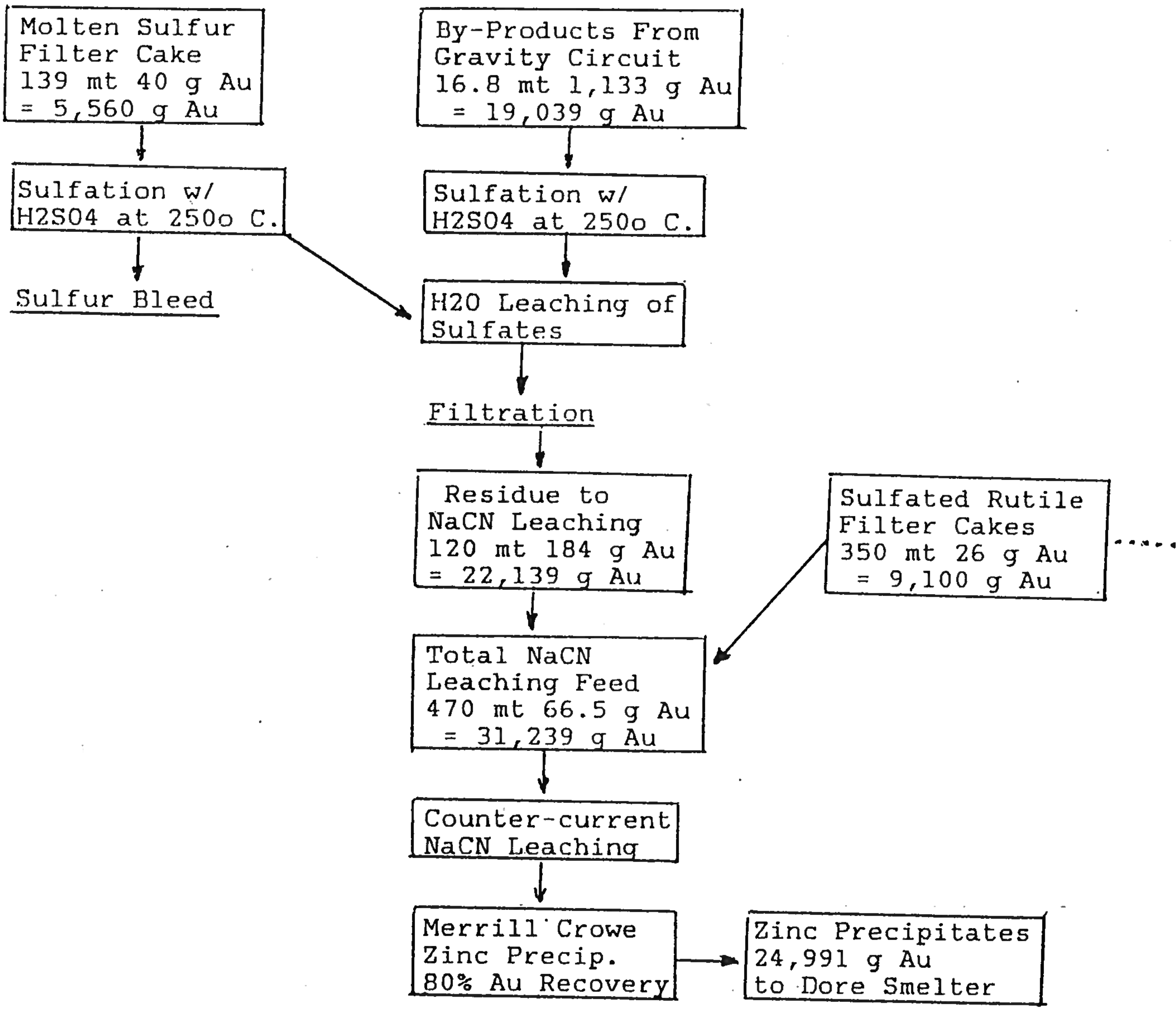


TABLE 32
NaCN LEACHING CIRCUIT



Gold Recovered From Gravity Circuit: 72,808 g Au

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Total Gold Recovered: 97,799 g Au