SAN JUAN (F) GRAHAM

PRODUCERS MINERALS CORPORATION

SAFFORD COPPER PROJECT

June 1975

D1

The [illegible] assaying done by atomic absorbsion. Presumably - when Mining of Peacock (PMC) began there was only a small ore body delineated in scanty drill hole information, used temporarily until the property could be drilled and more accurate data obtained. As drilling progressed the less dependable was phased out, and ultimate Pit data - into July 71 - was based almost entirely upon drill hole data from PMC and to a lesser extent by Scruggs. Assay data used was based upon total copper assays - or recoverable Cu. At depth presumably you have an increase in Chalcocite and chalcopyrite which could lead to an underground potential. You are surrounded by major companies spending millions obviously in the better than guesstimate approach, and there is little doubt of a major reserve in the area which will become a conglomerate.

This merely proves the statements above but also suggests the value of the PMC geographical locality. Adversely, it indicates that you will be under extreme pressure to sell - move out - give - up etc. and the extent to which you bring owners to your side - plus outside capital help - (if you need it for your own expansion) will determine your maximum return, and staying in business.

As far as I am personally concerned, Guy Anderson is a crook of the "first water" - may be deceased now but whatever legacy he left is not in your best interest. As witness your legal problems (concerning Black acid which was just plain harassment); your biggest achievement would be to bring the owners of the lease to your side - by whatever means - before spending much capital, with legal problems replete with answer and settled. Then - again based upon overall position "go to town" so to speak.

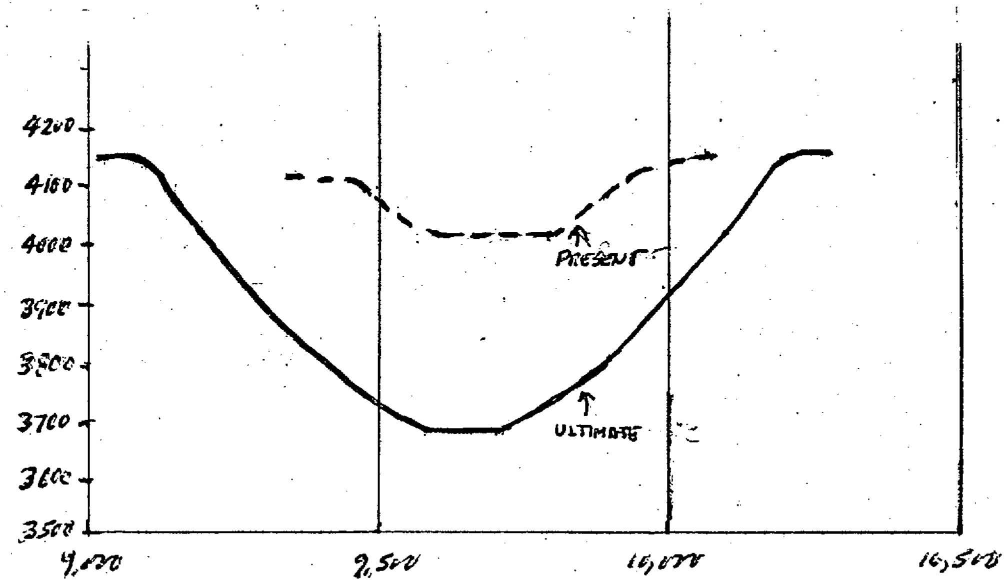

Fig. 3
San Juan Mine
N/S Cross section @ 11,000 Ref Line

PRODUCERS MINERALS CORPORATION

SAFFORD COPPER PROJECT

This report outlines the background information regarding Producers Minerals Corporation's copper property in Safford, Arizona. This property was acquired in 1969 and it has undergone considerable development, and exploration work to establish proved reserves. Following successful court litigation regarding the lease, Producers Minerals Corporation is currently developing plans to carry out further exploration work and to develop the currently proved reserves at an economic mining rate.

History of Property

The PMC property which consists of 10 patented mining claims and 84 unpatented claims is approximately 7 miles north of Safford, Arizona. The property is located just east of the Phelps Dodge property where Phelps Dodge has recently announced a discovery of approximately 400,000,000 tons of 0.7% copper in a deep underground ore body. It is just west of the property where Kennecott some years ago announced discovery of a large underground low grade deposit for which it has not yet completed plans for development. Maps of the property location and claims are attached hereto.

The PMC property is known as the San Juan Mine and it includes property which was mined around the turn of the century for isolated small veins of chalcocite copper. The property was inactive from the early part of this century until 1967. However, during the 1950's exploration was carried out on the property by El Paso Natural Gas and by Kennecott. It was acquired in about 1956 by Mr. Guy Anderson and Mr. Alf Claridge of Safford. In 1966 it was leased by Claridge and Anderson to the Phelan Sulfur Company who did only a very limited amount of development work without mining any significant ore. This lease was then assigned to Mr. Edward Scruggs who, with the Scruggs Mining Company, did a small amount of additional development work in 1968 and very early 1969.

✓ In 1969 Producers Minerals Corporation obtained an assignment of the lease from Scruggs and carried out a significant additional exploration and development program on the property. During the period from 1969 to

Evidently, a great deal of your production beyond 1971 has been leaching the 1.4M tons already mined, ie blasted, and law suits involved have precluded the expansion plans envisaged. They are perhaps settled by now.

Whether Essex is on your side or against including KCC P.D. interests - one way or another the parent was misled as witness discharge of their Senior Geologist and ultimate court decision in your favor. Eimon by professional standards - among his own kind is somewhat spurious and unreliable.. Whatever the outcome - again "none of my business", until this settlement is firmly established - you will be plagued with do-gooders both environmental and plain greed for personal aggrandizement.

Computer associates may be a fine firm and reserves may be accurate. However, without detailed personal attention - sections, level maps, slope definition etc. using polygons - (see attached general outline for your people including ore control) you would be ill-advised to spend capital monies on this basis.

Close scrutiny of all aspects is a "must" before any reasonable, cost-saving conclusion can be drawn, and this includes alternate mining and recovery methods.

If Chrysocolla is prevailing, it is subject to less recovery than - Malachite or other comparable oxides and carbonates. The assays, ore control and distribution are just as important to insure water and solution injection in areas of maximum recovery.

Suggest you concentrate on surface material because in depth drilling etc. is expensive. Get something going to prove that further exploration is justified and get a partner such as Selection Trust - (a good one), St Joe Lead, IMC, (but not KCC or P.D unless - they pay for your position.

1971 Producers Minerals mined approximately 1,400,000 tons of ore and established a leaching operation with production of copper of up to 10 to 12 tons per day. In the summer of 1971, due to the drop in copper prices, the mining of new ore was stopped. Leaching of existing ore heaps of approximately a million tons has continued until the present date. Plans were developed to start mining again in 1972 when prices began firming and more favorable contracts for sale of copper had been achieved. However, in April 1972, the owners brought a lawsuit against Producers Minerals Corporation trying to terminate the lease. The owners had, just prior to April 1972, entered into agreements with Essex International under which Essex International became a part owner of the property and would have operating rights if Producers Minerals was terminated through legal action. This lawsuit has prevented further development of the property. After a lengthy pretrial period, the case came to trial in May and June of 1974. Producers Minerals was successful in defending the lawsuit and has retained its leasehold interest. PMC is pursuing a counterclaim against Essex for interference in its contractual relationship with the owners. This suit is expected to come to trial in late 1975.

Reserves and Ore Type

Through Producers Minerals Corporation's exploration program, remaining proved ore reserves of approximately 15,500,000 tons of average grade of .52% copper have been proven out. These ore reserves are based on a cutoff of .35% copper and a stripping ratio of 1.05/1. The reserve estimates are based on studies by Computer Associates of Tucson, Arizona and are based on over 90 drill holes of approximately 500 feet depth. The cutoff copper level and stripping ratio were established in 1971 and it is believed that with today's higher copper prices, a lower cutoff ratio and a somewhat higher stripping ratio would be economic and would increase the proved reserves if recalculated to about 20,000,000 tons of slightly lower average copper content.

The principal mineralization on the property in the surface reserves is chrysocolla and certain other oxides of a similar type. Host rock is partially quartz monzonite and partially andesite. There are small veins of chalcocite through the ore body and it is believed that in the surface material, 5% to 10% of the total copper is sulfide copper in the leachable chalcocite form. Small localized amounts of cuprite also exist.

No deep drilling (depths of greater than 1,000 feet) has been carried out. In view of the success of PMC's neighbors, Phelps Dodge on

I agree deep drilling is certainly warranted as long as full cognizance of expense involved, and delay in actual results is emphasized. It would however strengthen your negotiating position and perhaps lead to better terms on a partnership basis.

At the time of my visit... → Area I = 15,000 Tons @ 0.6 % = 93 Tons

This 112,000 Tons should be fairly representative, It would appear with grade selection and screening some thought should be given to a small concentrating unit.

2 =	18,000 Tons	@	0.51 =	91.8
3 =	20,000 Tons	@	0.40 =	80.0
4 =	18,000 Tons	@	0.47 =	84.6
5 =	14,000 Tons	@	0.50 =	70.0
6 =	16,000 Tons	@	0.46 =	73.6
7 =	11,000 Tons	@	0.46 =	50.6
Total	= 112,000	avg.	0.485 =	543.6 Tons

At the time of my visit in checking reserves etc. it was shown:
15.6 M Tons in Main ore body - 0.52 % Tot Cu - S.R. (1.05-1) Using 0.35 cut-off
2.2 M Tons in North extension - 0.40 % Tot Cu - S.R. (2.39-1) Using 0.35 cut-off.

These reserves in themselves - based on future of Co. should indicate detailed capital studies for milling and max recovery opposed to leaching or combination of both.

Leaching - an old established process - Broken ore in place begun in the 1920s at Bingham Canyon. Later experimental work well known established (a) that oxygen, bacteria temp. etc had major effect (b) Small cost/lb of production (c) process is relatively simple but success depends upon → (1) Non-acid consuming character such as siliceous limestone as opposed to acid consuming unsiliceous Ls. Stone. (2) Impervious layer under dumps. (3) permeability of mineral zone and ability to form sulphuric & Ferric Sulphate (pyrite solvent) - The ferric sulphate is an active solvent for copper in chalcocite.

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(5) The availability of adequate H_2O - relatively low cost and re-circulation without appreciable loss - (rain bird too much evap. loss) 6. Easily soluble oxides Malachite, Cuprite azurite etc and distribution such that minerals are along fractures allowing direct contact soon by leaching solutions. As far as I can ascertain the spent acid is no less effective than the real stuff. I still would like to see some ponds of measurable size using crushed to $-\frac{3}{8}$". The cost would be relatively low but necessary for overall structure of extraction of I 15M tons.

one side and Kennecott on the other in finding large deep sulfide ore bodies, there appears to be a good possibility that a deep sulfide ore body similar in nature to that discovered by Phelps Dodge may be present underneath the surface oxides on Producers Minerals property. While there are no data to confirm this, it is believed that the prospects are sufficiently promissing that an exploration program for such deep sulfides is well warranted.

The surface oxide reserves of 15,500,000 tons are based on a relatively intensive drilling program over a relatively small amount of the total property of Producers Minerals Corporation, and diamond drilling in most areas stopped in ore both laterally and vertically. It is believed that additional exploration may significantly enlarge the proved reserves of surface oxide ore.

Leachability of the Oxide Ores

Producers Minerals started leaching of the oxide ores in 1970 and has gained three to four years experience on the leaching of these ores. Initially, problems were encountered with the leachability of the ore due to crush size, and only low recoveries were obtained. Extensive column leach tests were carried out which showed that recoveries in excess of 65% should be achievable over a period of 60 to 90 days with proper ore size. Crushing size was changed in the plant in the latter part of 1970 and much improved recoveries were achieved in 1971 before it was necessary to discontinue mining of new ore due to low copper prices.

To prove the extent of recovery achieved in the finer ore crush size in the heaps, a series of Becker drill holes have been made in the main heap. These heaps showed that total copper extraction has been 70 to 80% depending on the analysis and location of the hole, with an average extraction of about 75%. Allowing for normal process losses and the possibility that there may be areas which did not receive quite as high recoveries as these holes, it is believed that a demonstrated recovery of 70% has been achieved. This is recovery based on total copper and not on so called "acid soluble" copper measurement. With proper control of leach solution pH and ferric ion content, essentially all the copper in the surface ore body appears to be recoverable by leaching.

Leaching on this property has been carried out using spent alkylation acid available from Producers Minerals Corporation's affiliate in El Paso, Texas. This spent alkylation acid is treated to remove hydrocarbons prior to use in leaching. The spent acid has proved to be a very effective leaching

This needs to be very carefully handled - My impression was that you had not given sufficient thought or comparison to the overall picture, and conclusions may have been drawn because of the depressed copper price - with all the vagaries involved. If you are looking at long term insitu leaching fine - but with deep potential - activity in the area - more emphasis should be paid to early recovery - showing a good balance sheet to insure proper return to Producers Minerals - after negotiating a partnership involving more $M. - If producers can raise the funds so much the better but you must - show evidence to banks etc of production!

You are presently using H_2O from - Phelps Dodge - on a poor boy basis - unfounded with the mineral assets of some 18-20M tons of Mineable - recoverable material. So to what extent have you established H_2O capital cost delivery to site in the volume needed and to what extent have you laid plans or already implemented saving H_2O on the one hand, increasing g.p.m on the other? Have you drilled any wells? - what results.?

Lease Terms: The lease terms where owners retain 25% net profit in the operation is not unreasonable - Nor is net profit defined for Federal Income tax purposes. However the Minimum Monthly payment of 1500.00/Mo with No profit is questionable especially with obligations continuing. You have mentioned a modified lease which shows your concern - one strength is to give a little - take a little but emphasize to owners that you will not be a party to skull duggery!

Assignment of the lease from Scruggs provides royalty = 1 - 1½ % of Net Sales Price - of cement copper produced. Is the net sales price after Freight handling etc. as well as actual price received for Cu?

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agent and sufficient ferric ion has been maintained in the leach solutions to achieve the desired leaching of the small amount of sulfides and cuprite present.

Further test work has been carried out on leaching uncrushed ore which indicates that while recovery rate will be significantly lower initially than on crushed ore, satisfactory ultimate recoveries over a period of one to three years can be achieved. Significant cost savings due to elimination of the need for crushing and additional hauling can be achieved by leaching uncrushed ore. This route also permits leaching in place of about half of the total ore body, thereby eliminating the need for hauling this portion of the ore body. Full scale field demonstration of the techniques for leaching the uncrushed ore are now being carried out.

Details of the leaching program and associated data are included in Appendix A.

Water Supply

Producers Minerals has rights to water supply from wells next to the Gila River. Initially, water was obtained from this source and pumped by pipe line to the mine. Currently, Producers Minerals has an expanded source of water via pipeline from Phelps Dodge's property adjacent to PMC. Other sources of water could be developed, either from the Gila River or from deep well drilling on the property.

Lease Terms

The basic lease from Anderson to Scruggs is a 99 year lease under which the owners retain a 25% net profits interest in the operation. The net profits as defined in this agreement are net profits as calculated for federal income tax purposes. The lease also provides for a minimum monthly payment of $1, 500 if no profits are achieved. It is believed that for a number of legal and financial reasons that a modified lease with the owners can be negotiated under which the owners will substitute a royalty, based on a percentage of sales, in lieu of net profits as well as to modify certain other terms of the lease.

The assignment of the lease from Scruggs provides that Scruggs will receive a royalty equal to 1-1/2% of the net sales price of the cement copper produced.

During my tenure in England Selection Trust did work for Southwire along Engineering Lines and presumably Selection Trust would be a good partner in advancing PMC interests if capital is needed. They are very anxious to obtain a foothold in the U.S., and this has been evidenced to me on a number of project studies.

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1. I cannot concur that in place leaching - with no emphasis rock in place size is the way to go. This is the type of long term leaching used on Waste dumps - from which profitable ore has been extracted. Rain bird sprinklers - while they cover the area are subject to evaporation - to excess in the local climate where H_2O is at a premium. By studying drainage areas and using historical rainfall dams should be built to catch run-off and additional provision made to bring H_2O from Gila River rights.

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2. Am curious what if any tests have been made on a pilot plant basis to compare in place with proper sampling and portable crushing to say ± 3/8". This gets back to the fundamental - exposure of material to leaching solutions.

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3. Phase III is fine except that for money spent in capital using 9000 Tons/day of "run of Mine" why not send crushed material (sized by pilot tests for best recovery) to quickly reduce capital interest etc. I would expect that serious consideration of all factors would ultimately swing the pendulum toward some degree of crushing. (See Summary)

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4. As far as phase 4 is concerned based upon the reserves on Page 3 would certainly check the feasibility of an actual milling operation on a scale commensurate with corporate structure. Am speaking of present reserves but if finance permits I would drill holes (P-36) - (P-55) (P-56) - (P-59) - (P-62) (P-39) - (P-63) - (P-64) - all drilled in the order of best geological inference for most rapid information

The ore (crushed ore) should be placed in confined pad areas (your old previous ponds are an ideal spot with impervious sub-base) and close to recovery plant.

Copper Sales Contract

PMC has a contract with Southwire Corp. under which Southwire has the offtake rights for cement copper produced. This contract lasts until December 31, 1978. Under the contract production is sold to Southwire under a price equal to producers price less smelting and finance charges.

Mining Plan for Surface Oxides

Analysis of the various alternatives for mining and leaching of the surface oxide ores has led to the following program.

1. Phase I This phase, which will be carried out during the latter half of calendar year 1975, will be a field demonstration of leaching of uncrushed ores to demonstrate recovery rates achievable using this technique. Additional ore will be blasted and leached in place through the end of 1975. Projected copper production rate will be raised from the current level of two tons/day to five tons/day. Leaching technique will utilize circulation of barren solution with Rainbird plastic sprinklers and iron precipitation of copper in the present launder.

2. Phase II Assuming Phase I shows satisfactory recoveries from leaching of uncrushed ore as anticipated, starting in February 1976, ore will be hauled from the pit at the rate of approximately 5,000 tons/day. Production will be held at approximately 5 tons/day through July of 1976.

3. Phase III Starting in August 1976, mining rate will increase to approximately 9,000 tons/day. Copper production is scheduled to increase to 8 tons/day by November 1976, and to 9 tons/day by April 1977. An electrowinning facility for production of electrode copper in lieu of cement copper is planned to come onstream in November 1976.

4. Phase IV Starting in July 1977, mining rate will be increased to approximately 16,000 tons/day of ore plus waste. Production will be raised to 13tons/day of copper. This phase will be continued through July of 1980, by which time approximately half the total reserves of ore and associated waste will have been removed from the pit. For the subsequent five years ore will be blasted and leached in place in the pit. Leaching will continue for approximately two years after all ore has been blasted.

All ore hauled from the pit will be placed on dumps and leaching will continue on these dumps until the ore has been leached for an average of approximately two to two-and-a-half years.

The design copper recovery is shown on the attached figure and compared thereon to the laboratory data. The design leaching curve is based

Have no comment other than much work has been done - but I would question again, the recovery % of uncrushed material as opposed crushed. Simply said, with capital expansion involved, using proper ore control procedures to crush only profitable ore it does not cost that much more if well managed and Recovery, early should justify.

Computerized pit studies are fine - but you need Bench plans, sections - pit slopes established etc. and in detail. Your Maps and drawings are beautiful - well prepared and far superior to most. The detail necessary from here to show Bench plans etc and sections would be money well spent. See App. on Planning & Control.

The extent to which you have carried out the detailed studies since my visit may have answered many questions raised here. You mention detailed design and equipment selection. My only query would be "to what extent have you considered all alternates comparing capital, operating costs and recovery for the long range

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There is no question that the price of Cu will escalate to ± 0.90/lb by Mid-1976 to my mind - because the industry cannot survive without it and survival is a must in world economy. Money spent now will be well advised - but only after all facets are considered in terms of property potential.

I have no comment on Appendix Tables and information obviously well prepared and studied. Electrowinning saving iron etc are being currently proven beyond doubt.

My inference is that too much emphasis has been paid upon the "poor boy" approach - (for reasons which are none of my concern.)

On the other hand the geographical position of PMC indicates a certain command of the area - conditions indicating a strong bargaining position. Money spent wisely now, to improve operations and net profit has a two-fold reward:

(1.) Indicates PMC is a force to be reckoned with and has substantial assets in its own right.

(2) This force with reserves + strategic location - provides bargaining power with the majors!

on the more refractory monzonite host rock. The design curve assumes that the ratio of time to allow a given recovery in the field to that required in the laboratory is the same for uncrushed ore as previously determined for crushed material. The design leaching curve is currently subject to verification but appears reasonable based on data obtained to date and in other leaching operations. The extent to which concentrated acid will need to be applied to the monzonite ore to accelerate the decrepitation process is also subject to further field evaluation.

Detailed pit plans delineating which sections of the overall pit, as defined by prior Computer Associates' studies, will be mined in which sequence will be developed at the beginning of Phase II for the operation by further computer studies. Figures 1, 2 and 3 attached hereto show the rough outlines of the present and projected ultimate pit boundaries.

The attachments hereto show the projected economic results from this operation, the amount of ore blasted and hauled in each period, and the basic underlying economic data behind the financial projections. Included in these is an allowance for financing, via leasing a $500,000 expenditure for additional water supplies. All assumptions as to equipment are for economic calculations only. Detailed design and equipment selections will be carried out in the last half of 1975.

Overall, based on projected costs and recoveries, the proved reserves are shown to generate a cash profit before interest, taxes and royalties of $22,000,000 over a period of ten years, with profit growing from $286,000 in the fiscal year ending July 31, 1976, to $3,273,000 in the fiscal year ending July 31, 1981.

As shown in the attached Table 3, the total profit is increased some $5.6 million by the addition of electrowinning due the saving in iron cost and the additional revenue received by the manufacture of cathode copper in lieu of cement copper. The economics shown assume that all new equipment for both mining and for electrowinning have been leased rather than purchased.

Fig. 1
San Juan Mine
1974 Pit Contours

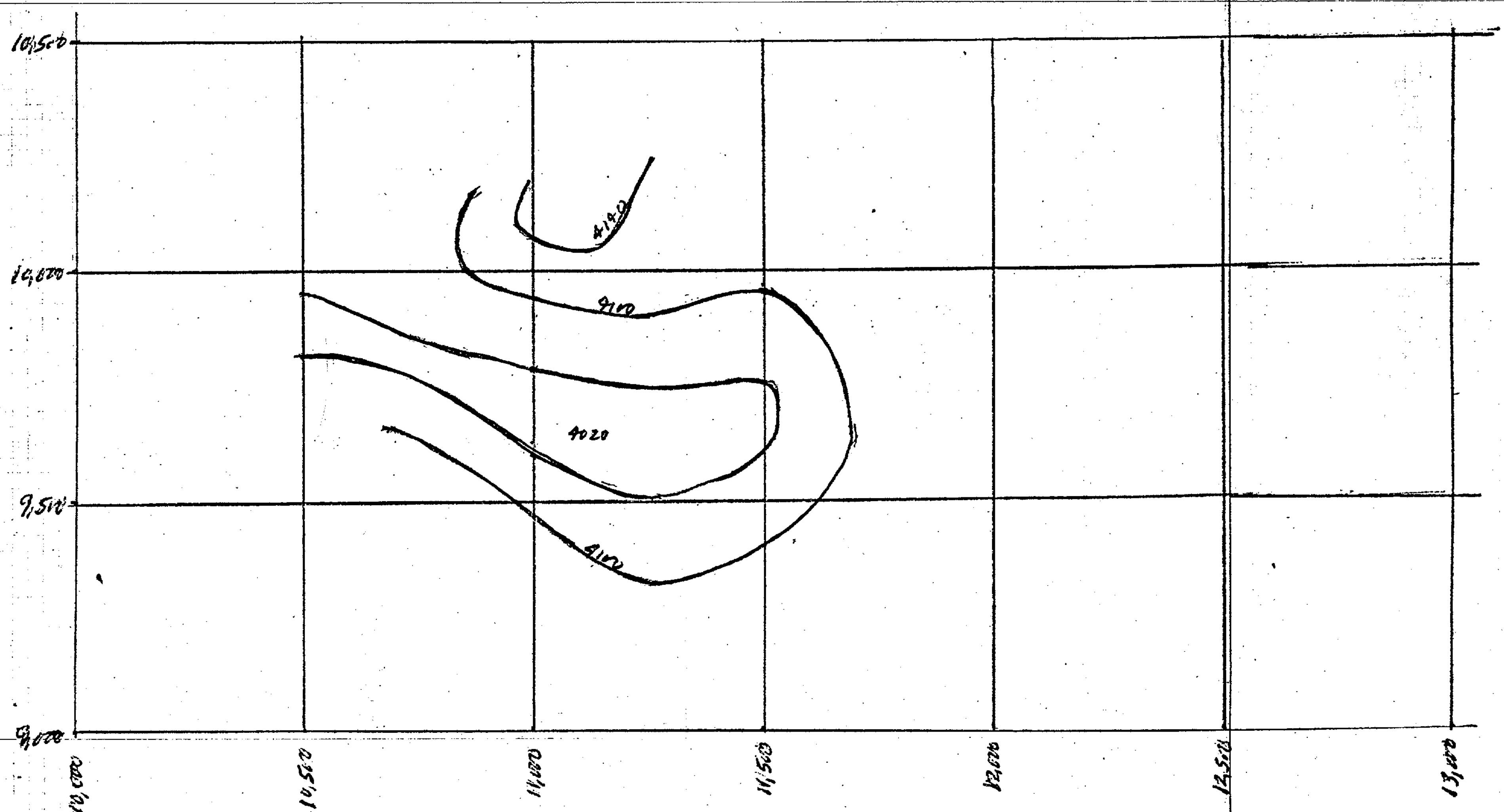


Fig. 2
San Juan Mine
Ultimate Pit Contours

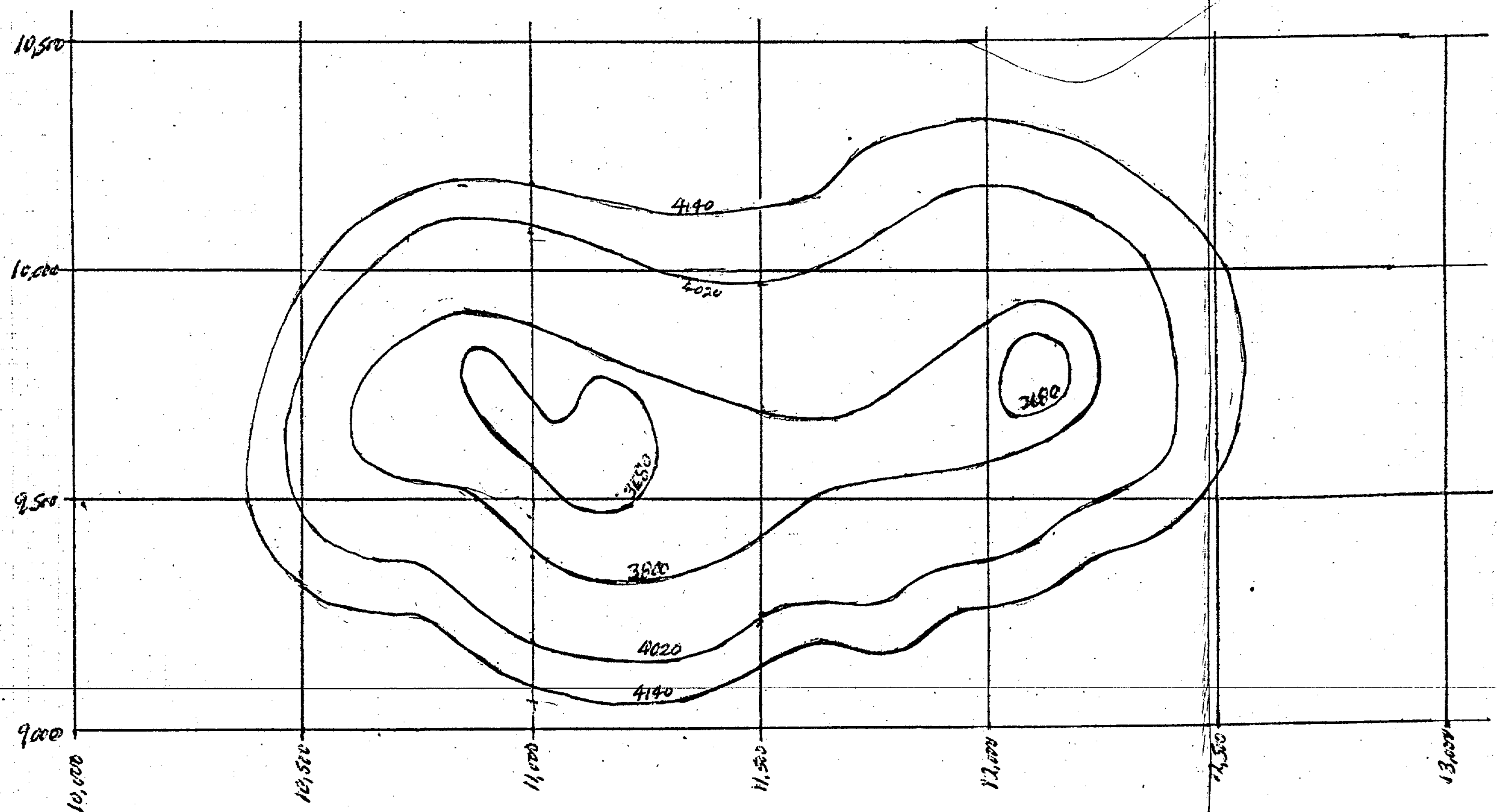


Table 1

Copper Production Projection
Based on "Prelim Curve Insitu Leach"

	Ore Tons	% Cu	Cu Tons	Period (3 months)									
					Fiscal 1976				Fiscal 1977				
				1	2	3	4	5	6	7	8	9	10
Cum % Recov				28	33	37	40	42	44	45.5	47	48	49
Inc. % Recov				28	5	4	3	2	2	1.5	1.5	1	1
From Leaching Old Heaps				0.8	0.80	0.7	0.7	0.6	0.6	0.5	0.4	0.4	0.3
Present Pit	80,000	0.7	540	1.68	0.30	0.24	0.18	0.12	0.12	0.09	0.09	0.06	0.06
Blast 1	62,500	1.0	625		1.94	0.35	0.25	0.21	0.14	0.14	0.10	0,10	0,07
Blast 2	137,500	0.8	1,100			3.42	0.61	0.49	0.37	0.24	0.24	0.18	0.18
Blast 3	137,500	0.8	1,100				3.42	0.61	0.49	0.37	0.24	0.24	0.18
Blast 4	167,500	0.6	1,002					3.11	0.56	0.45	0.34	0.22	0.22
Blast 5	167,500	0.6	1,002						3.11	0.56	0.45	0.34	0.22
Blast 6	305,000	0.6	1,831							5.69	1.02	0.81	0.61
Blast 7	305,000	0.6	1,831								5.69	1.02	.81
Blast 8	305,000	0.6	1,831									5.69	1.02
Blast 9	305,000	0.6	1,831										5.69
Total Produced T/D				2.48	3.04	4.71	5.19	5.14	5.39	8.04	8.57	9.06	9.36

Table 1A

Copper Leach Plan Summary

	Tons - Ore	Tons - Waste	Tons - Copper in Ore	% Copper
Total Reserve 3/74	15,500,000	16,275,000	80,600	0.52
Blasted Fiscal 1975	142,500		1,165	0.81
Blasted Fiscal 1976	610,000		4,204	0.69
Blasted Fiscal 1977	1,220,000	1,220,000	7,324	0.60
Balance Left 8/1/78	13,527,500	15,055,000	67,907	0.502
Hauled to 8/1/78	1,892,000	1,220,000		

Total Copper Recovered over time @ 50% 40,300

Total Copper Recovered to 8/1/78 from new ore = 223 + 1,642 +2,920 - 495 = 4,290

36,010

Ore Rate for 13T/D Production = $\frac{13.0}{9.4} \times 1,220 \times \frac{.60}{.502}$ = 2,016,000 T/Yr

Copper to Leach = 2,016 x .0052 = 10,120 T/Yr

Copper Recovered = 4,745 T/Yr

Years Blasting Left 13,527,500/2,016,000 = 6.71

Copper Recovered 6.71 x 4,745 = 31,839

Copper Recovered after all ore blasted = 36,010 - 31,839 = 4,171

Total Time = 8.71 years

Ore blasted = (15,055/13,527) x 2,016,000 = 2,243,000

Ore and waste hauled = 4,255,000 T/Yr (16,365 T/D on 5 day week)

Years hauling for 50% Removal = 15,887,500 - 3,112,000/4,255,000 = 3.0 years

Table 2

Drilling and Blasting Costs

Blasting Cost

Explosive used 0.5 Lb/Ton		
ANFO Cost	9¢/Lb	
Explosive Cost ¢/Ton		4.5
Labor Cost		0.5
		5.0

Drilling Cost

Drill hole spacing	12 x 15	
Hole size	7"	
1970 Cost Bur. Mines *		3.6¢/Ton
Factor to 1975 Cost		1.40
1975 Cost		5.0

Total Cost ¢/Ton = 10.0 Drilling and Blasting

* 1.8¢/Ton for 40,000 T/D mine 9" hole 12 x 24 spacing

Table 3

Hauling Costs PMC Mine

Cost Basis (1) May 1974 Empire Machinery Study for 10,250 T/D
Avg. Haul 1,850 FT 4% Avg. Grade
2 - 988 Loaders, 6 - 769B Trucks hauling 12,936 tons/shift

(2) Leasing 6 year life Lease Cost % original cost is interest 8%, ins & taxes 2%, depreciation 16.6 = 26.6% (equiv to 16% interest on avg investment)

(3) Costs on 5/75 by new quote total unit, tires estimated

Hauling Rate: 1977 - 1,952,000 T/Yr, 7,520 T/D (5 day week);
1978-80 - 3,664,000 T/Yr, 14,100 T/D (5 day week).

	988		769B	
	5/74	5/75	5/74	5/75
Purchase Price ex tires	101,107	122,156	109,304	136,248
Tires	9,385	10,605	6,830	7,718
Purchase Price incl tires	110,493	132,761	112,135	143,966
Operating Costs $/Hr				
Fuel 21¢/34¢	2.73	4.42	1.68	2.72
Lubricants	0.45	0.50	.39	.44
Tires	3.75	4.23	2.73	3.08
Repairs	13.02	14.71	7.37	8.32
Total	19.95	23.86	12.17	14.55
Lease Costs $/Year *		32,574		38,391
$/Hour		16.28		19.19
Labor Cost $/Hour	5.50			
Shift Cost				
Operating 7 Hr.		167.02		101.85
Lease 8 Hr.		130.30		153.52
Labor 8 Hr.		44.00		44.00
Total Shift		341.32		299.37
Total Cost 2 - 988		682.64		
6 - 769		1,796.22		
		2,478.86		

Rock Hauled 7,512 + 5,424 = 12,936 tons/shift

Cost/ton = 19.16¢

* Basis 6 year life (12,000 hours), 8% interest on original invest, 2% ins. & taxes

Table 4

Manufacturing Costs

$/Month

Fiscal Year	-75---	-----	----- 76	-----	-----	---- 77	-----	---78---
Quarter	4	1	2	3	4	1	2, 3, 4	1, 2, 3, 4
Production T/D	2.5	3	4.7	5.2	5.2	5.4	8.6	13
Ore & Waste Hauled T/D	0	0	0	5,200	5,200	9,400	9,400	16,300
Base Manufacturing								
Personnel (1)	6	6	9	9	9	9	10 (3)	10
Payroll & benefits (1)	7,000	7,000	10,000				11,000	11,000
Outside P. S.	1,200	1,200	1,200				2,200	2,200
Fuel & butane	3,600	4,000	4,500				5,000	6,000
Repairs, Supplies, Misc.	4,900	6,500	7,500				7,700	7,700
Admin Costs Safford	500	500	1,000				1,000	1,000
Insurance	200	200	200				500	500
Taxes	1,100	1,100	1,100				1,100	1,100
Lease Rental	1,500	1,500	1,500				1,500	1,500
Total	20,000	22,000	27,000	27,000	27,000	27,000	30,000	31,000
New Equipment Leasing								
Crawler			3,000				3,000	
Loader			2,000				3,000	
Motor Grader			2,000				2,000	
Rotary Drill			3,000				7,000	
Total	0	0	10,000	10,000	10,000	10,000	15,000	15,000
Other								
NY Admin	-	-	1,700	2,300		2,600	2,600	4,600
Water Supply Lease (2)	-	-	0	0		0	8,400	10,400
Total	0	0	1,700	2,300		2,600	11,000	15,000

NOTES

(1) For leaching & admin only. Purchased maintenance labor under outside P. S. Other personnel under blasting & hauling costs.

(2) Based on leasing $500,000 pipeline & supply wells from Gila River.

(3) Plus personnel Base
1 clerk/steno
1 mining engineer
1 lab assistant
1 launder operator

Table 5

Electrowinning Plant Costs

Basis - 13 T/D Copper Production 4,745 T/Yr.
Pregnant Solution @ 3 gpl, solution rate = 785 gpm (92% service factor)
Power cost 1.5¢/kwh. Labor 2 men/shift (8 men vs. 14 in BM).
Labor Rate $5.

	Bureau Mines	PMC	
	1970 Cost	1970 Cost	1975 Cost
Copper T/Yr	7,000	4,745	4,745
Solution Rate gpm	3,000	785	785
Cost Index Chem Plant	125.7	125.7	179.6
Investment M$			
Solvent Extraction	3,203	1,432	2,051
Electrolytic	2,390	1,892	2,710
Offsites	560	333	476
	6,153	3,657	5,237
Operating Costs			
Solvent	157	41.0	58.5
Power	113 (0.7¢)	76.6	164.1 (1.5¢)
Labor	163	93.1	124.1
Maintenance	157	93.3	133.6
Other	93	63.0	90.0
Total Direct	683	367.0	570.3
¢/Lb. Copper	-	3.86	6.01

Leasing Cost 10 year

Interest	7.0%	original cost (14% avg. capital)
Depreciation	10.0	
Taxes, ins.	2.0	
	19.0	= $1,000,000/yr

Table 6

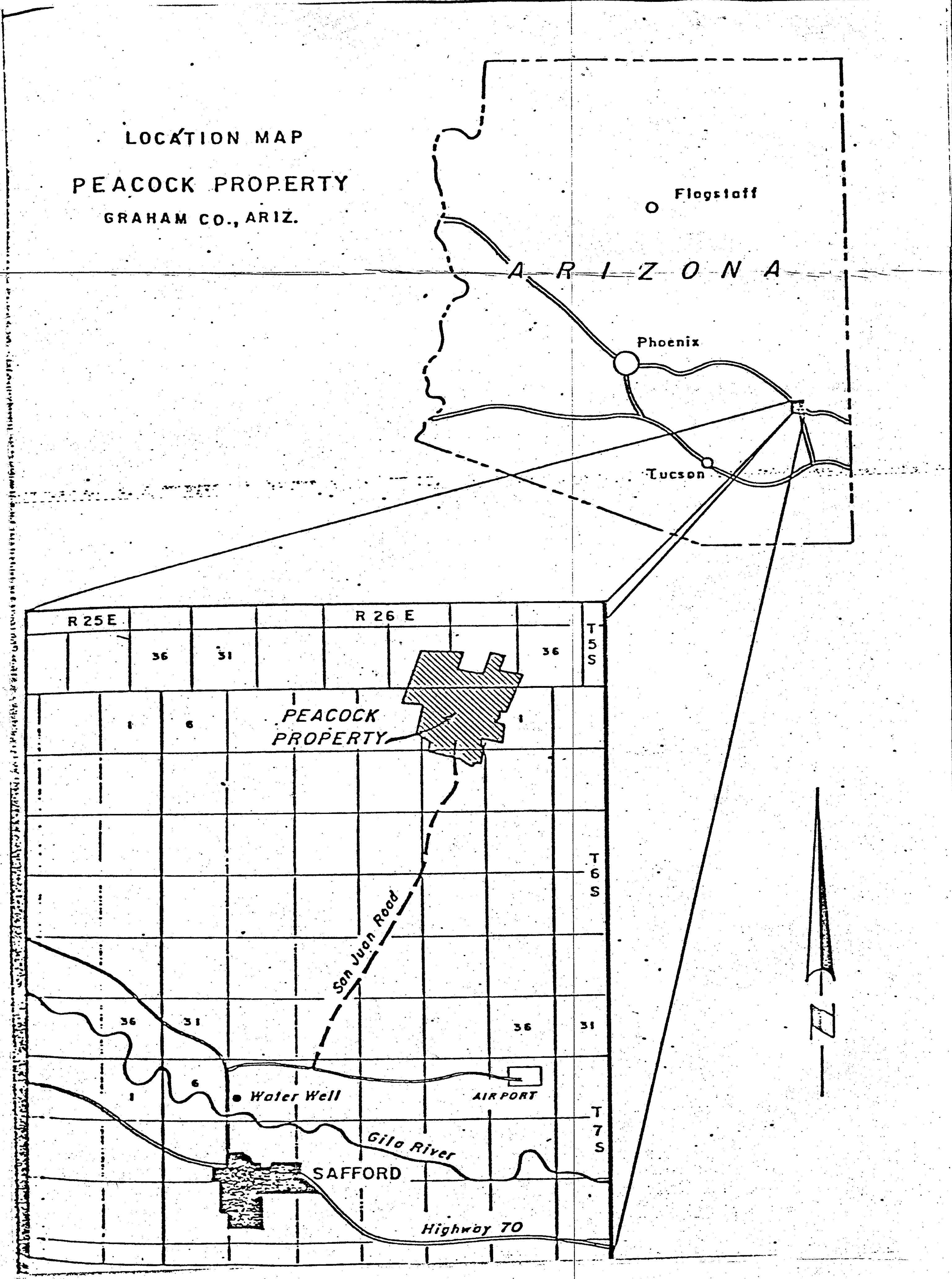
PMC QUARTERLY FINANCIAL PROJECTION 1975-77

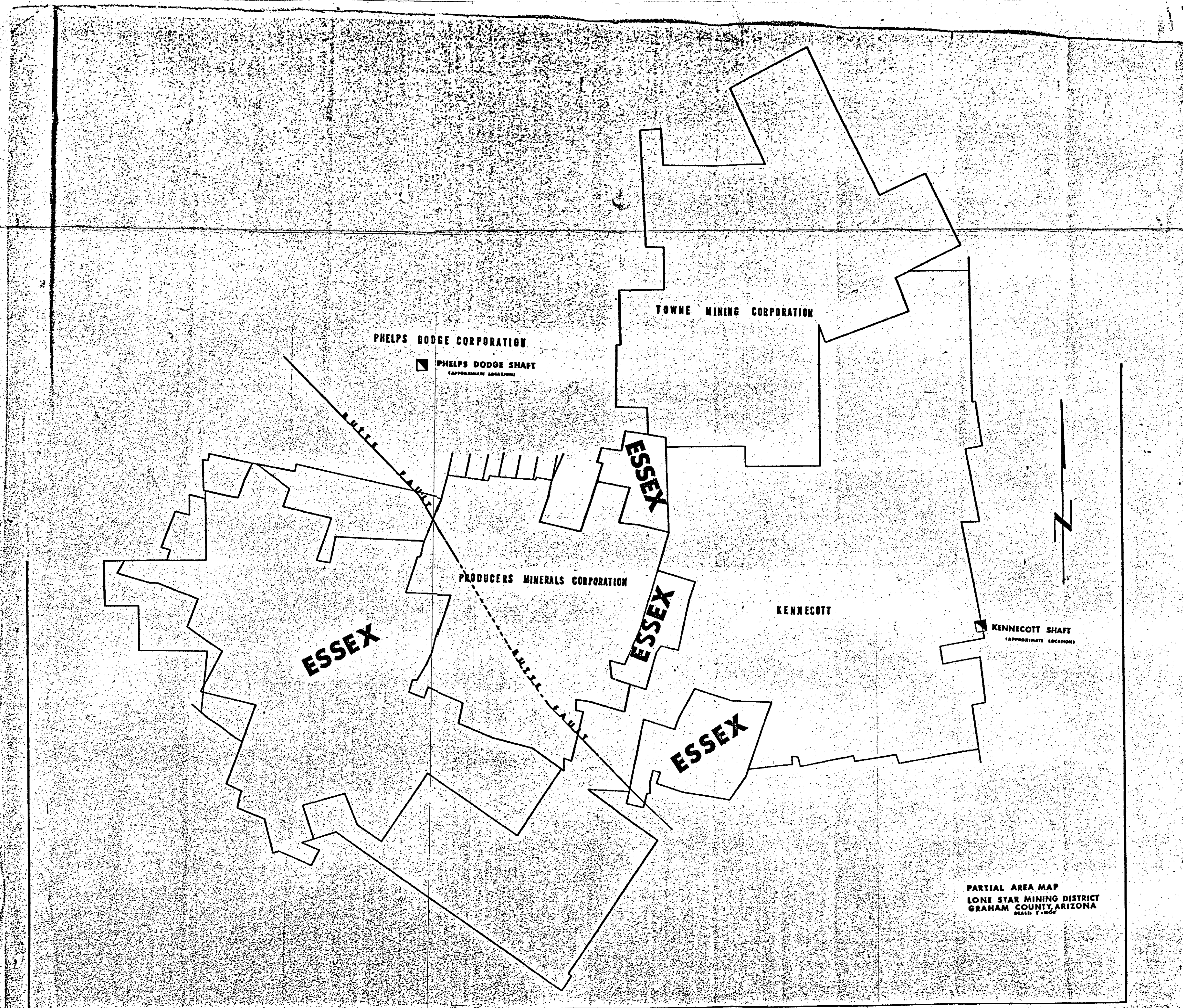
		Fiscal 1976				Fiscal 1977			
Copper Price	63	63	63	63	63	73	73	73	73
Period (Quarter)	4	1	2	3	4	1	2	3	4
Production T/D	2.5	3	4.7	5.2	5.2	5.4	8.0	8.6	9.1
Waste Blasted M Tons						305	305	305	305
Ore Blasted M Tons	63	138	138	167	167	305	305	305	305
Waste & Ore Hauled	0	0	0	336	336	610	610	610	610
Revenue	191	229	359	397	397	508	752	808	855
Acid & Iron	62	75	116	129	129	134	198	213	225
Royalties Scruggs	4	4	7	8	8	10	10	0	0
Margin	125	150	236	260	260	364	544	595	630
Base Manf.	60	67	80	80	80	80	90	90	90
Blasting Costs @ 10¢/Ton	6	14	14	16	16	61	61	61	61
Hauling Costs @ 20¢/Ton	0	0	0	67	67	122	122	122	122
New Equip Leasing			29	29	30	30	44	44	44
Other	0	5	7	8	8	8	32	32	32
Total Costs	66	86	130	200	201	301	349	349	349
Oper Cash Flow before interest, depreciation, royalties	59	64	106	60	59	63	195	246	281
Sensitivity 5¢/# Copper Price	22	26	41	45	45	47	69	75	79

Table 7

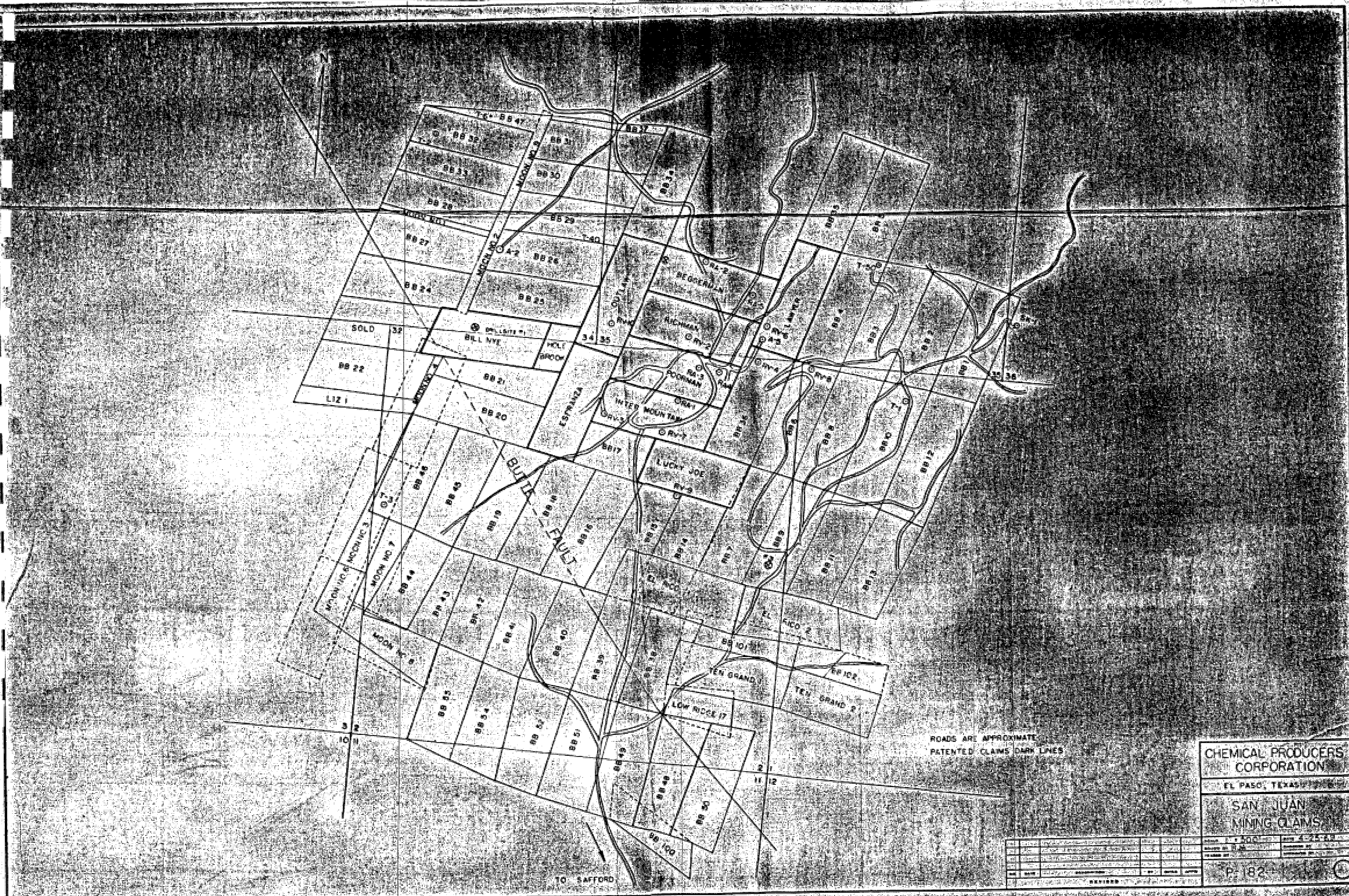
Nartex Copper Mining Operations

Year Ending July 31	1976	1977	1978	1979	1980	1981	1982	1983	1984	1985	1986	Total
Copper Price	63	73	73	73	73	73	73	73	73	73	73	
Production T/D	4.5	8	13	13	13	13	13	13	13	4.3	3.4	
T/Yr	1,642	2,920	4,745	4,745	4,745	4,745	4,745	4,745	4,745	1,566	1,229	40,572
Ore Blasted M Tons	610	1,220	2,012	2,012	2,012	2,012	2,012	2,012	1,428	0	0	15,358
Waste Blasted Tons	0	1,220	2,243	2,243	2,243	2,243	2,243	2,243	1,542	0	0	16,275
Ore & Waste Hauled Tons	672	2,440	4,255	4,255	4,255	0	0	0	0	0	0	15,877
Cement Copper Case												
Revenue	1,375	3,026	4,916							1,622	1,274	41,709
Acid & Iron	202	799	1,300							429	332	
Scruggs Royalties	24	46	73							24	19	624
Margin	904	2,181	3,543	3,543	3,543	3,543	3,543	3,543	3,543	1,169	918	
Base Manufacturing Costs	307	350	375			375			375	300	300	
Blasting Costs @ 10¢/ton	60	244	425			425			302	0	0	
Hauling Costs @ 20¢/ton	135	487	851			0			0	0	0	
Base Equipment Leasing Costs	88	163	188			188			188	188	188	
Other	28	105	183			183			183	28	0	
Total Costs	618	1,349	2,022	2,022	2,022	1,171	1,171	1,171	1,048	516	488	
Profit before Interest, Taxes, Royalties	286	832	1,521	1,521	1,521	2,372	2,372	2,372	2,495	653	430	16,497
Electrowinning Case												
Added Revenue		(867)	(1,673)							(552)	(433)	13,563
Added Oper Costs		300	580							191	150	
Iron Cost Savings Cost		(418)	(808)							(267)	(210)	
Added Leasing Cost		750	1,000							1,000	1,000	
Added Profit		235	901	901	901	901	901	901	901	(373)	(507)	
Profit before Int., Taxes, Royalties	286	1,067	2,422	2,422	2,422	3,273	3,273	3,273	3,396	280	(77)	22,037


LOCATION MAP
PEACOCK PROPERTY
GRAHAM CO., ARIZ.
Flagstaff
ARIZONA
Phoenix
Tucson
R 25 E
R 26 E
T 5 S
T 6 S
T 7 S
36
31
1
6
PEACOCK PROPERTY
San Juan Road
Water Well
AIR PORT
Gila River
SAFFORD
Highway 70
N



PHELPS DODGE CORPORATION
PHELPS DODGE SHAFT
(APPROXIMATE LOCATION)
TOWNE MINING CORPORATION
ESSEX
PRODUCERS MINERALS CORPORATION
ESSEX
KENNECOTT
KENNECOTT SHAFT
(APPROXIMATE LOCATION)
ESSEX
ESSEX
N
PARTIAL AREA MAP
LONE STAR MINING DISTRICT
GRAHAM COUNTY, ARIZONA


N
BB 47
BB 32
BB 31
BB 33
BB 30
MOON NO 9
BB 28
BB 29
BB 27
MOON NO 2
A-2
BB 26
T-40
BB 24
BB 25
BEGGERMAN
RICHMAN
LAWYER
SOLD
BILL NYE
HOLE BROOK
34 35
BB 22
BB 21
LIZ 1
BB 20
ESPRANZA
POORMAN
INTER MOUNTAIN
RV-7
BB 17
LUCKY JOE
RV-9
BUTTE FAULT
BB 46
BB 45
BB 19
BB 18
BB 16
BB 14
BB 12
BB 11
BB 13
MOON NO 7
MOON NO 3
BB 44
BB 43
BB 42
BB 41
BB 40
BB 39
MOON NO 8
EL RICO 2
BB 101
TEN GRAND
EP 102
TEN GRAND 2
LOW RIDGE 17
BB 55
BB 54
BB 52
BB 51
BB 49
BB 48
BB 50
BB 100
3 2
10 11
2 1
11 12
35 36
TO SAFFORD
ROADS ARE APPROXIMATE
PATENTED CLAIMS DARK LINES
CHEMICAL PRODUCERS CORPORATION
EL PASO, TEXAS
SAN JUAN MINING CLAIMS
P-182

PATENTED LODE MINING CLAIMS	SURVEY NO.
Hole Brook	3299
Esperanza	"
Bill Mye	"
Beggarman	"
Richman	"
Poorman	"
Intermountain	"
Outlaw	"
Lawyer	"
Lucky Joe	"

Located in Section 2, T. 6S, R. 26E and Sections 34 and 35, Township 5 S, R. 26 E, Gila and Salt River Meridian, Graham County, Arizona. Patent recorded in Book 5 of Patents at Pages 63 - 69.

UNPATENTED LODE MINING CLAIMS	DOCKET	PAGE
Blue Bird 1 to 17	34	511-527
Blue Bird 18 to 29	34	559-570
Blue Bird 22 to 33 - (Amended)	54	354-356
Blue Bird 30 to 35	35	82-87
Blue Bird 36	36	151
Blue Bird 37	38	165
Blue Bird 38 to 46	37	457-465
Blue Bird 47 to 48	38	166-167
Blue Bird 49 to 52	37	466-469
Blue Bird 53 to 55	39	331-333
Blue Bird 100 to 104, except 103	47	339-341
Blue Bird 100 - (Amended)	59	442
Ten Grand 1 to 2	37	270-271
Rico, also known as El Rico 1 to 2	36	149-150
Moon 1 to 4	54	350-353
Moon 5 to 6	55	474-475
Moon 7 to 8	58	226-227
Solo	53	530
Liz 1	54	348
Low Ridge 17	59	440

Located in Sections 1, 2, 3, 11, Township 6 S, R. 26E and Sections 33, 34, 35, T. 5S, R. 26E, Gila and Salt River Meridian, Graham County, Arizona

APPENDIX A

PRODUCERS MINERALS CORPORATION

LEACHING OF SAN JUAN ORE IN SAFFORD, ARIZONA

Introduction

In 1969 Producers Minerals Corporation obtained a lease on the San Juan copper property about seven miles north of Safford, Arizona. Prior to Producers Minerals' operation, a very small experimental leaching operation had been carried out by a prior operator. A drilling program initiated by Producers Minerals proved out about three million tons of copper ore reserves by early 1970; and this has later been expanded to about fifteen and a half million tons of .52% copper ore with a .35% copper cutoff and a 1.05/1 stripping ratio. The ore body defined to date is a surface ore body going to a depth of 500 feet or less. The primary copper mineral is chrysocolla, although there are veinlets of chalcocite and occasional occurrences of cuprite. It's believed that of the total copper present, between 5 and 10% is in the sulfide form and probably at the low end of that range. The host rock is quartz monzonite on the north side of the pit and andesite on the south side.

Pilot Plant Data

Pilot plant tests were initiated in 1969 prior to commercial production. An ore sample was selected with the assistance of consultants to determine leachability of the ore. Large scale tests were run in a 4-1/2 ft. diameter column leach testing unit. The data obtained in these column leach tests, designated 1969-1 and 1969-2, are summarized in Table 1. These data indicated that a recovery of approximately 60% should be achievable in 60 days of leaching. Column leach tests were run with equilibrium recycle leach solution after precipitation of the copper by iron. Copper production and recovery was measured by cement assays. Leaching solution was made up using skimmed spent alkylation acid. The use of this type of acid is discussed further below.

Based on the favorable column leach tests, commercial production and leaching was started in February of 1970 but, as discussed below, ran into problems of low recovery. To determine the reasons for the low recoveries, a series of additional column leach tests were run in 4-1/2 ft. concrete columns. These tests, known as 1970-1, 1970-2, 1970-3, and 1970-5, are summarized on Tables 3 and 4 and are shown in comparison with both plant and the earlier pilot plant tests on Figure 1. It had been suspected that the low recoveries were due to too coarse a crush and the 1970 test series were developed to show the effect of particle size. Test 1970-1 was made on a coarse material ranging in particle size from 1/2 inch to 2+ inches. Test 1970-2 was made on material crushed to smaller than 1/2 inch. Comparison of the two tests shows a marked increase in rate of recovery with the smaller crush size. Test 1970-3 also used the crush size of less than 1/2 inch but, in addition, used a higher initial acidity level of 45 grams per liter. This dropped off to the 14 - 15 grams per liter level used in the Tests 1 and 2 only after the ninth day of this test. This test showed a higher initial rate of recovery but about the same ultimate recovery as the Test 2. Test 1970-5 used a somewhat coarser crush, being mainly material up to 3/4 inch in size with a wider size range. Results of this test indicated that very satisfactory recoveries could be obtained with this larger crush size. Based on comparison of these 1970 tests, which were completed in early 1971, it was decided that a crush size of 3/4 inch or smaller was probably optimum for this ore body.

It can be noted from Figure 1. that the relationship between recovery and time is approximately linear over the range of 15 - 60% recovery on a semi-log plot. This type of relationship is one that would be expected if the reaction was rapid on the surface exposed copper and the overall rate was limited by diffusion of leaching solution into the interior of the ore particles. This mechanism would also explain the extreme dependence of overall rate of recovery on crush size. Clearly, it would be anticipated that this type of dependence would be a function of the porosity of the host rock with less dependence being shown with more porous host rocks.

The recovery curves shown versus time from pilot plant data are based on continuous leaching of a single portion of ore. In leaching of a commercial heap, conventionally new crushed ore is placed on top of ore which has been previously leached and the leach pads moved up over the new ore. As a result, the ore on the top of the heap has been leached for a relatively short time while the ore on the bottom of the heap has been leached for a long time. A conventional measure of recovery from a field operation is to measure the actual copper produced and divide it by the tons of new copper added to the heap over a given period of time. Clearly, this percent recovery is different than that obtained by leaching all the ore for the same period of time. The relationship between the recovery for the continuous addition of new ore to a heap with that of recovery from ore continuously leached for the same time is developed in Appendix 1 for the case in which the rate of addition of new ore is constant. As shown in the appendix, the percent recovery after a given time interval in this case is equal to the percent recovery for ore leached continuously less the slope of the leach curve. Alternatively, as shown in the appendix, an equivalent time can be developed; this time being defined as the time it would have taken to leach the heap all as one body to get the same percent recovery obtained in the actual case of leaching the body by adding fresh ore continuously over a longer period of time. As shown in the appendix, this equivalent time is 37% of the total elapsed time involved during which ore is continuously and uniformly added to the heap.

All copper recoveries shown are based on total copper measurements. Experience with this ore suggests that "acid soluble copper" measurements are unreliable. Conventional tests for "acid soluble copper" appear to fail to indicate the presence of more slowly dissolving oxide copper compounds (possibly dioptase) and do not reflect presence of leachable sulfides such as chalcocite or covellite. The San Juan surface ore is predominantly chrysocolla with 5 - 10% chalcocite plus cuprite and essentially no chalcopyrite. Thus, essentially all the copper is leachable provided sufficient ferric iron is present in the leach solution to handle the minor amounts of chalcocite and cuprite.

Field Leaching Experience

Leaching was started in the field in February of 1970. Initially a coarse crush size was selected in an effort to minimize possible fines handling problems. Initial crush size was set at about 70% less than 1-1/2 inch with some ore chunks ranging in size up to 3-1/2 or 4 inches. Blinding of the coarse leach heaps due to fines was observed early in field operations and screening of fines was initiated after the first month. Fines were stockpiled for later processing. However, low recoveries were still experienced. In June 1970 isolated heap tests were run on heaps 15 - 18 in which the barren and pregnant solutions were measured and analyzed directly from these heaps. The data for the isolated heap tests, presented in Table 2, showed a recovery of only about 11% after ten days. The first five months' operation of the field showed a 20% recovery of the total copper added to the heap. These data, as well as the 15 - 18 data, are plotted on Figure 1 with the field data being adjusted to an equivalent time as outlined before. It can be noted that the actual operations for the first five months appear to be a reasonable extrapolation of the isolated heap tests with this coarse material.

Crush size was reduced to 95% less than 1-1/2 inches in June of 1970, and the 1970 series of pilot plant tests discussed above were initiated to determine the effect of particle size and establish the optimum crush size. Somewhat improved recoveries were obtained in the latter part of 1970 and new crushing equipment was installed in December which permitted a three-stage crush and reduction of size to 90% less than 3/4 inch. An isolated heap test on Heap 33 of this size was run in December of 1970. This showed improved results with approximately 27% recovery after 10 days. These data are shown on Table 2 and plotted on Figure 1. Recovery levels here are approximately the same as the 1969 pilot plant data but still significantly below the 1970 - 5 pilot unit tests on comparable size ore.

Actual field operation showed increased production in the first six months of 1971 on the finer crush ore. The data for the first six months of 1971 adjusted on equivalent time basis are plotted on Figure 1 and appear to be a reasonable extrapolation of the Heap 33 data. Copper recovery in the first six months of 1971, as presented on Figure 1, was adjusted to allow for production from other parts of the heaps. This adjustment is shown in Appendix 2.

Mining of new ore was stopped in July of 1971 and the heap has continued to be leached continuously since that time. In late 1973, a series of six Becker drill holes were made in the main heap in the area where the finer crushed ore was to determine the percent copper remaining. In January 1974, a check of these results was done by an outside laboratory and three additional Becker drill holes were made. These data and the analysis of the recovery from them are shown on Tables 6 and 7. The location of the holes in the heap are shown on Figure 2. From these data the percent copper extracted was estimated by two methods. Method 1, shown on Table 6, compared the average copper remaining with the average initial copper content based on production records for the average fine crush ore placed on the total heap. This showed a range of 71 - 79% extracted or an average of 75%. In Method 2, shown on Table 7, the actual level of copper at each hole location was averaged based on production records of the assays of the production that went on each heap at each location. This analysis showed essentially the same total percent extraction. The overall level of 75% extraction is shown on Figure 1 and appears to be a reasonable extrapolation of the data reflecting Heap 33 and the first six months 1971 results.

Sands and Slimes Leaching

The fines stockpiled in early operations were processed in a wet classifier, starting in April 1971, to separate them into a sands fraction and a slimes fraction; the slimes being the material which would not settle out. The sands were placed in separate heaps and leached by ponding or sprinkling. The slimes were allowed to settle in a pond. The classifying liquid used was barren solution. This operation continued until December 1971 at which time all of the fines stockpiled had been processed. Table 8 summarizes the field data obtained. It was not practical to obtain deep ore samples of the sand heaps. However, samples obtained from five different locations in December 1971 showed an average copper content of 0.17% compared to initial copper content of 0.89% or recovery of 81%. A recheck of the sands heap with additional sampling in 1973 showed levels of about

0.2% or recovery of 77%.

The slimes were allowed to settle over a period of two years and the original copper content of 1.36% estimated from production records was reduced to about 0.4%. However, the latter is based on only two spot samples and may not be truly representative.

Material Balance Analysis

The Becker drill data and the sampling of the sands heaps indicate overall an extraction of about 75% of the total copper. Assuming process losses in the league of 5%, this would mean a recovery of copper of about 70% after leaching for slightly less than 3 years. This recovery relates only to the finer crush material as produced after December of 1970. Such a level recovery appears to be at least as good, if not better than that reported by other major leaching operations. To check the reasonableness of the recovery based on the Becker drill data, an attempt was made to carry out an overall material balance of the operation to compare copper shipments and copper placed on the heap. Table 9 summarizes the production from February 1, 1970 to December 31, 1970. During this period some 723,000 tons of ore of .87% copper were placed on the heaps. Of this, approximately 109,000 tons were taken out of the main heap, recrushed and leached in the separate area, and about 89,000 tons of fines produced were stockpiled. Copper shipped during the period was about 1,663 tons or 26% of the copper placed on the main heap. Table 10 shows the estimated material balance for the period January 1, 1971 to December 31, 1973. Finer crushed ore, totaling 431,000 tons, was produced during this period. Of this ore, it is estimated that around 65,000 tons were on the front edges of the heap and therefore not leached and about 366,000 tons, averaging .67% copper, were leached; and based on the Becker drill data about 74% of the copper therein was extracted. During the same period the sands and slimes, as discussed previously, were leached to an extraction level of about 77% and 70%, respectively. As old coarse ore underlay part of the new finer crush material, allowances must be made for copper from this source and from recrushed material and low grade material, which were also subject to leaching during this period. Judgment estimates based on available data have been made to allow for the estimated production from these sources. Thus, using the Becker drill data extraction rates and these judgment estimates, a total amount extracted of about 4,106 tons is estimated. Deducting estimated process losses of 5% gives a net production, estimated from the extraction rates indicated from Becker drill data, of 3,900 tons which compares with an actual shipment of 3,400 tons over this period. The difference of 465 tons is unaccounted and represents the approximately 10% of the copper in the ore to leach. This unaccounted for difference may reflect one of the following: errors in assays in the Becker drill holes and sands samples; errors in the input quantities of copper due to assay or tonnage measurement errors; errors in estimation of the production from recrush, low grade and old coarse ore; errors in process losses; or non-representativeness of the sampling of the leached heap and sands. This type of material balance is essentially impossible to make in an absolutely rigorous manner for a field operation over a long period of time such as this. However, it's believed a sufficiently reasonable closure has been obtained so that the levels of extraction indicated by the Becker drill data and the sands sampling can be assumed to be reasonably representative of that obtainable from leaching this ore body with the finer crush technique.

Acid Consumption

Acid consumption as observed in the field and as compared with the pilot plant is summarized in the table below.

		Acid Lb/Lb Copper
Field	2/70 - 12/70	14.41
	1/71 - 6/71	9.10
	May 1971	6.50
Pilot Plant	1970-1	3.94
	1970-2	4.12
	1970-3	3.50
	1970-5	4.02

It can be noted above that initial acid consumption was relatively high and dropped to a level of about 9 lb/lb of copper in the first six months of 1971. The best results were achieved in May of '71, a level of 6.5 lb/lb. Pilot plant acid consumptions ranged from around 3.5 to 4 lb/lb. It is believed that the high initial acid consumption was related to the low copper recoveries associated with the coarse ore size. Acid is consumed by the other non-copper bearing minerals. With the lower efficiency due to coarse ore size, the selectivity of acid usage for extracting copper is poor. Further improvement in field operations below the 6.5 to 9 lb/lb level demonstrated in the first half of 1971 may be achieveable. However, if the heaps are leached for a long period of time to obtain higher ultimate recoveries, somewhat higher acid consumption than that obtained in the pilot plant will probably result.

Iron Consumption

The iron consumption in the launder obtained in the field and compared with the pilot plant is summarized in the following table.

		Iron Consumed Lb/Lb Copper
Field	2/70 - 12/70	2.78
	1/71 - 3/71	1.96
	4/71 - 6/71	1.17
Pilot Plant	1970-1	1.03
	1970-2	1.22
	1970-3	1.19
	1970-5	1.81

It should be noted that the field iron consumption improved from over 2 lb. /lb. in 1970 to a level of 1, 17 in the second quarter of 1971 comparable to that in the pilot unit. Major factors influencing iron consumption are the acidity level of the solution, the amount and type of iron to which the pregnant solution is exposed in the launder, and the ferric iron content of the circulating solution.

Use of Spent Alkylation Acid

Producers Minerals Corporation has used spent alkylation acid for leaching at the San Juan property. This acid is obtained from an affiliated company which supplies fresh acid to Chevron's refinery in El Paso, Texas, and receives back the spent alkylation acid. The spent acid is available at a lower cost and has proved to be an effective leaching agent. Producers Minerals' affiliated company had previously sold the spent alkylation acid to the operator of the property before it acquired the lease.

Spent alkylation acid from a refinery contains about 90% sulfuric acid, about 5% water and about 5% hydrocarbons. The hydrocarbons are cyclic diene sulfonates which are effective surface active agents and are extremely polar acidic compounds. On dilution of the acid, most of the hydrocarbons come out of solution if the dilution is carried down to a range of 15 to 65%. The dilution process, particularly to concentration levels of 30 - 65%, results in the release of a large amount of heat and a rapid rise in temperature unless external cooling is used. Under these higher temperature conditions the diene hydrocarbons tend to polymerize into tars or coke. However, with dilution under lower temperature conditions, the hydrocarbons separate as a liquid oil. On fully diluting down to a 1% or so sulfuric acid solution, the hydrocarbons are suspended in the form of an emulsion within the aqueous solution and do not form a separable phase.

Comparative tests of fresh versus spent acid were run in 1968 and again in 1970 which showed that on the San Juan ore, spent acid is just as effective a leaching agent as fresh acid. These data are summarized in Table 5. Spent acid has the significant advantage in PMC's case of inhibiting bacterial oxidation of ferrous iron to ferric which would subsequently be reduced back to ferrous in the launder causing a significantly higher iron consumption.

If it is desired to leach cuprite or chalcocite (as contrasted to chrysocolla) the existence of a minimum level of ferric iron is desirable.

To determine the effect of the hydrocarbons on the growth of the bacterium thiobacillus ferro oxidans, the bacterium which oxidizes ferrous iron to ferric in many leach heaps, PMC had an outside firm carry out certain research experiments. These experiments indicated that at the concentration of hydrocarbons that existed in the barren solution the growth of the bacteria was inhibited, and considerably lower concentrations would be required to allow this type of bacteria growth. However, these studies also pointed out that at the low pH desired for leaching the refractory chrysocolla, bacteria growth would also be inhibited by the acidity level alone. Thus, bacterial oxidation of ferrous to ferric iron would probably not be feasible even using fresh acid in the processing of this ore at the acidity levels desirable to get rapid extraction. As ferric iron is required in leaching only for dissolution of chalcocite and cuprite, and as these minerals are only present in very minor degree in this ore, the presence of ferric iron is not critical and too much ferric iron is undesirable. Analyses of PMC's circulating barren solution indicates that it has a sufficient level of ferric iron in it to carry out the leaching of the small amount of the chalcocite and cuprite present. The exact source of this ferric iron has not been firmly established but it is believed to have come either by dissolution of ferric iron containing minerals in the ore or by the auto-oxidation of ferrous iron to ferric due to the presence of trace amounts of SO_2 dissolved in the sulfuric acid.

PMC has employed spent acid using the following three different techniques.

(1) For the period 1970-73, spent acid was diluted to about 65%, the hydrocarbons skimmed and the skimmed acid added to the barren solution. While this method preremoved the majority of hydrocarbons, the method was relatively costly.

(2) During the period 1973-74, concentrated spent acid was distributed over uncrushed low grade ore. The acid was allowed to be soaked up by the rock and then the low grade was leached with barren solution. Field observations show that with this technique most of the hydrocarbons in the acid polymerize to small granules of coke on the top few inches of ore. No

plugging of the heap or tar deposits in the heap were encountered. Over a period of about one year application of concentrated in this way resulted in the advantageous breakup of the uncrushed ore into small particles with the exposure of much more surface for leaching. It is believed this results from the heat generated from certain chemical reactions occurring. This approach can result in higher acid consumption.

(3) During 1974-75 PMC added spent acid directly into the line carrying the tail solution from the launder to the barren pond. This method provided rapid dilution down to one percent and without any significant tar separation. Very minor amounts of tar collect on the side walls of the barren pond. No problems with plugging of equipment or tar depositions on the heaps have been encountered. It is believed that the hydrocarbons probably eventually polymerize to small coke granules which drop out of solution without adverse effect.

PMC's experience indicates that spent acid can be used employing any of the above techniques, the choice hinging on economics. The only important cautions are to avoid dilution to the intermediate range of 15-70% acid unless skimming is to be practiced; and in the case of dilution to 1% without skimming to be sure that rapid mixing occurs.

PMC Data on Leaching Uncrushed Ore

The previously reported data indicate very high leaching recoveries on the San Juan ore when the ore is crushed to a fine size. However, the cost of hauling and crushing is expensive and there would be significant economic advantages for leaching uncrushed ore if satisfactory ultimate recoveries can be obtained by such leaching, even if the length of time required to obtain a given recovery is considerably longer than for finely crushed ore. The economies come from two sources: first, the direct elimination of the crushing cost and the associated additional hauling; and second, if satisfactory recoveries can be obtained by leaching uncrushed ore, then only part of the total ore needs to be hauled from the pit while the remaining part of the ore body can be leached in place, completely eliminating the hauling costs for this part. Preliminary estimates indicate that if this technique is followed, it would only be necessary to haul approximately 50% of the total ore and waste from the pit.

PMC's experience, showing fairly rapid reduction in ore size from the application of concentrated acid, has led to additional experimental work to determine under what conditions leaching of uncrushed ore can provide sufficient decrepitation of the ore over a period of one to two years so that satisfactory ultimate recoveries can be achieved. This rate of decrepitation and penetration of acid into the interior of ore chunks is highly dependent on the nature of the host rock and the distribution of the copper within the host rock. Part of the San Juan ore body is in relatively dense monzonite host rock in which part of the copper is on the fracture planes, but part is broadly disseminated through the host rock. This appears to be the most difficult material to break down. The remainder of the San Juan ore body is in an andesite host rock which is much more friable and in which the copper appears to be primarily on the fracture planes. Due to the differences in host rock, small leaching tests were run on both andesite and monzonite ores. These data are summarized in Table 11. The test on andesite ore was run on a run-of-mine ore size which varied from zero to three inches. The ore was initially soaked with a partially diluted spent acid and then leached with normal barren. This run indicated high recoveries of approximately 40% in eleven days and 60-70% in twenty-eight days of leaching. These data can be considered approximate only as overall material balance was not obtained in this test.

Two tests were run on the monzonite ore. In the first, the ore was soaked with concentrated spent acid and then leached with barren solution. In the second, the soaking was done with a partially diluted spent acid. The primary objectives of these tests were to measure the amount of physical disintegration of the large ore chunks over a period of time. These data indicate that the concentrated acid was significantly more effective in reducing the size of the ore chunks. Satisfactory levels of recovery of approximately 36% in sixty days were obtained with a presoak with concentrated acid with an acid consumption of 12 Lbs/Lb copper. Inadequate material balance was obtained in the test with partially diluted acid soak to get reliable comparative recovery, although the recovery appeared to be significantly lower.

From these data, it appears that satisfactory ultimate recoveries can be obtained on leaching uncrushed ore. In the case of andesite, it appears that recoveries are higher, and no pre-treatment with concentrated or partially diluted acid may be required. To get high ultimate recoveries in the case of monzonite, it appears that periodic soaking with concentrated acid may be desirable. However, the data indicate that if this is done in a controlled manner the acid consumption, even with the application of concentrated acid to the host rock, can be maintained at reasonable levels.

Following completion of the laboratory tests, plant scale leaching of uncrushed ore was initiated in the pit to confirm satisfactory production rates could be achieved in commercial operation.

TABLE 1

Column Leach Tests 1969-1 and 1969-2

Test Basis
- Column Leach Test in diameter concrete columns
- Ore charge 28,380 Lbs. (Test 1), composite surface sample from pit area
- Equilibrium recycle of leach solution after precipitation of copper by iron
- Copper production and recovery as measured by cement assays
- Skimmed spent alkylation acid for makeup

Days	Barren GPM	Copper GPL Preg	Copper GPL Barren	Cumulative Recovery Lbs Cu	Cumulative Recovery %
Test 1969-1					
3.3	0.56	2.48	0.61	46.2	14.0
10.3	0.56	0.73	0.13	64.8	19.6
13.3	0.56	0.86	0.16	95.7	29.0
19	0.56	0.49	0.11	128.4	38.9
26	0.56	0.40	.13	147.1	44.6
33	0.56	0.34	.16	158.8	48.1
40	1.12	0.19	.10	172.9	52.4
47	1.12	0.18	.08	180.6	54.7
52	2.33	0.12	.06	192.2	58.2
Test 1969-2					
8.6	0.50	1.18	0.38	66.1	24.0
15.6	0.50	2.24	0.49	101.2	36.9
22.6	0.50	0.89	0.22	124.3	45.3
29.6	0.59	0.36	0.16	134.4	49.0
36.6	0.59	0.15	0.10	140.1	51.1

Ore Charge Assays

	Test 1 Wt %	Test 1 % Tot. Cu	Test 1 % Acid Sol. Cu	Test 2 Wt %	Test 2 % Tot. Cu	Test 2 % Acid Sol. Cu
+ 1.05 in	20.1	1.40	1.34	26.0	1.12	1.04
+ .742 in	37.8	1.18	1.10	30.8	1.22	1.18
+ .525 in	16.2	0.94	0.88	15.5	0.97	0.92
+ .371 in	9.3	0.98	0.88	10.3	1.20	1.14
+ 3 Mesh	4.4	1.00	0.96	6.1	1.18	1.16
+ 9 Mesh	7.1	1.12	1.06	6.8	1.16	1.08
- 9 Mesh	4.1	1.34	1.26	4.5	1.36	1.26
	100.0	1.164	1.093	100.0	1.15	1.09

TABLE 2

Isolated Heap Tests

Test Heaps 15 - 18

Test Summary
- June 1970 Test
- Heap Tons 49, 190, % Copper 0.92, Area 45, 000 square feet
- Ore preparation 2 stage crush

Days	Barren GPM	GPL Copper		GPL Acid		Cumulative
		Preg	Barren	Preg	Barren	%Recovery
1	582	1.18	.12	8.0	14.8	0.8
2	892	1.29	.14	6.3	12.9	2.1
3	1200	1.03	.12	5.7	8.4	3.5
4	1200	.93	.12	5.4	9.9	4.9
5	1200	.93	.15	7.3	12.2	6.1
6	1260	.99	.25	9.6	12.3	7.3
7	1260	.88	.18	9.4	12.8	8.5
8	1260	.69	.18	7.7	9.6	9.4
9	1260	.59	.10	6.8	8.9	10.2
10	1260	.54	.09	8.0	10.0	10.9
11	1260	.55	.10	8.0	9.6	11.7

Test Heap 33

- December 1970 Test
- 38, 500 Ton Heap, 0.70% Copper.

Days	Cumulative % Recovery
1	2.6
2	7.0
3	11.0
4	14.5
5	17.5
6	20.6
7	22.4
8	24.2
9	26.0
10	27.5
11	28.5
12	29.5
13	30.5

TABLE 3

Column Tests 1970-1 and 1970-2

Test Basis
- Column 4'6" ID x 18'5"
- Recirculated barren solution with copper precipitated by iron
- Flow rate 3 gpm/100 sf, barren acidity approx 15 gpl
- Crusher Feed ore sample
- Copper recovery by cement assay
- Skimmed spent acid for makeup

Ore Charge Analysis

	1970 - 1 Wt % on	% Tot. Cu	% Acid Sol. Cu	1970 - 2 Wt % on	% Tot. Cu	% Acid Sol. Cu
+ 1"	66.8	0.96				
+ 3/4"	21.4	1.03				
+ 1/2"	7.3	.99				
+ 1/4"	4.0	1.12		54.6	.94	
+ 10 Mesh	0.5	1.82		40.0	1.07	
- 10 Mesh				5.4	1.46	
		0.95	0.90		.98	.79

Test Data

1970 - 1 Day	% Recovery	Cum. Consumption/Lb Cu Acid	Iron	1970 - 2 Day	% Recovery	Cum Consumption/Lb Cu Acid	Iron
4	11.6		1.07	3	13.4		1.31
7	17.1		.98	5	26.5		1.13
11	23.4		1.01	7	36.2		1.12
16	28.1		1.00	9	43.2		1.14
17	29.0	3.94	1.03	12	51.7		1.15
				16	58.9		1.22
				17	61.0	4.12	1.22

TABLE 4

Column Leach Tests 1970-3, 1970-5

Test Basis

- Same pilot unit as 1970-1 and 1970-2
- Flow rate approx 3 gpm/100 sf
- Copper recovery by cement assays
- Crusher feed ore sample
- Barren acidity
  1970-3 Initial 45 gpl ., Day 3 25 gpl, Day 9+ 14-15 gpl
  1970-5 " 20 gpl , " " 16 gpl, " " 9-13 gpl
- Skimmed spent acid for makeup

Ore Charge Analysis

	1970-3 Wt%	1970-3 % Tot Cu	1970-3 % Acid Sol Cu	1970-5 Wt%	1970-5 % Tot Cu	1970-5 % Acid Sol Cu
+ 1-1/2 in				0.0		
+ 3/4				31.0		
+ 3/8				39.1		
+ 1/2						
+ 1/4	56.3	0.93		12.0		
+ 1/8				9.5		
- 1/8				8.4		
+ 10 Mesh	40.5	1.10				
= 10 Mesh	3.2	1.32				
		0.98	0.83		0.739	-

Test Data

1970-3 Day	% Recovery	Cum Consumption/Lb Cu Acid	Iron	1970-5 Day	% Recovery	Cum Consumptio/Lb Cu Acid	Iron
3	25.6			4	29.8		
5	38.6			8	42.8		
8	48.0			18	52.6		
13	54.6	3.5	1.19	43	62.9	4.02	1.81

TABLE 5

Comparison Tests Spent Acid vs Fresh

Test 1968-1

Basis - 1/2 inch ore (3.0% Copper) in glass cylinders treated with 7% acid for 16 hours, 3.5% acid for 6 days.
Spent acid prepared by dilution, skimming and filtration.

	Spent Acid	Fresh Acid
% Recovery	44.8	45.1
Acid consumption	- - - Same - - - - - -	

Test 1970-1

Basis - Bottle roll test 500 gms ore, 1,000 ml solution
Acid strength 20 gpl, test duration 24 hours
Spent acid prepared by skimming.

		Spent Acid	Fresh Acid
Charge	% Total Cu	.80	.80
	% Acid Soluble Cu	.78	.78
% Recovery			
	Total Cu	76.5	75.5
	Acid Soluble Cu	78.5	77.5

TABLE 6

PMC COPPER EXTRACTION

BASED ON

BECKER DRILL DATA

METHOD I

Assay by Method	Depth	% Copper Remaining in Ore PMC Colorometric	Jacobs Long Iodide
Hole N-Center	25	.100	-
Hole S-Center	55	.115	.131
Hole W-South	41	.098	
Hole W-North	40	.123	.159
Hole E-South	50	.218	
Hole E-North	43	.195	.220
Peacock 1A	60		.210
Peacock 2	39		.162
Peacock 3	24		.277
Average		.141	.193
Average initial copper content of ore before leaching		.670	.670
% of copper extracted		78.9	71.2
Overall average % extracted		75.0	

TABLE 7

PMC COPPER EXTRACTION

BASED ON

BECKER DRILL DATA

METHOD II

	- - - - - PMC - - - - -			- - - - Jacobs - - - - - -		
Hole	Initial % Copper	Final % Copper	% Extraction	Initial % Copper	Final % Copper	% Extraction
N - Center	.654	.100	84.7			
S - Center	.705	.125	82.2	.705	.148	79.0
W - South	.685	.104	84.8			
W - North	.597	.117	80.4	.597	.140	76.5
E - South	.509	.194	61.8			
E - North	.877	.125	85.7	.877	.205	76.6
Peacock -1A			-	.607	.256	57.4
Peacock - 2			-	.695	.162	76.6
Peacock 3			-	.527	.278	47.3
			80.0			68.9

Overall Average 74.4

TABLE 8

Sands and Slimes Leaching

Field Data

Sands Leaching

Period - Sands leached from March 1971 to December 1971. Fines stockpile processed thru classifier to separate sands from slimes. Sands leached in separate heaps.

Sands charged to Heaps 159,000 tons 0.89% Copper

December 1971 Sands Heap Samples

	% Copper
High North Pit	.21
Upper West Pit	.21
Upper East Pit	.18
Work Face West Pit	.13
Work Face East Pit	.13
	0.172

% Recovery = (.89 - .172)/.89 = 80.6%

Recheck of Sands in 1973 showed avg. of 0.20% Copper.

Slimes Leaching

Slimes charged to pond 53,100 tons 1.36% Copper

December 1973 Spot samples 0.40% Cu (Avg. 2 samples)

% Recovery (1.36 - .40)/1.36 = 70%

TABLE 9

PMC SUMMARY OF PRODUCTION

FEB. 1, 1970 to DEC. 31, 1970

	Tons	% Copper	Tons Copper
Ore to Heaps	723,060	0.87	6,290
Ore from Heap Recrushed	108,940	0.50	
Fines Stockpiled	89,075	1.16	
Copper Shipped			1,663
Shipments % Copper to Heap			26

TABLE 10

PMC SUMMARY OF PRODUCTION

JAN. 1, 1971 - DEC. 31, 1973

	Tons Ore To Leach	% Copper Initial	Tons Copper To Leach	% Copper Final	Tons Extracted	% Recovery
No. 1 Dump						
New ore to heap	431, 160	0.67	2, 888			
Ore not leached	65, 200	0.67	437	—	—	—
New ore leached	365, 960	0.67	2, 451	0. 17	1, 814	74
Sands leached	159, 000	0. 89	1, 415	0. 20	1, 089	77
Slimes leached	53, 100	1. 36	721	0. 40	504	70
From recrush (1)					148	
From old ore (2)					292	
From low grade					259	
Total			4, 587		4, 106	
Process losses					205	
					3, 901	
Shipments					3, 436	
Non closure					465	
Non closure % new copper to leach			10			

NOTES: (1) 77, 100 tons to recrush in 1971 at 0. 46% (35% recovery=124 tons), 109, 000 old recrush from 1970 added 15% recovery + 24 tons; total 148 tons.

(2) Approx. 250, 000 tons coarse ore (originally 0. 87% copper) under fine ore, added 13% recovery gives 292 tons.

TABLE 11

PMC Test Data on Uncrushed Ore

Test	1974 - 1	1975 - 1	1975 - 2
	Andesite	Monzonite	
Ore Size	0 - 3"	4 - 6"	
Initial Acid Soak	Partially Diluted Spent	Concentrated Spent	Partially Diluted Spent
Heads % Copper		0.71	-
% Recovery	40% in 11 days	36%	-
	60 - 70% in 28 days	in 60 days	-
Material Balance	NA	94.5%	-
% of Heads reduced in size to less than 1-1/8 inches	-	8.3%	2.4%
	-	in 60 days	in 60 days
Acid Consumption Lb/Lb Copper	15	13	5.6

APPENDIX 1

Equivalent Time for Continuously Built Heap

For leaching an ore

f = fraction extracted at end of t days of leaching

By experimental observation

$f = a + b \ln t$ where a & b are constants.

If ore is continuously added at constant rate on top of heap being leached, and

"F" = fraction of total copper in heap extracted at end of time "T"

and p = fraction of heap measured from top

Then $$F = \int_0^1 f \, dp = \int_0^1 (a + b \ln t) \, dp$$

If rate of new ore added is constant, then length of time "t" any fraction dp has been under leach is proportional to depth in heap, or

if $p = 1.0$	$t = T$
$p = 0.0$	$t = o$
$p = p$	$t = pT$

so $dp = \frac{dt}{T}$

or
$$F = \frac{1}{T} \int_0^T (a + b \ln t) \, dt$$

$$= \frac{1}{T} \left[a t + b t \ln t - b t \right]_0^T$$

$$= \frac{1}{T} (aT + bT \ln T - bT)$$

$$F = (a + b \ln T - b)$$

Since $a + b \ln T = f_T$ i.e. % extracted if all had been leached for entire time T

~~Then $F = f_t - b$~~

Defining Te as equivalent time corresponding to F

Then $a \;\&\; b \ln Te = a \;\&\; b \ln T - b$

or $\ln \frac{Te}{T} = -1$

or $\frac{Te}{T} = 1/e = 0.367$

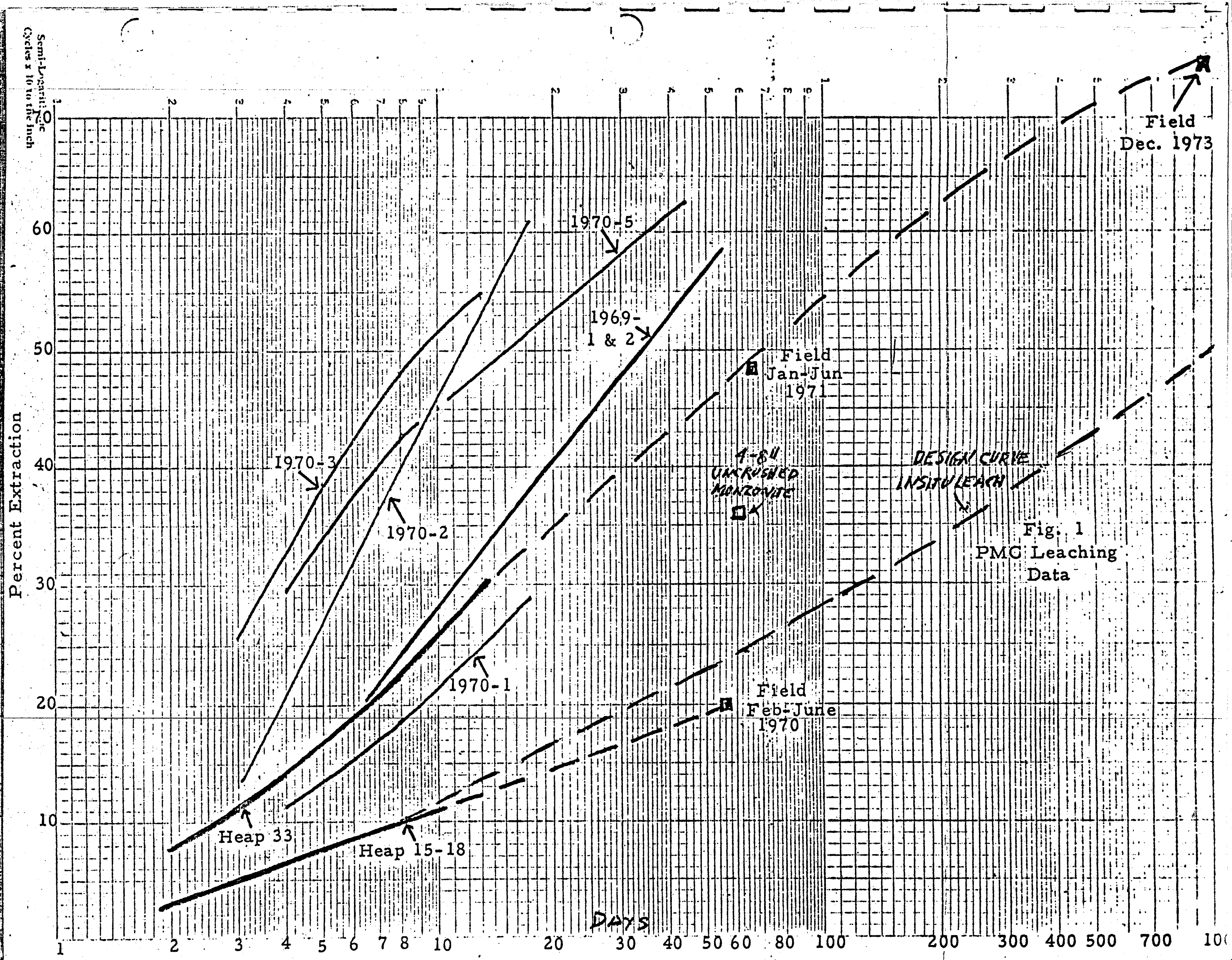


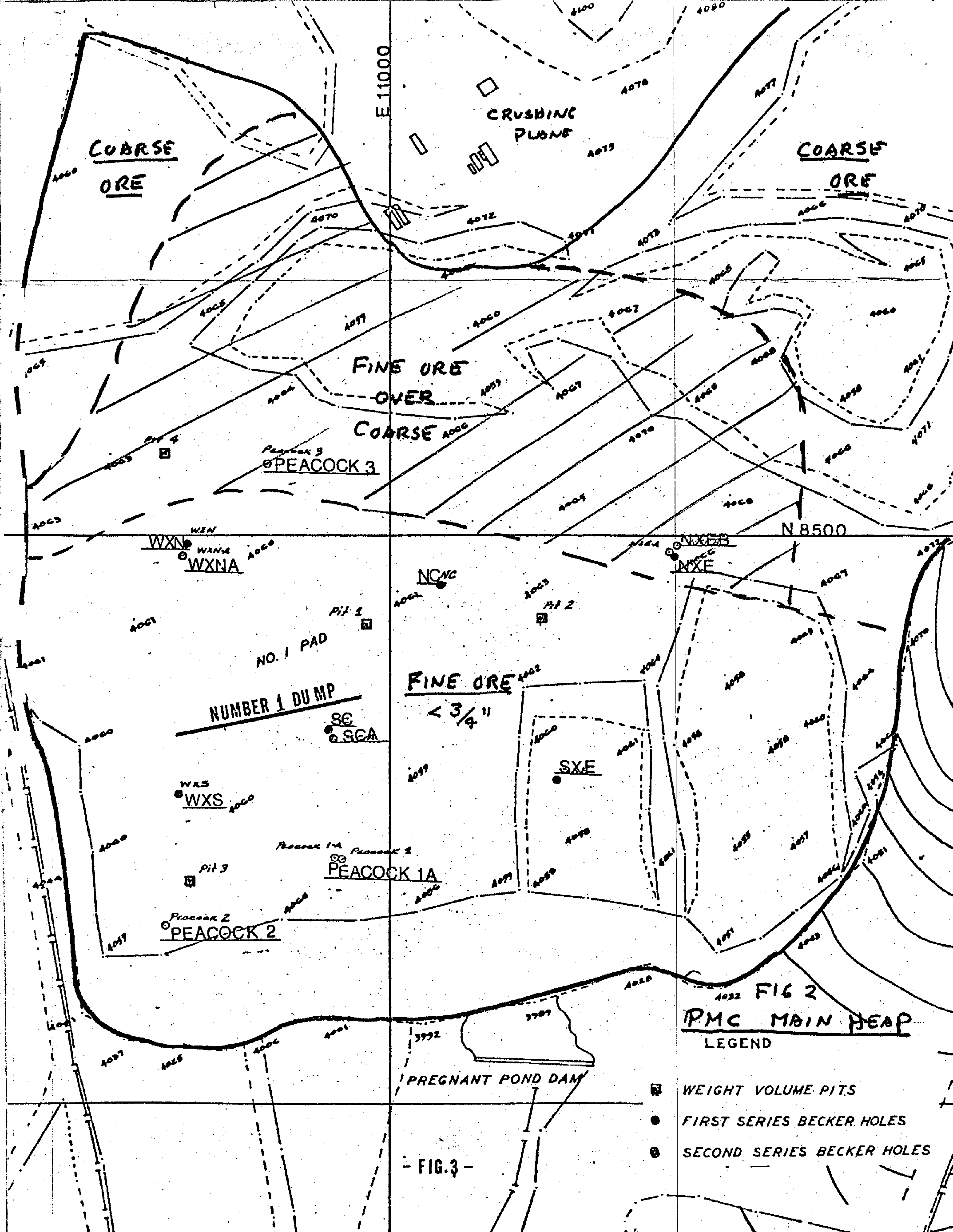
Fig. 1
PMC Leaching
Data

APPENDIX 2

COPPER EXTRACTION 1971

Period Jan 1 - June 30, 1971

Ore to Main Heap	Tons		431,160
	% Copper		0.67
Copper to Main Heap	Tons		2,872
Copper Shipped	Tons		1,560
From Fines		312	
From low grade old ore, & recrush		125	
Subtotal		437	437
Copper from Main Heap			1,123
Losses & Nonclosure			269
Copper Extracted	Tons		1,392
Copper Extracted	% Copper to Heap		49.5%



COARSE ORE
CRUSHING PLANT
COARSE ORE
E 11000
FINE ORE OVER COARSE
PEACOCK 3
Pit 4
N 8500
WXN
WXNA
NC
NXEB
NXE
NXCC
Pit 1
Pit 2
NO. 1 PAD
NUMBER 1 DUMP
FINE ORE < 3/4"
SC
SCA
SXE
WXS
PEACOCK 1A
Peacock 1-A
Peacock 1
Pit 3
PEACOCK 2
PREGNANT POND DAM
FIG 2
PMC MAIN HEAP
LEGEND
WEIGHT VOLUME PITS
FIRST SERIES BECKER HOLES
SECOND SERIES BECKER HOLES

- FIG.3 -

APPENDIX B

University of Arizona Press, 1966

THE SAFFORD COPPER DEPOSIT, LONE STAR MINING DISTRICT, GRAHAM COUNTY, ARIZONA

BY R. F. ROBINSON AND ANNAN COOK

FIGURE 2.—Schematic geologic view of the Gila Mountains near Safford, Ariz.