62 PAGES   824528

Accompanying Maps include in Report are:

1. Geological Map
2. X-Sections = Diagrams 9 to 16.

} can't find in Kamloops, Ed Becker will ask Utah for extra copy.

This ~~xerox~~ copy for Ken ~~to xerox + return borrowed copy to Kamad~~

GEOLOGICAL, METALLURGICAL
AND MINING
REVIEW OF THE KAMAD SILVER CO. LTD.
HOMESTAKE PROPERTY

UTAH MINES LTD.

APRIL, 1984

GEOLOGY

ENGINEERING

METALL-

MINING

BARITE

UTAH MINES LTD.

EXPLORATION DEPARTMENT

INTER-OFFICE MEMO

SUITE 1600, 1050 W. PENDER STREET • VANCOUVER, B.C., CANADA V6E 3S7

(604) 683-6921

ATE: April 18, 1984

FILE NO:

D: G.F. Andrews
VANCOUVER

COPIES TO:

ROM: D.N. leNobel/K.W. Pickering
VANCOUVER /ISLAND COPPER

JBJECT: THE KAMAD HOMESTAKE PROPERTY

FERENCE:

The attached report presents the results from the examination and evaluation on the above by staff from the Mining Technical Services and Exploration Groups. An overview of the Homestake geology and a current tonnage reserve estimate precedes the section on Geology and a similar introduction is presented before the second major section in the report on the Metallurgical and Mining aspects of the property. This report summarizes well the subject matter.

D.N.leNobel/K.W. Pickering

DNleN & KWP/cs

Attach.

GEOLOGY

ENGINEERING

METALL-

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(604) 683-6921

TE: April 18, 1984 FILE NO:

: D.N. LeNobel COPIES TO:

OM: D.N. Duncan/P. Burt

BJECT: KAMAD SILVER - HOMESTAKE MINE REPORT

FERENCE:

Attached please find a copy of the geological report on Kamad Silver's Homestake Mine. In brief, the deposit consists of lenticular banded barite ore zones which are probably volcanigenic in origin. The ore consists of barite, sphalerite, galena, proustite and tetrahedrite. The mine area is intensely faulted, with two styles of faulting evident - block faulting and thrust faulting. Three ore zones - 300, 400 and 500 - have been reported in previous investigations of the mine area. It is proposed that the 300 zone is actually an overthrust extension of the 400 zone. The 300 and 400 zones may also be overthrust extensions of the 500 zone, but data is insufficient to determine this conclusively.

Ore reserves calculated for the mine area are:

measured (probable) = 234,742 tonnes
maximum possible = 823,550 tonnes

These reserves are based on data which is, at best, tenuous. Further drilling is required in the mine area to increase the reliability of ore reserve estimates. Homestake Creek follows a fault trace, with the east side down-dropped relative to the west side. Geochemical sampling and diamond drilling on the east side of the fault may lead to the discovery of new ore pockets, thus increasing reserves.

Phil Burt and I have spent approximately 1-1/2 months working on this report. Two weeks were spent in the field, involving geological mapping and sampling of the mine workings and surface exposures. An attempt was made to examine the Kamad 7 claim block, but deep snow prevented an adequate appraisal. Approximately one month has been spent in the office sorting through the data, preparing maps, updating cross sections, calculating ore reserves and writing the report.

DND/bln

GEOLOGICAL
REVIEW OF THE KAMAD SILVER CO.
HOMESTAKE PROPERTY

By: P.D. Burt,
D.N. Duncan and
J.R. Deighton
Utah Mines Ltd.

April 1984

ABSTRACT

Kamad silver Property is located in the Kamloops Mining Division of British Columbia and is approximately centred on 119°49'9"W longitude; 51°6'40"N latitude. This report concerns the Homestake Mine area of the property.

Four major rock types were observed in the area, these are; quartz veins, slate, chlorite schist and quartz-sericite-talc schist. The area is intensely faulted with at least three major fault sets. Economic grade ore is contained in lenticular banded barite zones. Mineralization within these zones consists of barite, sphalerite, galena, proustite (ruby silver) and tetrahedrite - in order of abundance. The complex faulting in the area has accentuated the lenticular nature of the ore zones making accurate delineation of the zones difficult. It is proposed that the 300 and 400 ore zones were originally one ore zone, with the 300 zone representing an overthrust, down-dip extension of the 400 zone. This may also be the case with the 500 zone, with the 300 and 400 zones being overthrust extensions, but the data is insufficient to determine this conclusively. Calculated reserves for the Homestake Mine area are:

measured (probable) reserves = 234,742 tonnes
maximum possible reserves = 823,550 tonnes

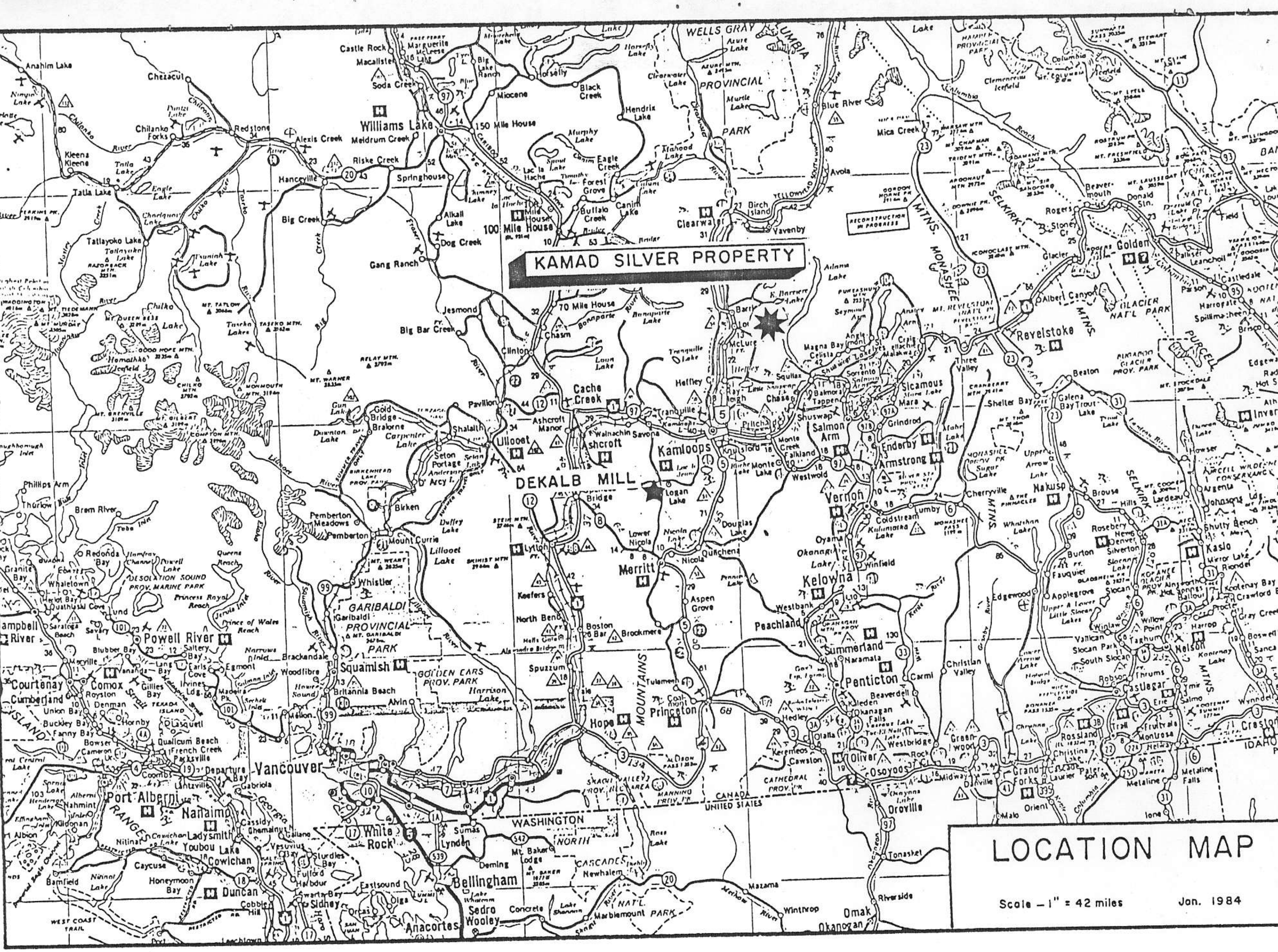
LOCATION AND ACCESS

The Kamad Property is located in the Kamloops Mining Division of British Columbia, approximately 60 kilometres northeast of the city of Kamloops. The property is centred on 119°49'9" longitude, 51°6'40" latitude and is covered by N.T.S. map sheet 82-M-4.

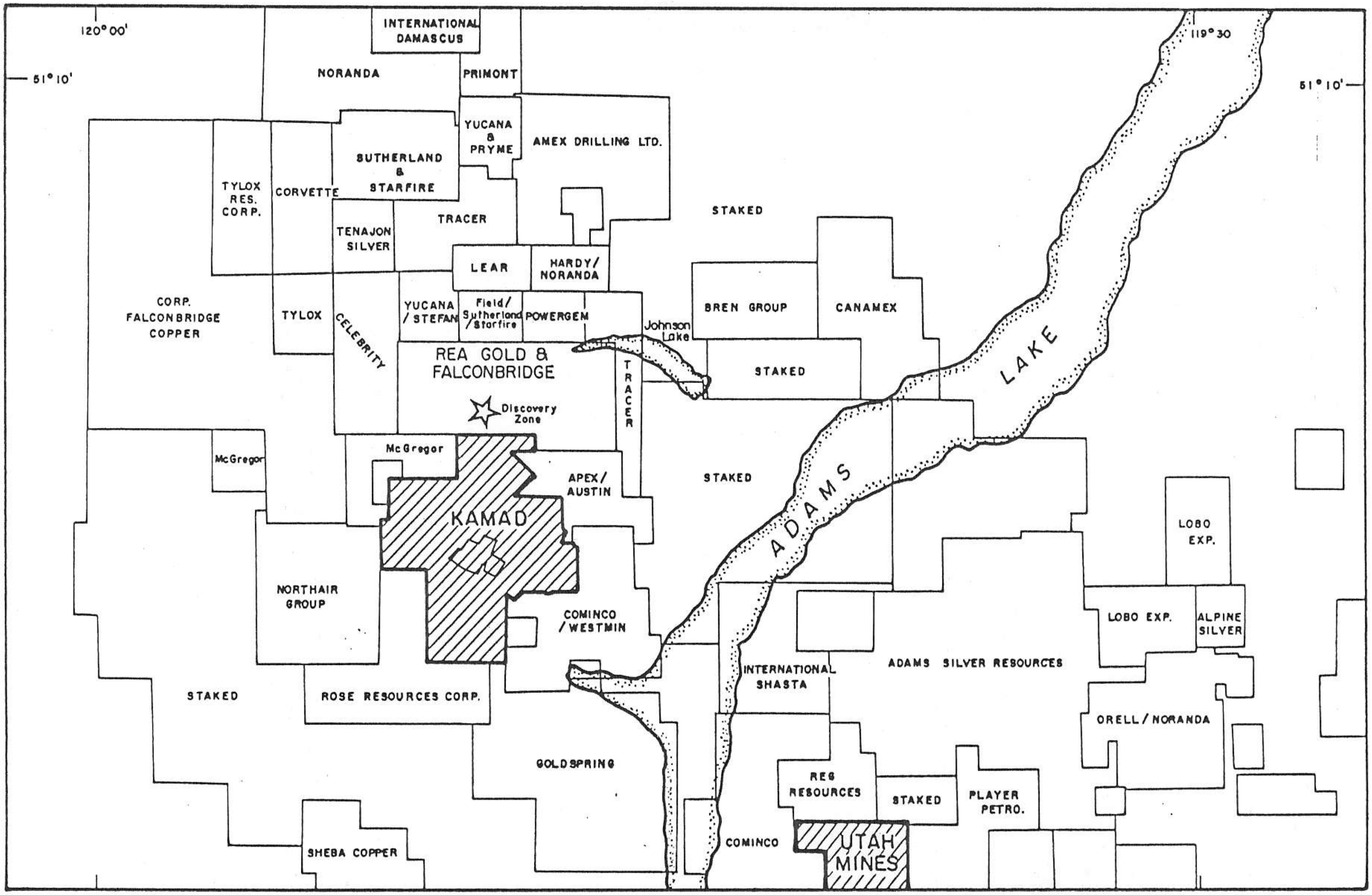
Access to the property is by Highway #5 from Kamloops to Louis Creek, then by the Agate Bay road approximately 29 kilometres east toward Squaam (Agate) Bay. Access to the main workings is provided by a rough dirt road up the north side of the Sinmax Creek valley to the portal. Drill and logging roads provide limited access to other portions of the property.

PROPERTY AND TITLE

The Kamad Property consists of seven (7) Crown Grants and 128 mineral claims. The property is controlled by Kamad Silver Co. Ltd., a junior mining company which trades on the Vancouver Stock Exchange. Kamad Silver Co. Ltd. acquired the property in the late 1960's from Allied Mining Ltd. In 1973, Kamad signed an agreement with Canadian Reserve Oil and Gas Ltd. (now owned by Getty Oil) which gives Canadian Reserve $5,000,000.00 from 50% of net cash flow after capital cost recovery. Alternately, this debt can be retired in full by 1991. The underground mine workings (Homestake Mine) are located within the area covered by the seven Crown Grants.



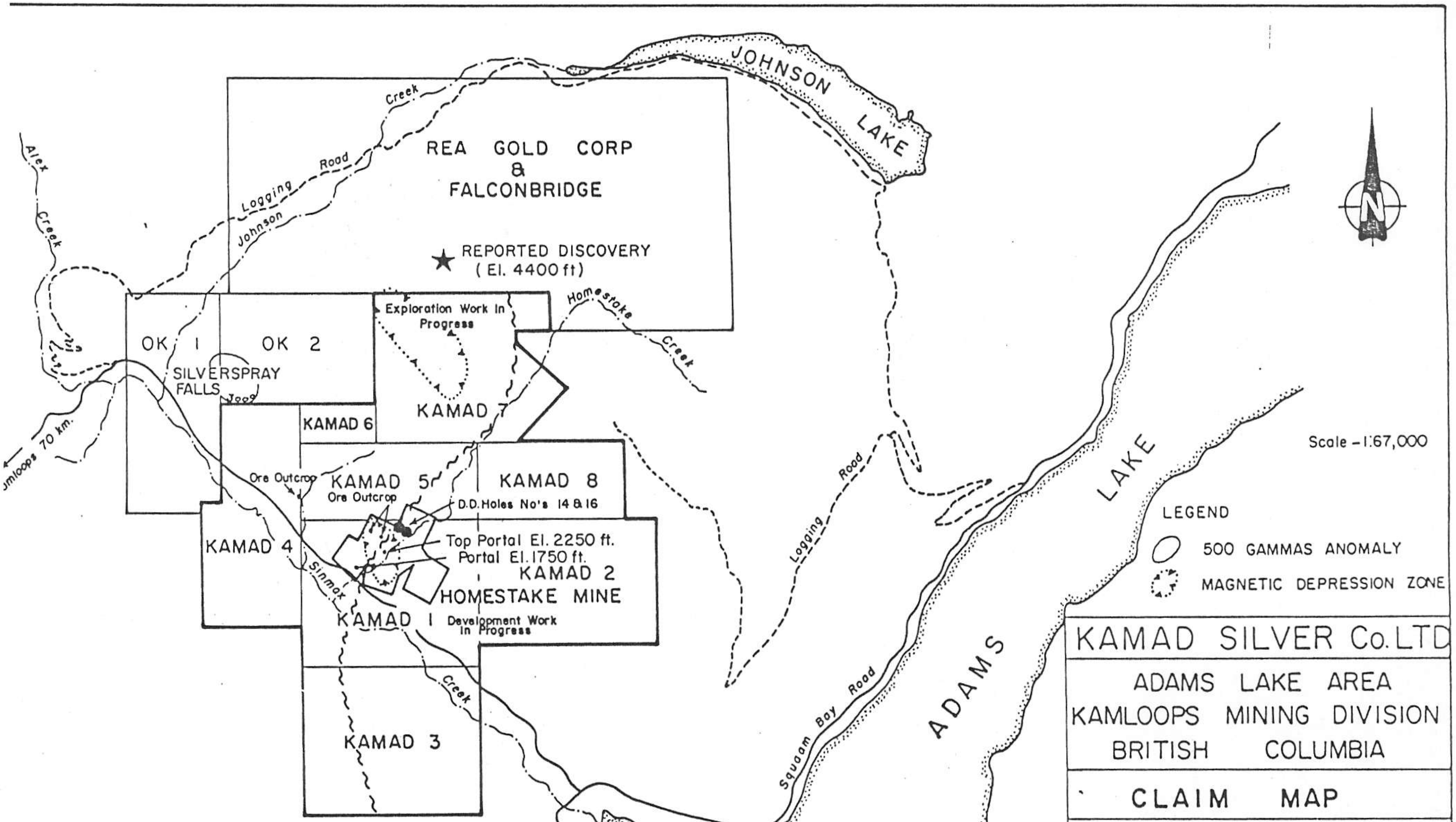
APPENDIX | BARITE MARKET | MINING | METALL-URGICAL | ENGINEERING



ADAMS LAKE AREA

CLAIM STATUS MAP

Jan. 1984



ERAGE ASSAYS FROM DIAMOND DRILL
)LES 14 & 16 AT 300 ZONE PENETRATION

)LE	PENETRATION FOOTAGE	OZ. SILVER	% GOLD	% LEAD	% ZINC	% COPPER	% BARITE
).14	13.5'	39.31	0.112	2.99	5.17	1.070	39.01
). 16	10.0'	22.87	0.238	9.99	21.43	2.239	18.16

PHYSIOGRAPHY AND CLIMATE

The Kamad Property lies within the Shuswap Highland physiographic subdivision of the Interior Plateau (Holland, 1964). The property area is typical of this subdivision, moderately sloping plateau areas dissected by creeks and rivers into large, polygonal upland areas. The property straddles the Sinmax Creek valley, the main drainage system in the area. The valley is U-shaped, with steep to vertical walls and a relatively flat bottom. Homestake Mine (Kamad's underground workings) is located on the steep, northeastern wall of the valley. A few small tributaries of Sinmax Creek (e.g. Homestake Creek) deeply incise the valley walls. The valley walls in the vicinity of the mine workings are unstable and are susceptible to slides and slumps, especially during spring runoff.

The climate of the area is semi-arid, typical of the south-central interior, with hot summers and moderate to cold winters.

REGIONAL GEOLOGY

The Kamad deposit lies within the Eagle Bay Formation, a rock unit that forms an east-west band approximately 100 km long by 20 km wide, located about 50 km northeast of Kamloops on Adams Lake.

The Eagle Bay Formation is a lithologically and structurally complex suite of intermixed proximal volcanic and volcaniclastic rocks, clastic sediments and a prominent carbonate member. Okulitch (1979) equated the Eagle Bay with the Lardeau assemblage of the Kootenay Arc and assigned a tentative Cambro-Ordovician age. On the basis of recent zircon dating of felsic rocks and micro fossils in the carbonate members, Preto (1981, B.C. Dept. of Mines) has assigned a Devonian-Mississippian age bracket to this formation.

Complex deformation and rapid facies changes within the Eagle Bay have thus far prevented definition of an internal stratigraphy. Preto outlined folding defined as early, tight to isoclinal with upright NE dipping axial planes and a general westerly vergence. These early folds are re-folded by later structures with general northerly trends. A late normal faulting event with NNE trends is also present.

In general terms, spatial distribution of lithologies comprises a predominantly volcanic central package, trending NW through the Squaam Bay-Johnson Lake area, flanked by sedimentary packages to the north and south. Within the volcanic belt, a felsic package to the south structurally underlies a more intermediate to basic package to the north. The felsic package consists of foliated rhyolite flows, sericite-chlorite schists and phyllites derived from felsic to intermediate tuffs, and locally interlayered chert, cherty tuff and impure limestone.

A distinctive limonitic unit, the Homestake Schist which is comprised of quartz sericite-pyrite schist, is associated with Pb-Zn-Ag-Ba (Au) mineralization at the Kamad property and near Skwaam Bay. A basic to intermediate volcanic package consisting of greenschists derived from massive and pillowed flows, breccias and tuffs appear to overlie the quartz-sericite schists. Inclusions of interbedded weakly calcareous phyllites produce thin banded, calc-silicate units which are associated with pyrrhotite-pyrite-chalcopyrite mineralization in the area.

A fine-grained clastic sedimentary package - now phyllite - with varying amounts of andesite appears to structurally overlie the volcanic package in the core of the major Eagle Bay Synclinarian. The coarse-grained clastic sediment package - north and south of the volcanic package - is dominated by an interlayered grit-quartzitephyllite assemblage and includes a thin but persistent limestone horizon. A second, more significant, carbonate horizon - the Tshinakin Limestone - outcrops within the central volcanic package but its stratigraphic and structural relationships to the volcanics is in doubt.

GEOLOGY

Lithology

The mine geology consists of altered and faulted schists with varying amounts of quartz, sericite, talc and chlorite. These appear to be metamorphic products of lapilli tuffs ranging from andesite to rhyodacite in composition.

Changes in the amount of each component yields rock types such as:

quartz - sericite schist
quartz - talc - sericite schist
quartz - sericite - talc - chlorite schist
chlorite schist

Other combinations of the four components were also observed, but only in minor amounts and are not considered to be important.

All of the drill hole logs were inspected and the rock types were roughly grouped into the major rock types seen during the underground mapping. In some cases this was very difficult to do because of the number of different people that logged the core.

It is unknown whether the four major components are primary constituents or alteration products, but it is likely that they are a combination of both. Some mobilization of more volatile elements such as chlorite may have occured during metamorphism.

The gradational change from one rock type to another may represent facies changes or differential metamorphism although it is most likely that facies changes are the most important variation. This gradation makes the location of contacts on section very difficult particularly when the contacts are irregular. It is not uncommon to have major facies changes over short distances in tuffs.

Following is a description of the major rock types with the section code in brackets.

a.) Quartz Veins (Unit 1)

These are found throughout the mine area and vary in number, attitude and thickness. There appear to be three phases of quartz injection; pre-tectonic, syn-tectonic and post-tectonic. Most of the quartz veining is associated with faulting, particularly the large thrust faults which will be described later in this report. Areas surrounding and along thrust fault planes generally contain both quartz veins up to 2' wide and schist which exhibits silica and chlorite alteration.

Graphitic Schist

This rock has been described in diamond drill logs but is not found underground or on surface. In some areas underground the thrust faults have developed gouge zones up to 4 feet wide which appear schistose and graphitic. On surface some slate can be observed but this has not been altered to the graphitic schist as described in the diamond drill logs. Faulting of this unit may produce a gouge that could be called a schist but, as no slate was seen underground, this is unlikely.

Slate (Unit 2)

This term is used to describe a metapelite with a high carbon content. Others have described this black, highly fractured rock as argillite, argillaceous schist or graphitic schist. The slate unit is found on surface only and represents a period of quiescence and deep marine sedimentation. The boundaries of the slate appear to be faulted with a definite thrust fault along the upper contact.

Chlorite Schist (Unit 3)

This is a highly gradational rock with chlorite contents ranging from 90% down. In cross sections and drill holes, any rocks described as containing chlorite or described as being greenish have been designated to this category.

It is probable that the chlorite is a metamorphic product from the more mafic volcanics such as andesite or dacite tuffs and the range of chlorite in schist represents an original range of tuff compositions and facies changes.

An excellent example of chlorite schist grading to a quartz-talc schist can be found in number 1 cross-cut. Here and in number 3 cross-cut, the rocks appear to be fragmental although metamorphism has destroyed most of the primary texture. Associated with the chlorite schist in this area is a silica flooding which has bleached and destroyed the chlorite. This may represent a vent area for mineralization although low metal assays were returned in samples taken from this area.

Quartz-Sericite-Talc Schist (Unit 4)

This rock unit is similar to the chlorite schist in that it is highly gradational with respect to the three components.

The quartz content of unit 4 varies from 30% to 80% and can be roughly divided into two types of schist.

i.) Thick banded schist contains 75% quartz as bands up to 2" with 1/4" schist bands between. In many places the quartz bands have been boudined into small lenses.

ii.) Thin banded quartz schist contains talc and sericite bands in greater or equal thicknesses to the quartz bands. The quartz has been boudined in some places as with the thick banded variety.

In both banded types the quartz in some areas of the mine (end of 1750 level, bottom of 2217 raise) has a light bluish cast. It is not known at this time whether this blue quartz has any spatial relationship to the ore, but this should be investigated.

In several areas underground particularly near the beginning of the "New Crosscut" the schist has small (1mm) spots of white and orange feldspars which probably represent original lapilli.

Structural Geology

The most important structures in the study area are the low angle reverse faults that are postulated from underground and surface observations. These form shear zones and gouge up to 10 feet in width and are invariably filled with quartz and scattered sulphides. The amount of movement on the thrusts is not known but if the 400 and 300 ore bodies were originally the same, then the movement may be 160 feet. There appears to be at least two of these major structures although unfortunately most of the large shear zones underground are caved and inaccessible.

There are two other main directions of faulting which form a complimentary fault set with approximately 60 degree separation. One set is roughly parallel to schistosity and the other set has an east-west strike. Movement on the stronger set of faults (parallel to schistosity) is generally hanging wall up and amount of movement varies from a few inches to several tens of feet (?).

An important topographic feature in the main area is Homestake Creek. It is likely that the creek represents a fault which has the east side downdropped. The amount of time which was spent mapping the surface did not allow for detailed mapping of the Homestake creek area. Further work should be carried out to try to determine the distance and direction of displacement on this fault as the Kamad ore horizons may be repeated to the east.

The surface geology map used for this report (Mackenzie, 1969) appears to be quite accurate although covering a small area only. Some geological mapping was carried out to obtain vertical sections on either side of the main workings. This has been plotted on the accompanying map.

ENGINEERING
METALL-
MINING
BARITE
APPENDIX

MINERALIZATION

Mineralization observed in the mine workings and in surface exposures in the mine area occurs in five different modes. These are:

1.) Banded barite with sulphides.
2.) Quartz-sericite-talc Schist with sulphide lenses, wisps and disseminations.
3.) Chlorite schist with sulphide lenses, wisps and disseminations.
4.) Silica flooding zones with sulphide lenses, wisps and disseminations.
5.) Quartz veins with sulphide lenses and disseminations primarily in selvage zones.

The sulphides observed in the mine area are (in order of abundance) pyrite, sphalerite, galena, ruby silver (proustite), tetrahedrite and chalcopyrite. Pyrite is present as disseminations and wisps throughout the rocks in varying amounts and will not be considered in discussions of sulphide mineralization except where present in unusual amounts.

The banded barite mineralization is the only type which occurs with sufficiently high assay grades, thickness and lateral extent to be considered as ore. The barite ore is distinctly banded with light grey to white bands of relatively pure barite and darker grey bands containig barite, sphalerite, galena and tetrahedrite. Ruby silver (proustite) is present as disseminations in the banded barite and has a slight tendancy to increase in abundance toward the top of the barite zones.

As observed in surface outcrop and from available underground data (drill holes and mine workings) the barite zones tend to be lenticular in shape and pinch out rapidly. An example of this is the Barite Bluff (see Map 1, Surface Geology Map) which attains a maximum thickness of approximately 35 feet, but thins to 0 feet within 100 feet of this maximum. The barite zones appear to pinch out most rapidly up and down dip. Parallel to strike the zones, while still lenticular, persist for greater distances. All the barite zones observed are bounded by low angle faults in the hanging wall and footwall. These faults are subparallel to the schistocity of the rocks and the banding in the barite zones. Quartz veins subparallel to the banding in the barite zones (observed in the "Rope Raise") are boudined, with elongation parallel to strike. The lenticular nature of the barite zones is probably primary, but has been accentuated by faulting and folding (boudinage). The boudin shape of the barite zones, very lensey parallel to dip and more elongated parallel to strike, indicates that tectonism had an effect in the overall shape of the zones.

The barite zones terminate laterally in two ways, by gradual thinning and by interfingering with adjacent rocks. The Barite Bluff (see Map 1) has examples of both types. The western portion of the bluff pinches out by gradual thinning over approximately 50 feet. The eastern portion terminates by interfingering with the adjacent rocks resulting in lenses of barite and sulphides interlayered with quartz-sericite-talc schist. The interfingered texture could be primary (ie. a facies change) or secondary (ie. boudinage and shearing along the thrust faults in nature, but is probably a combination of both. The ore observed in the "New Crosscut" (see Map 2) is very similar in texture to the eastern margin of the Barite Bluff. In this crosscut lenses of barite containing sulphides, including abundant ruby silver, are interlayered with quartz-sericite-talc schist. This may indicate that the subdrift is in the margin of a massive banded barite body.

Previous investigators of the mine area (MacKenzie, 1973; Hodgson, 1970) have postulated three distinct zones of economic banded barite ore. These zones are, in ascending order, the 500 zone, 400 zone and 300 zone. Unfortunately, most of the mine workings within the barite ore zones are inaccessible due to caving. However, those zones which were observed appear very similar in texture and composition. As mentioned previously, the barite zones are fault bounded and are also separated by large zones of thrust faulting (especially in the case of the 300 and 400 zones). This leads to some doubt as to the original number of ore zones. An examination of drill and underground workings data indicates that the 300 and 400 zones may in fact be one zone which has been overthrust upon itself (ie. stacked). This may also be the case with the 500 zone, with the 300 and 400 zones representing overthrust, down-dip extensions of the 500 zone. However, the data in this instance is less complete and not definitive. Thus, there may have originally been one barite ore zone which has, by thrust faulting, been stacked upon itself. Slight undulations along fault surfaces could cut the ore zones along strike and result in increased lateral discontinuity. This would explain the discontinuous nature of the 400 zone along strike. This model explains the striking similarities in textures and mineralogy of the three barite ore zones. The model has a significant impact on the exploration for extensions of the ore zones. The down-dip extension of barite zones will terminate abruptly in a thrust fault, but will continue up section and up dip in the hanging wall of the thrust. The overall effect is a stacking of the ore zones vertically. Also, more ore zones, as yet unlocated, could be present as lenticular pods of limited lateral extent below the 500 zone if more thrust faults are present below this zone.

Mineralization in the quartz-sericite-talc schists and the chlorite schists is similar in nature. Galena, sphalerite and tetrahedrite are present as fine grained lenses, wisps and disseminations, usually associated with pyrite and lenticular quartz bands. Mineralization in these rocks is limited in extent, with the exception of pervasive disseminated pyrite which is less than 2%. Barite was not observed in the schists and the mineralization is not of economic grade. Mineralization of this type was observed in the western portion of the mine workings (see Map 2) in the 2202 drift and the 2203 drift.

Mineralization in the silica flood zones is similar in nature to mineralization in the schists. Galena, sphalerite and tetrahedrite are present as lenses, wisps and disseminations associated with pyrite. The sulphides appear to be more coarsely crystalline than in the schists and are present in slightly greater amounts (less than 5%). No barite was observed in these zones and the mineralization is sub-economic. Mineralization of this type was observed in cross cuts #1 and #3 of the 2202 drift (see Map 2).

Mineralization in quartz veins occurs as sulphide lenses and disseminations primarily in the selvage zones. Galena sphalerite, tetrahedrite and chalcopyrite (very minor), as well as pyrite were observed in quartz veins. In most instances the sulphide content is low (less than 1%). However, in the 500 vein in the 2202 drift in the northwestern portion of the mine (see Map 2) a lense of sulphides averaging two inches in thickness was present in the selvage zone for a distance of approximately 20 feet along the vein. This occurrence is unusual, as the sulphides occurred most often as coarsely crystalline disseminations in the selvage.

Assay data, from numerous sources, for six elements - gold, silver, copper, lead, zinc and barium - are availale from the mine workings, diamond drill core and surface exposures. However, many samples were not analysed for all six elements. Samples considered in this report are chip channel samples. Grab samples are discounted due to their inherent inaccuracy. Check samples were taken by Utah from the mine workings and surface exposures and the assay results compare favourably with the previous assays. All six elements, with the possible exception of copper, have the highest grades in the banded barite zone. This is not surprising, as the barite zones are the only mineralized areas of economic grade.

No Widths recorded! ?

A correlation matrix for gold, silver, copper, lead and zinc is shown below:

	Ag	Cu	Pb	Zn
Au	.04	.03	.24	.15
Ag		.14	.43	.51
Cu			.34	.34
Pb				.80

Unfortunately the correlation with barium, which was attempted, did not function on the computer. However, an examination of assay data shows that the correlation between barium and the five other elements is good, with the exception of copper. Again, this is not surprising as the grades of gold, silver, lead and zinc increase markedly in the banded barite ore zones.

The poor correlation between copper versus gold, silver and barium is interesting as it infers a different mechanism for the emplacement of copper. In the mine workings and on the surface, most copper stain was associated with quartz veining. The higher correlation between copper, lead and zinc can be explained by the presence of galena and sphalerite in the quartz veins. The poor correlation between copper and silver indicates that ruby silver is probably the principle silver sulphide in the mine area, rather than tetrahedrite.

The lenticular shape, banding and composition of the barite ore zones, as well as the lithology of the rocks in the mine area (probably meta-volcanics), indicate that the deposit is volcanogenic. The high barite content of the ore zones, relatively low lead content, lack of a rhyolite dome and lack of massive sulphides indicate that the deposit is distal. In classic volcanogenic deposits, high barite zones are located above and/or laterally distant from the volcanic source vent. The banded barite zones were probably formed by the pooling of ore solutions in depressions in a seafloor. This model is complicated by the intense faulting in the area, which has destroyed the vertical and horizontal zonation expected with these deposits. The destruction of this zonation makes the location of the source vent, a prime exploration target, very difficult. This vent could be in the vicinity of the mine workings, but extensive drilling, geochemical surveying and geophysical surveying would be required to discover its location.

The more disseminated mineralization in the quartz-sericite-talc schist and chlorite schist is probably primary (possibly diagenetic). Remobilization and recrystallization due to metamorphism has probably aided in concentrating the sulphides into lenses and wisps. The sulphide mineralization in quartz veins and zones of silica flooding is probably secondary, being related to the silica injection. The generally more coarsely crystalline character of the sulphides tends to support this hypothesis.

ORE RESERVES

The following parameters were used in the calculation of ore reserves:

1.) $72.00 per short ton is the cut-off grade.
2.) Metals used to calculate ore are gold, silver, lead, zinc and barite, as in the report by leNobel, Deighton and Zwaan (February, 1984).
3.) Metal values used are (U.S. currency):
   - Au - $375/oz
   - Ag - $8.75/oz
   - Pb - $0.25/lb
   - Zn - $0.50/lb
   - $BaSO_4$ - $120/ton of concentrate
4.) Ore density is assumed to be 3.5 gm/cc.

The cross sections shown in Diagrams 9 through 16 in the map pocket were used in the calculation of reserves. For measured ore, the reserves are restricted to within 50 feet up and down dip of a data point (drill hole or mine workings). Ore volume was calculated by measuring the ore zone area on each cross section and projecting the zone 25 feet (one-half the distance between sections) in both directions perpendicular to the section. Ore value was calculated using the assay information made available by Kamad Silver Co. Ltd. Maximum possible ore reserves were calculaed by projecting the ore zones to their maximum possible extent on each section (within 50 feet of sub-economic surface exposure, drill intersection or mine workings intersection) parallel to the dip of the rocks.

The resulting reserves are:

Measured (probable) Reserves = 234,742 tonnes
Maximum Possible Reserves = 823,550 tonnes

It must be noted that if the barite cannot be sold (ie. used in the ore calculations) the reserves may be reduced by at least 50%.

GUIDE TO FURTHER EXPLORATION

Volcanogenic deposits are typically made up of small but high grade pods or lenses of ore. To increase the tonnage in this area a well designed exploration program must be run to obtain the maximum amount of information for the least amount of expenditure. The program which should be set up for the Kamad mine area can be separated into three parts as follows:

1.) Exploration of blank data areas throughout the underground workings.
2.) Exploration of possible down dip extensions of the ore zones from the 1750 level.
3.) Exploration for possible easterly and westerly extensions of the ore pods by surface work.

The above three areas could be explored either concurrently or separately.

Areas underground where there are blank spots should be drilled off on sections to intersect the barite horizons at a spacing of a maximum of 100 feet. Spacings between sections could also be 100 feet but 50 foot intervals is suggested. Locations for these holes are plotted on the cross sections accompanying this report (Diagrams 9 to 16).

Phase 2 of the exploration should involve drilling of the ore zone intersected by the 1982 diamond drilling. This drilling should take place from the 1750 level. Long holes (+400') drilled from the end of the level would serve to intersect the 500 ore horizon up to section 10 and would intersect possible extensions of the 300 and 400 zones down dip. These also are plotted on the cross-sections as proposed drill holes. Another fan of holes drilled perpendicular to the 1750 tunnel 100 feet back from the end of the level would give information on the up dip extension of this ore shoot.

Surface work on Phase 3 exploration should be designed to test for eastern and western extensions of the known ore horizons. This phase would include geochemistry, geology and geophysics with drilling if anomalous zones are encoutered. The geochemistry should begin with soil sample lines run along contours spaced 200 feet (slope distance) apart, with samples on each line spaced 100 feet apart. Each line should be approximately 2000 feet long and be run from either side of the workings east and west. The lines should begin along the bottom of the slope and extend to the elevation of the cliffs above the workings. Geophysics over this type of deposit would be difficult because of the high pyrite content and the numerous faults in the area. For this type of ore the best geophysical technique would be gravity but because of the steep slope on either side of the current work area, this technique would be impossible. Other types of geophysical surveys are not particularly suited to the target ore because of the low sulphide content of the ore bodies and the high sulphide content of the surrounding rock.

Before any exploratory work can be done in the mine area it is imperative that a good base map be obtained and the mine workings and roads be surveyed accurately.

As there is some indication that the main valley may represent a breached anticline some prospecting and mapping should be done on the southern part of the claims. There have been some pits dug in that area but it is not known what was found.

CONCLUSIONS AND RECOMMENDATIONS

As a result of field investigations and a re-evaluation of available data on the Homestake Mine area of the Kamad Silver Property the following conclusions have been made:

1.) Four major rock types are present in the mine area. These are:

    i.) Quartz veins
    ii.) Slate
    iii.) Chlorite schist
    iv.) Quartz-sericite-talc schist

2.) Contacts between the chlorite schists and the quartz-sericite-talc schists are irregular and gradational making accurate mapping of these zones difficult. These two rock types were probably mafic and acid volcanic tuffs, respectively, which have been metamorphosed.

3.) The area is highly faulted with at least three main fault sets and two types of faulting; block faulting and thrust faulting.

4.) Ore grade mineralization is restricted to zones of banded barite which are lenticular in shaped, fault bounded and volcanogenic in origin.

5.) The volcanic vent source for the barite zones may be present in the mine area and is a prime target for exploration.

6.) There are probably only two original ore zones in the area, with the 300 and 400 zones representing one zone which has been overthrusted upon itself. The 500 zone may also be part of this zone but the available data is insufficient to make a definite conclusion.

7.) Ore reserves are estimated to be:

    Measured (probable) = 234,742 tonnes
    Maximum possible = 823,550 tonnes

It is recommended that:

1.) A good base map with accurate topography, roads, drill hole locations, grid location and mine portal be acquired before any further exploration is undertaken.

2.) The underground workings should be remapped in detail with all rock types, quartz veins, faults and foliations shown. The geological map included in this report is accurate given the time spent in the workings, but a mine geology map should be more detailed.

3.) An effort should be made to standardize rock types when logging core and mapping the workings and surface geology. All mapping and core logging should be done carefully, completely and in detail. The complex nature of the deposit necessitates care in geological examinations as any detail may have an impact on the exploration for ore zone extensions.

4.) Three areas should be examined, using geological mapping and diamond drilling, in an effort to extend the ore reserves. These are (in order of priority):

   i.) blank data areas throughout the workings.
   ii.) possible down dip extensions of the 300 and 500 zones with drilling from the 1750 level.
   iii.) possible easterly and westerly ore extensions with mapping, geochemical sampling and drilling from the surface.

5.) Diamond drilling should be done in advance of any extension of the underground workings. This will reduce the chance of ore zone sterilization and unnecessary expenses. An experienced geologist or engineer should oversee this underground work to maximize data retrieval and direct operations.

6.) Assaying of drill core should be complete on either side of and through potential ore zones. Sampling of core to date has been sporadic at best and an effort should be made to remedy this problem.

7.) All data must be kept in a coherent package and in one location. This will make the data quickly and easily accessible for examination.

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BARITE

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212 BROOKSBANK AVE.
NORTH VANCOUVER, B.C.
CANADA V7J 2C1

TELEPHONE: (604) 984-0221
TELEX: 043-52597

• ANALYTICAL CHEMISTS • GEOCHEMISTS • REGISTERED ASSAYERS

CERTIFICATE OF ASSAY

: UTAH MINES LIMITED

1600 - 1050 W. PENDER ST.
VANCOUVER, B.C.
V6E 3S7

ATTN: PHIL BURT

CERT. # : A8410865-001-
INVOICE # : I8410865
DATE : 19-APR-84
P.O. # : NONE

Sample description	Prep code	Cu %	Pb %	Zn %	Ba NAA %	Ag FA oz/T	Au FA oz/T
34 DT 01	207	0.04	0.35	0.40	0.13	0.04	<0.003
34 DT 02	207	<0.01	0.24	0.15	0.02	0.06	<0.003
34 DT 03	207	0.03	0.24	0.32	0.10	0.05	<0.003
34 DT 04	207	<0.01	0.06	0.05	0.14	0.02	<0.003
34 DT 05	207	0.03	0.25	0.64	0.06	0.12	<0.003
34 DT 06	207	<0.01	0.04	0.03	0.03	0.06	<0.003
34 DT 07	207	0.03	0.01	0.12	0.04	0.04	0.003
34 DT 08	207	0.14	0.64	0.45	0.33	0.34	<0.003
34 DT 09	207	0.01	0.16	0.20	0.17	0.03	<0.003
4 DT 10	207	0.18	0.94	0.70	0.13	0.18	<0.003
4 DT 11	207	0.25	0.61	1.74	0.32	0.28	<0.003
4 DT 12	207	0.03	0.23	0.35	0.33	0.10	<0.003
4 DT 13	207	0.04	0.55	1.30	0.17	0.09	<0.003
4 DT 14	207	0.03	0.05	1.15	0.27	0.18	0.032
4 DT 15	207	<0.01	0.05	0.04	0.27	0.02	<0.003
4 DT 16	207	<0.01	<0.01	0.01	0.26	0.02	<0.003
4 DT 17	207	<0.01	<0.01	<0.01	0.31	0.03	<0.003
4 DT 18	207	<0.01	<0.01	0.01	0.38	0.03	0.003
4 DT 19	207	<0.01	<0.01	0.01	0.35	0.02	<0.003
4 DT 20	207	0.07	0.27	0.51	3.32	2.06	0.006
4 DT 21	207	0.03	0.15	0.28	3.18	1.22	0.005
4 DT 22	207	0.19	1.14	1.77	45.60	8.00	0.003
4 DT 23	207	0.02	0.04	0.04	1.62	0.62	0.003
4 DT 24	207	0.29	1.90	3.53	46.80	7.18	0.005
4 DT 25	207	0.33	2.05	3.36	50.10	15.72	0.005
4 DT 26	207	0.33	1.28	1.92	50.90	17.20	0.014
4 DT 27	207	0.06	0.08	0.24	0.75	0.26	<0.003
4 DT 28	207	0.53	0.36	0.28	0.21	0.80	<0.003
4 DT 29	207	0.02	0.41	0.39	1.30	0.24	<0.003
4 DT 30	207	0.20	2.42	1.81	49.80	1.40	0.003
4 DT 31	207	0.20	1.90	2.87	38.40	0.64	0.003
4 DT 32	207	0.04	1.27	2.00	26.90	0.22	<0.003
4 DT 33	207	0.05	0.22	1.49	1.78	0.16	<0.003
4 DT 34	207	<0.01	0.01	0.04	0.27	0.03	<0.003
4 DT 35	207	0.12	1.42	1.20	55.40	6.98	0.003
4 KBT 01	207	<0.01	0.08	0.41	0.23	0.02	<0.003

Anna Christie
....
Registered Assayer, Province of British Columbia

MEMBER
CANADIAN TESTING
ASSOCIATION

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CERTIFICATE OF ASSAY

: UTAH MINES LIMITED

1600 - 1050 W. PENDER ST.
VANCOUVER, B.C.
V6E 3S7

ATTN: PHIL BURT

CERT. # : A8410865-002-
INVOICE # : I8410865
DATE : 19-APR-84
P.O. # : NONE

Sample description	Prep code	Cu %	Pb %	Zn %	Ba NAA %	Ag FA oz/T	Au FA oz/T
84 KBT 02	207	<0.01	0.02	0.01	0.15	0.02	<0.003
84 KBT 03	207	<0.01	<0.01	0.01	0.19	0.01	<0.003
84 KBT 04	207	<0.01	<0.01	0.02	0.18	0.03	<0.003
84 KBT 05	207	0.03	11.60	12.00	0.07	2.86	0.003
84 KBT 06	207	<0.01	0.20	0.22	0.17	0.04	<0.003
84 KBT 07	207	0.02	0.20	0.17	0.14	0.03	<0.003
84 KBT 08	207	0.26	1.06	0.94	0.20	0.76	<0.003
84 KBT 09	207	0.21	1.05	0.94	0.17	0.46	0.003
84 KBT 10	207	<0.01	0.14	0.24	0.23	0.08	<0.003
84 KBT 11	207	0.39	0.09	0.25	0.09	0.76	<0.003
84 KBT 12	207	<0.01	0.03	0.03	0.20	0.04	<0.003
84 KBT 13	207	<0.01	0.09	0.16	0.24	0.02	<0.003
84 KBT 14	207	<0.01	0.01	0.01	0.20	0.01	<0.003
84 KBT 15	207	<0.01	0.07	0.14	0.16	0.10	<0.003
84 KBT 16	207	<0.01	<0.01	0.17	0.14	0.04	<0.003
84 KBT 17	207	0.04	0.12	0.33	0.20	0.20	<0.003
84 KBT 18	207	0.06	0.13	0.39	0.14	0.28	<0.003
84 KBT 19	207	0.01	0.23	0.55	0.11	0.22	<0.003
84 KBT 20	207	0.01	0.13	0.23	0.12	0.15	<0.003
84 KBT 21	207	<0.01	0.10	0.06	0.27	0.05	<0.003
84 KBT 22	207	<0.01	0.06	0.07	0.29	0.02	<0.003
84 KBT 23	207	0.03	0.18	0.26	0.30	0.15	<0.003
84 KBT 24	207	0.02	0.34	0.55	0.34	0.14	<0.003
84 KBT 25	207	<0.01	0.07	0.12	0.71	0.04	<0.003
84 KBT 26	207	<0.01	0.01	0.01	0.51	0.04	<0.003
84 KBT 27	207	0.17	0.71	1.39	39.00	2.00	0.006
84 KBT 28	207	0.11	0.52	0.86	18.10	4.60	0.005
84 KBT 29	207	0.18	0.87	1.46	20.70	3.24	0.005
84 KBT 30	207	0.02	0.40	0.70	3.60	1.38	<0.003
84 KBT 31	207	<0.01	0.01	0.02	0.60	0.03	<0.003
84 KBT 32	207	0.25	0.63	1.24	52.90	6.16	0.003
84 KBT 33	207	0.11	0.18	0.17	2.19	0.74	0.003
84 KBT 34	207	0.12	0.32	0.86	2.47	0.88	<0.003
84 KBT 35	207	0.10	0.05	0.09	3.00	0.44	<0.003
84 KBT 36	207	0.07	1.08	1.64	0.25	0.40	<0.003
84 KBT 37	207	1.26	0.16	0.66	0.12	0.28	<0.003
84 KBT 38	207	0.02	0.05	0.31	0.12	0.02	<0.003
84 KBT 39	207	0.05	0.41	0.51	1.13	0.98	<0.003
84 KBT 40	207	1.35	1.68	2.53	24.70	24.76	0.036

Anne Christie

Registered Assayer, Province of British Colum

MINING AND METALLURGICAL

REVIEW OF THE KAMAD SILVER CO.

HOMESTAKE PROPERTY

By: Jaap P. Zwaan (P.Eng.)
Utah Mines Ltd.
April, 1984

Introduction

On January 23 and 24, 1984, a tour was made by three representatives of Utah Mines Ltd., of the existing mine facilities on the Homestake property and the Dekalb mill. Kamad has an option to purchase this mill and it intends to modify this fully equipped facility to treat the Homestake ore.

A major shareholder of Kamad, toured the Utah group and acted as co-ordinator for information supplied to Utah subsequent to the visit.

Preliminary review of this information, indicated that a more detailed site visit was required. Two teams visited the property in late February and early March. The exploration geologists spent two weeks on the site for in depth mapping and an assessment of the potential for increased reserves. The MTS members visited the mine site for an evaluation of the existing underground workings and the collection of metallurgical rock samples.

A. Metallurgical Work

Metallurgical work done at the ICM laboratory will be included in a separate report but can be summarized as follows: Tests showed that four sulphide concentrates could be easily obtained with the following metal recoveries.

Product	Metal					
	Ag	Au	Pb	Zn	Cu	Ba
Feed Grade	12.6 oz/ton	0.016 oz/ton	1.7%	2.8%	0.40%	29.6%
	Metal Recoveries					
Copper Conc.	89.0	56.0	-	-	87.1	-
Lead "	6.0	14.0	84.5	-	-	-
Zinc "	1.7	15.0	-	78.8	-	-
Barite "	$\sim$0.6	4.0	-	-	-	75.8

Grinding Tests performed estimated a Work Index of about 7.9 KWH/ton.

B. Mining

Three mining scenarios were considered for the Homestake property. These were milling rates of (i) 300, (ii) 600 and (iii) 1000 tons per day. For the first two rates, the ore was to be hauled 100 road miles to the Dekalb mill. While the 1000 tons per day case considers a new mill at the mine site.

The extent of the reserves on the Homestake property are unknown but with the potential existing for increased reserves, higher extraction rates were considered to test economies of scale on the mining and milling costs.

C. Geological Work

The geological mapping of the Homestake property suggests that increased reserves are indicated but will be difficult and very expensive to find.

A more detailed report will be enclosed separately.

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B. MINING

(i) Milling rate 300 tons per day

Mining:

An open stope and rib pillar mining method at a mining rate of 300 tons per day is proposed. With dilution estimated at 15 percent and a mining recovery of 80 percent, the mineable reserves, as estimated by Kilbon, become 253,600 tons, grading 5.75 oz/ton of silver, 0.015 oz/ton of gold, 1.08% lead, 1.90% zinc, 0.24% copper and 31.9% barite.

Presently two adits provide access to the deposit, the 1750 and 2250 -foot level. Pre-production development will require the rehabilitation and upgrading of the 1750 level to allow for truck haulage, the rehabilitation of an existing raise for ventilation/ore pass from the 1750 level to the 2250 level, and the preparation of stopes. Mining will advance up the dip of the deposit with development of sub levels in the ore. Blasted ore will be removed by slushing down dip to a loading level where LHD-units will move it to the ore pass. Haul trucks will than transport the material from the ore pass to a stockpile outside the 1750 portal. From here the ore is hauled to the Dekalb mill for processing.

Milling:

The Dekalb mill is located 100 road miles from the mine site. This mill was originally used to produce an copper concentrate. With modifications and additional equipment this mill should be able to produce three concentrates:

(i) A copper-lead concentrate

(ii) A zinc concentrate

(iii) A barite concentrate

Most of the gold and silver should report to the first concentrate with some minor amounts to the second concentrate.

Capital Costs

An estimated $5.0 million would be sufficient to put the Homestake mine into production and modify the Dekalb mill. The major spending of this money would be as follows:

(i)	Purchase of the Dekalb mill	$1,500,000
(ii)	Mine Development and Surface Facilities	$2,500,000
(iii)	Mill Modifications	$1,000,000
	Total:	$5,000,000

page ...

Operating Costs

The total operating cost per ton milled is estimated at $72.00/ton. The costs are summarized as follows:

Description	Cost per ton milled
Mining	$ 38.50
Milling	20.00
Transportation to mill	10.00
Administration	3.50
Total Operating Cost	$ 72.00

This operating cost estimate is based mainly on operating data supplied from the Kilborn report, plus the Canadian Mining Journal's 1984 Reference Manual and Buyers Guide.

Revenue:

Net smelter return is estimated to be $110 per ton of ore. This value is based on the following metal prices.

Silver	@	US $8.25/oz
Gold	@	US $ 375/oz
Lead	@	US $0.25/lb
Zinc	@	US $0.50/lb
Ba	@	US $ 120/ton of concentrate

Further Assumptions

Investment	@ $5,000,000 -	
Milling	@ 101,000 tpy	
Costs	@ $72/ton milled	
Revenue	@ $110/ton milled	
Reserve	@ 253,000 tons	2.5 yrs mine life

Economic Evaluation

A financial model for the project using the previously listed assumptions indicated no return on investment when ignoring revenue for Barite.

With full Barite revenue included the return on investment is 24.2%.

A sensitivity on reserves indicated an increase in return on investment with more reserves.

page ...

(ii) <u>Milling rate 600 tons per day</u>

The mining and milling scenario for this option was considered to be the same as for option (i). The main differences for this option were (a) the mineable reserves were assumed to be 2.2 million tons (10 year life) with the same grades, (b) capital requirements were $10.0 million, (the increase mainly due to increased equipment requirements and additional underground development) and (c) net smelter return was estimated at $87.06. The decrease in NSR is due to lower assumed metal recoveries than used in option (i).

<u>Economic Evaluation</u>

Due to the lower smelter return and increased capital costs the project did not generate sufficient return on investment under the assumptions made.

(iii) <u>Milling rate 1000 tons per day</u>

<u>Mining</u>:

For this scenario the <u>mineable</u> reserves were assumed to be 3,650,000 tons with the same grades as (i). Due to the shallow dip of the ore zones and incompetent hanging wall the most suitable mining approach for this deposit is by the room and pillar method (Figure 1). With these conditions, dilution is expected to be about 20 percent and due to the loss of ore in non-recoverable pillars the total recovery of the ore is estimated to be 75 percent.

Initial development considered for this option is as follows:

(i) Rehabilitation of the 1750 level

(ii) The driving of a 20 percent ramp starting at the 1750

(iii) The boring of a new raise for an orepass

(iv) The development of two levels at 100 foot centers

(v) The preparation of seven stopes.

With the exception of the stopes all development is expected to be done in the hanging wall of the orebody. This will provide more stable ground with the only drawback that all the material to be removed during development is waste. Stope draw pockets are at 48 foot centers on each level (Figure 2). Broken ore will be slushed into these pockets, from which load-units will transfer it to the orepass. From here trucks will haul the ore to a crusher near the 1750 portal.

FIGURE 1: ROOM AND PILLAR MINING OF THE SHALLOW DIPPING

HOMESTAKE OREBODY

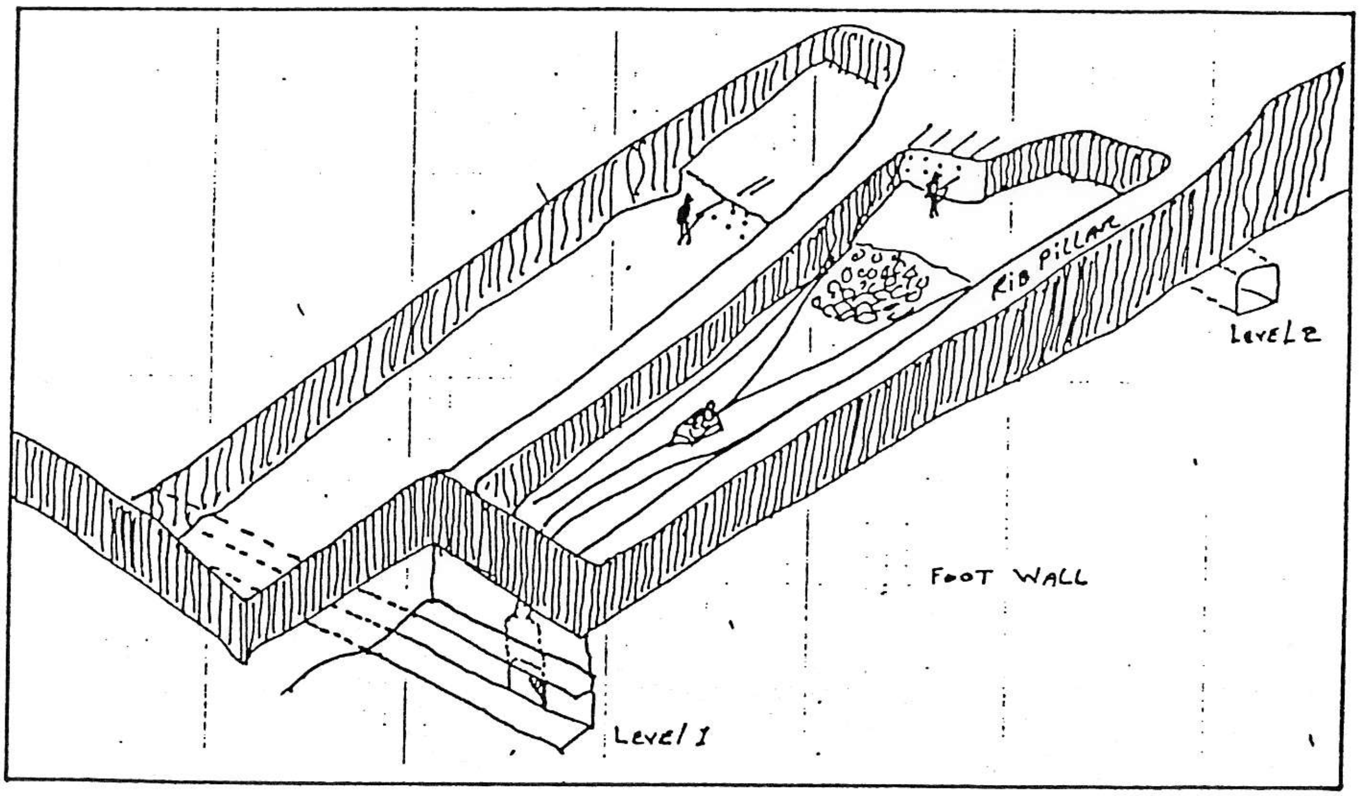
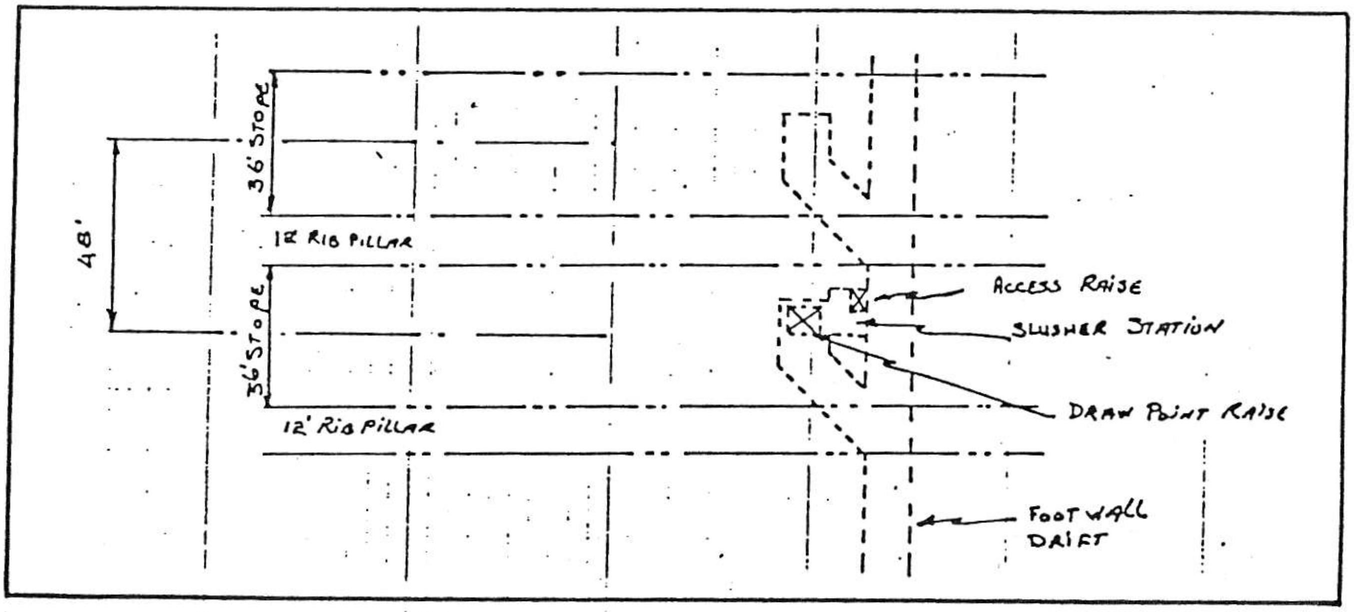


FIGURE 2: STOPE DEVELOPMENT PLAN



On-going developments during mining will involve the preparation of new stopes as the existing ones are mined out and the establishment of new ramp levels as each level is exhausted. In order to provide flexibility in mine planning, access has to be available on two levels and seven stopes at all times.

Milling:

Because of the higher milling rate of 1000 tons per day it was not considered practical to haul and process the ore in the Dekalb mill. A closed circuit mill was designed to be built at the mine site. In order to provide a better saleable product four concentrates would be produced by this mill, they are:

(i) Cu-concentrate
(ii) Pb-concentrate
(iii) Zn-concentrate
(iv) Barite-concentrate

The flow sheet of the crushing, grinding and flotation circuits is as in the attached diagrams. (Appendix A).

Capital Costs

The pre-production expenditures are estimated as follows:

- Development	\$ 3,017,000
- U/G Equipment	2,382,000
- Milling Complex	17,200,000
- Tailings Dam + Water Recycle	2,655,000
- Surface Facilities	6,737,000
- Environmental	1,500,000
- Exploration	1,434,000
Sub. Total:	\$34,925,000
- Feasibility, Design Supervision and Administration	\$ 5,239,000
Total:	\$40,164,000

Operating Costs

It is estimated that the operating costs for the project will be about $54.62 per ton milled, which is broken down as follows:

	$/Ton Milled
- Development	12.68
- Mining - Wages	10.73
- Supplies	6.16
- Milling	22.23
- G & A	2.82
	$54.62

Revenue

Net smelter return is estimated to be $74.71 (excluding Barite revenue) per ton of ore milled. This value is based on the following metal prices:

Silver	@	US	$10.00/oz
Gold	@	US	$400.00/oz
Lead	@	US	$ 0.25/1b
Zinc	@	US	$ 0.50/1b
Copper	@	US	$ 0.70/1b

Including revenue for Barite the value would be $94.06 per ton milled. (Barite @ US $0.04/1b).

Analysis:

Excluding the revenue for Barite the after tax rate of return generated for the project on its own is 7.3 percent. With the inclusion of Barite revenue, this rate increases to 20.8%. The Barite revenue was excluded from the project as it was felt that at milling rates of 1000 tons per day, the associated production of about 88,000 tons of Barite, would flood the market and would not be easily saleable. For the project to realize an acceptable rate of return an increase in metal grades of about 18 percent would have to be found to match revenue attributed to Barite sales.

METALL-URGICAL
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APPENDIX

Summary:

The 40 kg sample of silver, copper, lead, zinc, barytes ore received in the Island Copper Met Lab was dried, crushed, mixed and assayed. Individual samples were then ground and subjected to differential flotation to produce five separate products. The five products were: copper-silver concentrate, lead concentrate, zinc concentrate, bulk sulphides concentrate and barytes concentrate.

The ore assayed 0.40% Cu, 1.69% Pb, 2.80% Zn, 2.2% Fe, 12.6 oz/ton Ag, 0.016 oz/ton Au, and 50.3% $BaSO_4$. The concentrates produced are listed in Table I.

Recoveries of the metals to their specific concentrates were very good at the fine grind used, ranging between 75% and 90%. Overall silver recovery was 99.1% with 89% of the silver in the copper-silver concentrate. Upgrading was quite reasonable with only one cleaner stage required for each product.

The ore proved very easy to grind with a work index of 5.6 kWh/ton, but crushing work index was probably substantially higher due to the sticky, platey aspect of the rock.

Figure 1, following, shows a "best case" flow chart of the laboratory circuit used to produce the above five concentrates.

The reagent usages, all of which are quoted in pounds per feed ton, were not optimized due to the time constraints, therefore further lab and pilot plant work would be required to reduce both reagent and grinding costs.

BARITE

APPENDIX

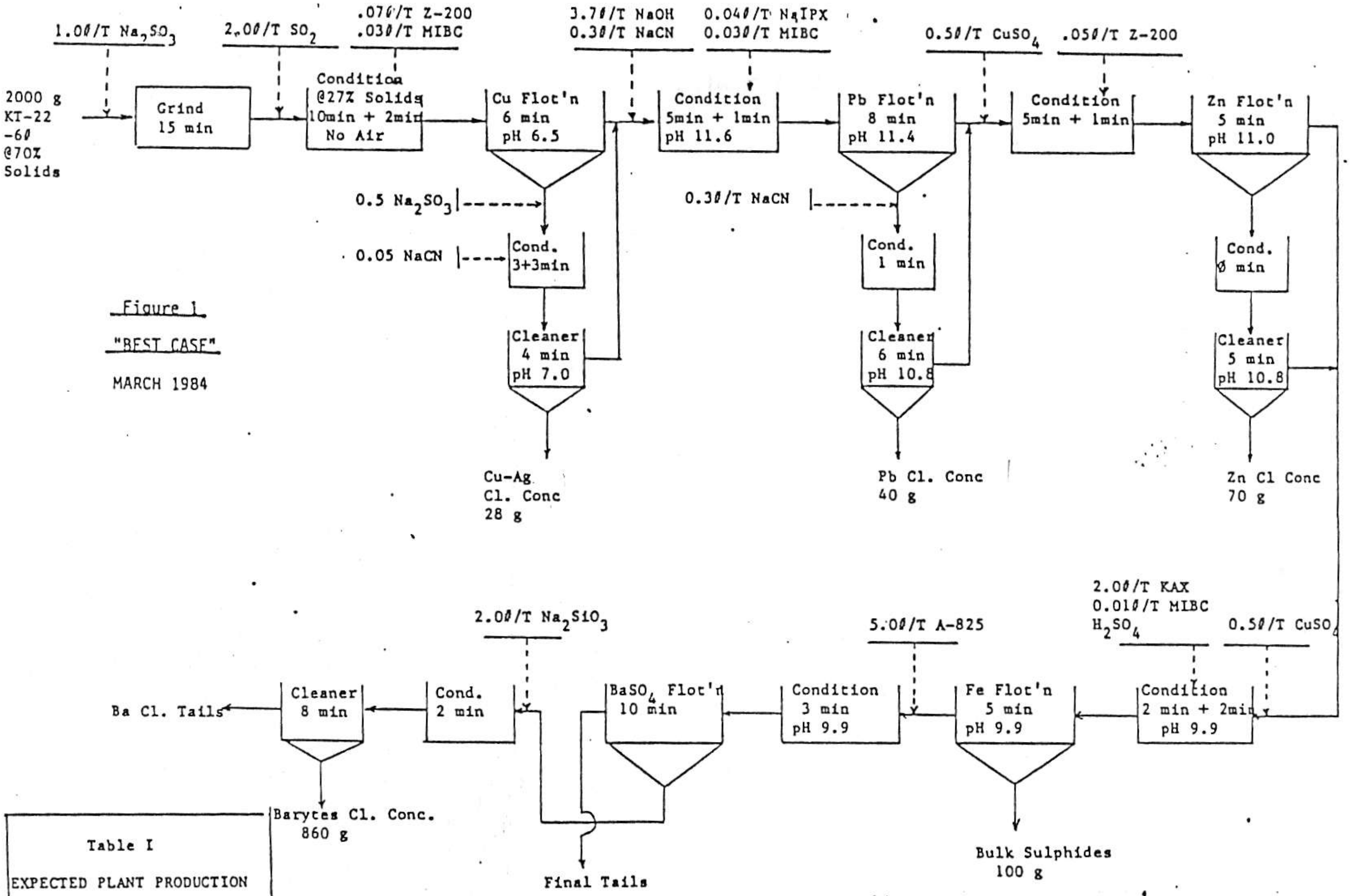


Figure 1

"BEST CASE"

MARCH 1984

Table I

EXPECTED PLANT PRODUCTION

Product	ASSAYS Ag oz/ton	Cu %	Pb %	Zn %	Au oz/ton	Fe %	Antimony* %	Sulphur* %	Barytes %	% DISTRIBUTION Ag	Cu	Pb	Zn	Au	Ba
Cu-Ag Concentrate	784	24.8	6.6	12.1	0.6	3.2	9.2	22.3	5	89.0	87.1	5.6	6.2	56	0.2
Lead Concentrate	34	0.7	69.0	9.3	0.1	2.1	0.2	18.0	ø	6.0	3.4	84.5	6.9	14	ø
Zinc Concentrate	6	0.2	1.3	58.6	0.07	2.0	0.1	31.5	2	1.7	2.0	2.8	78.8	15	0.2
Bulk Sulphides	6	0.4	1.4	0.3	0.03	23.0	0.2	28.0	7	2.4	4.2	4.5	6.6	9	1.0
Barytes Concentrate	0.2	0.01	0.03	0.03	< .001	0.6	< .01	13.1	90.5	0.6	1.6	1.1	0.3	4	75.8
										99.7	97.3	98.5	98.8	98.0	77.2

* Calculated sulphur & antimony assays

Discussion:

Sample Preparation & Grinding

Grab samples of ore from the property were received in Island Copper's Met Lab for testwork on March 2/84. The samples were combined, crushed & screened to pass 6 mesh and split into 20 charges of 2000 grams each. Prior to flotation, the samples were ground at 70% solids for 15 minutes in a $7\frac{1}{2}$ x 12 inch laboratory rod mill charged with 23.21 kg of 5/8 x $1\frac{1}{2}$" steel rods. The resulting sizing was 0.3% +100 mesh and 21.1% +325 mesh; comparative grinding with an I.C.M. standard work index ore (Wi = 15.7 kWh/t) gave sizing of about 58% +100 mesh. Calculated work index for the KT-22 ore was 5.6 kWh/ton based on Bond's equation:

$$W = 10\,Wi\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right)$$

where W = work output of mill on ore

Wi = work index of ore in kWh/ton

P = 80% passing size of product in microns

F = 80% passing size of feed in microns

Flotation

Flotation of the ground feed was performed as appended in the experimental flowcharts for each of the six tests performed. The basic method used in five of the six tests was a separate rougher flotation of each of the five individual concentrates followed by upgrading. Copper and silver (in the form of tetrahedrite) were floated first while depressing galena with sodium sulphite in the grind and sulphur dioxide in the conditioner. Galena was then reactivated by raising the pH with caustic soda and collected with sodium isopropyl xanthate while depressing sphalerite with sodium cyanide. Sphalerite was then activated with copper sulphate and recovered prior to a "bulk sulphide" flotation. Three quarters of the pyrites with most of the other remaining sulphides reported to the bulk sulphide concentrate when dosed with an excess of potassium amyl xanthate and copper sulphate. Barytes was then recovered by conditioning with Aero-promoter 825, an anionic sulphonate collector and subsequent flotation.

Upgrading of the individual concentrates involved cleaner and recleaner flotation depending upon the material and the test. As can be seen from the appended flowchart, various reagent combinations were tested to upgrade the products.

Test 4 involved combined flotation of all sulphide minerals in the bulk float, followed by barytes flotation as a test of the effects of the differential flotation reagents on barytes recovery.

BARITE

Results and Conclusions:

General

The results of the individual tests are appended and the "Best Case" results are listed on Table 2. The best case results are not the results of any given test, but are accumulated from the most reasonable concentrates of each of the five products and are considered to be expected plant results.

Crushing and Grinding

Jaw crushing proceeded very easily but subsequent cone crushing was quite difficult due to the sticky, platey nature of the ore. This would accordingly increase the crushing work index, which was not calculated at this time. Rod milling to a final product size of 0.3% plus 100 mesh and 78.9% passing 325 mesh indicated a comparative work index of 5.6 kWh/ton. This translates to 7.9 kWh/ton to reduce a feed from 80% passing 3/8 inch to 80% passing 325 mesh or about 10.6 horsepower per ton per hour.

rotopactor alternative to cone?

a) Copper-Silver Flotation

Copper and silver apparently occur simultaneously in the form of a sulphosalt of antimony (i.e. tetrahedrite) with a copper to silver ratio of about 9.6 and a copper to antimony ratio of about 2.7. This dark greyish-brown mineral was easily collected by Z-200 (a thionocarbamate) in the presence of sulphite and sulphur dioxide at a pH of 6-7 but tended to depress in the presence of excess cyanide and cyanate. Copper concentrate grades in the order of 25% Cu should be attained at 87% recovery. This final product, containing 784 oz/ton silver, was about 45-50% tetrahedrite with the balance being in order: sphalerite, gangue, galena, pyrites and barytes.

The combined addition of sulphur dioxide and sodium cyanide to the cleaners is not advisable as they tend to react as:

$$SO_2 + CN^- + O_2 + H_2O \longrightarrow CNO^- + H_2SO_4$$

This reaction substantially reduced the depressing effects until high doses were reached, at which point depression of tetrahedrite seemed to occur along with sphalerite depression. Collector then had to be added to reactivate the copper, as seen in Test #6. Future consideration should be given to the use of zinc sulphate for sphalerite depression.

b) Lead Flotation

The galena, depressed during copper flotation by SO_2 and acidified Na_2SO_3, was reactivated by increasing the pH to 11.6 and adding sodium isopropyl xanthate. Cyanide was used to depress the sphalerite and a first stage cleaner concentrate of about 69% Pb at 85% recovery was produced. The final product was about 80% galena with the balance being, in order:

BARITE

Table 2

SUMMARIZED "BEST CASE" RESULTS

Flotation Product	Weight %	Primary Metal		Silver (oz/ton)		Gold (oz/ton)		Diluents %						
		Grade	Rec'y	Grade	Rec'y	Grade	Rec'y	Cu	Pb	Zn	Sb	Fe	$BaSO_4$	Gangue
Cu-Ag Cleaner Conc.	1.4	Cu - 24.8	87.1	784.1	89.0	.62	56	-	6.6	12.1	9.2	3.2	5	14
Lead Cleaner Conc.	2.0	Pb - 69.0*	84.5	33.5	6.0	.11	14	0.65*	-	9.3*	0.24*	2.1*	Ø	0.2
Zinc Cleaner Conc	3.5	Zn - 58.6	78.8	6.1	1.7	.067	15	0.23	1.3	-	0.09	2.0	2	4.4
Bulk Sulphides Conc.	5.0	Fe - 23.0	75.0	5.8	2.4	.029	9	0.4	1.4	0.3	0.2	-	7	41
Barytes Rougher Conc.	63.4	$BaSO_4$ - 78.4	96.2	0.15	0.8	.0009	4	0.01	0.03	0.03	< 0.01	0.6	-	20
Barytes Cleaner Conc.	43	$BaSO_4$ - 90.5	75.8	N/A	~0.6	N/A	N/A	0.01	0.03	0.03	< 0.01	0.6	-	8

min $BaSO_4$ of 4.20 S.G. requires 93.5% $BaSO_4$ so requires further process cleaning HBf.

* Averaged Results

N/A - not assayed

c) Zinc Flotation

The sphalerite proved to be very silver-white in colour, indicating a high purity. After being depressed by cyanide in the lead float, it was easily reactivated by copper sulphate and collected with Z-200. The zinc upgraded to about 59% in one stage of cleaning without further reagent addition. The final product was about 88% sphalerite at a recovery of 79% with the balance being, in order: gangue, pyrites, barytes, galena and tetrahedrite and containing 6.1 oz/ton of silver.

d) The bulk sulphides, primarily a pyrite concentrate, were activated with copper sulphate and collected with very large doses of potassium amyl xanthate (i.e. 1-2 lbs/ton) at a pH of 9.9. The rougher concentrate graded 23.0% Fe or about 50% pyrites and contained 5.8 oz/ton of silver.

e) Barytes Flotation

Barytes recovery proved to be very dependant upon dosage of potassium amyl xanthate added to the bulk sulphide float. Insufficient xanthate (i.e. 1.0 lbs/ton) provided barytes recoveries of only about 60% whereas 2.0 lbs/ton increased recovery to 96% in the rougher concentrate. This high recovery was also dependant upon the preliminary flotation of separate metals concentrates and the associated reagent additions. The Aero-promoter 825 (an anionic sulphonate type) used in this testwork is probably not the optimum as evidenced by the strong dependancy upon xanthate addition. Future testwork should investigate an oxalate type collector for the barytes.

Stepanfloat apparently not tested!

// NB.

Barytes grades in the order of 90% $BaSO_4$ should be easily attained by using two pounds per ton of sodium silicate in cleaner flotation. This material would have a specific gravity of about 4.3 and an iron content of less than 1%. It may not however, be suitable as a drilling mud due to its strong hyrophobic tendancy when it is dry. This problem is probably directly due to the large xanthate additions necessary to float the barytes. If a barytes collector could be found which was independant of xanthate, then it is possible that the material would be more saleable.

Stepanfloat

The second cleaner concentrate from the second test was screen analyzed and proved to be 0.5% +200 mesh and 16.9% +325 mesh, which is too coarse for normal drilling mud and would have to be reground. — WRONG.

! A.P.I. Spec is: Max 3% +200
Min 5% +325

∴ almost perfectly meets API

BARITE

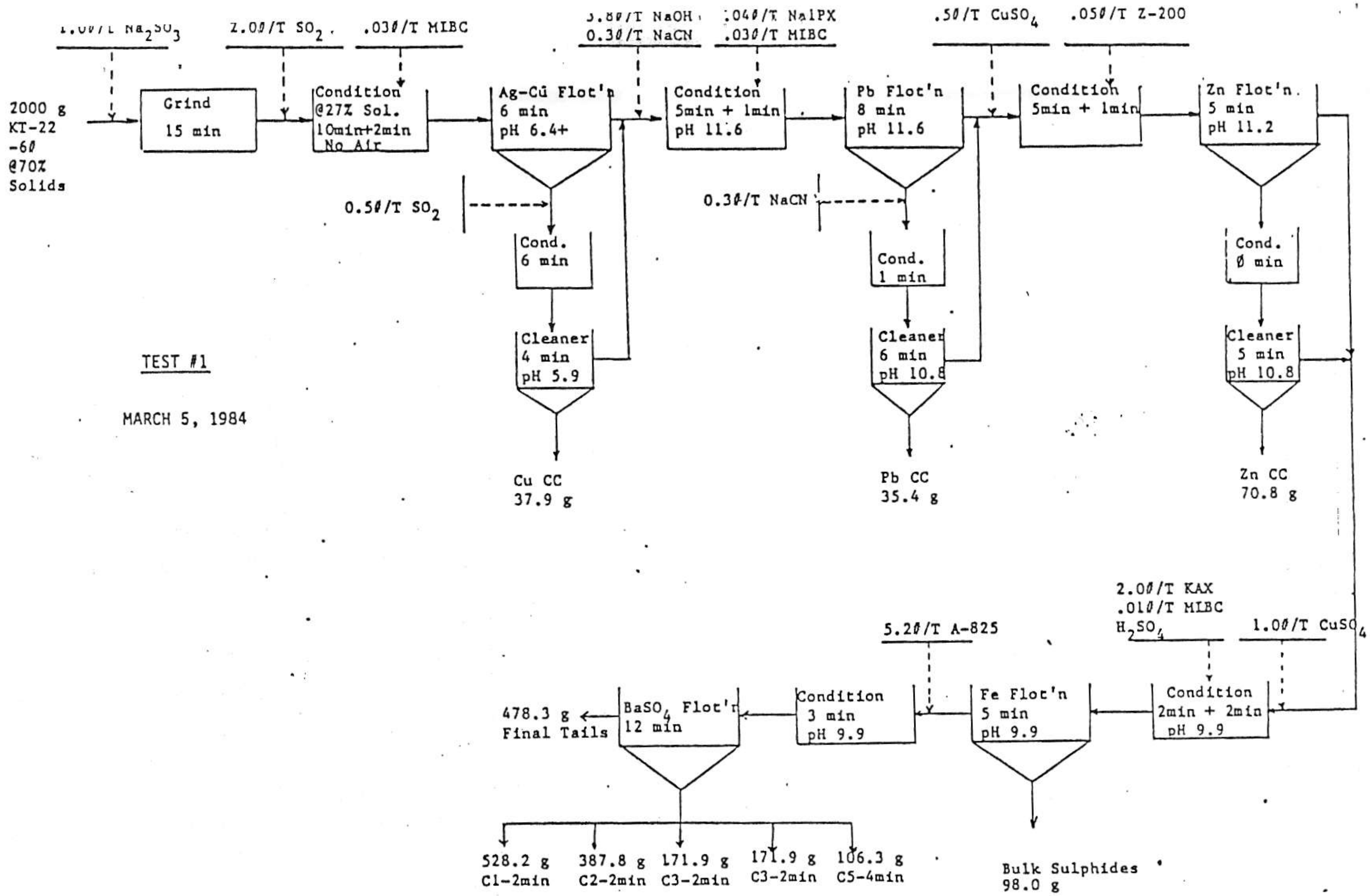
APPENDIX

MASS BALANCE

TEST #1

Product	Weight (gm)	Assay						Units					Distribution %				
		Cu%	Pb%	Zn%	Ag ppm	$BaSO_4$	Fe%	Cu	Pb	Zn	Ag	Ba	Cu	Pb	Zn	Ag	$BaSO_4$
Cu 1st Cl. Conc	37.9	18.37	6.9	21.15	19900	~5	5.9	696	261.5	801.6	754210	~190	87.1	7.9	15.2	90.3	0.2
Pb 1st Cl. Conc	35.4	.415	75.0	7.03	573	~0	1.2	14.7	2655.	248.9	20280	ø	1.8	80.6	4.7	2.4	ø
Zn 1st Cl. Conc	70.8	.335	2.41	58.6	350	~2	2.3	23.7	170.7	4145	24780	~140	3.0	5.2	78.8	3.0	0.2
Bulk Sulphides	98.0	.398	1.37	.32	282	7.3	23.1	39.0	133.8	31.4	27640	715	4.9	4.1	0.6	3.3	0.7
Barytes																	
Ro Conc 1	528.2	.007	.018	.0017	2.8	87.4	.15	3.7	9.5	0.9	1480	46170					44.7
Ro Conc 2	387.8	.008	.024	.0017	9.0	84.7	.21	3.1	9.3	0.5	3490	32850					31.8
Ro Conc 3	171.9	.013	.036	.0028	4.2	80.9	.35	2.2	6.2	0.5	310	13910					13.5
Ro Conc 4	73.9	.022	.054	.0045	4.8	60.5	.68	1.6	4.0	0.3	510	4470					4.3
Ro Conc 5	106.3	.044	.104	.0085	6.5	18.6	1.84	4.2	11.1	0.1	690	1980					1.9
Combined Ro Conc	1268.1	.012	.032	.0018	5.1	78.4	.37	14.8	40.1	2.3	6480	99380	1.8	1.2	0.1	0.8	96.2
Final Tails	478.3	.023	.069	.061	3.2	5.8	1.4	11.0	33.0	2.9	531	2774	1.4	1.0	0.6	0.2	2.7

TEST #1 Reagent	Stage Addition (lbs/Feed Ton)												
	Condition Time (min)	Grind	Cu Ro. Condi	Cu Cl. Floc'n	Cu 2nd CL Floc'n	Pb Ro. Cond.	Pb Cl. Floc'n	Pb 2nd Cl Floc'n	Zn Ro. Cond.	Bulk Sulphides	Barytes Ro. Fd.	Barytes Cl. Fd.	Barytes 2nd Cl Fd
Na_2SO_3	15	1.0	No Air										
SO_2	10, 6		2.0	0.5									
Z-200	2, 1		.07						.05				
NaCN	5, 1					0.3	0.3						
NaOH	1					3.7							
Na IPX	1					.04							
MIBC	2, 1, 2		.03			.03				.01			
$CuSO_4 . 5H_2O$	5, 2								.05	1.0			
KAX	2									2.0			
Aero-Promoter 825	3										5.2		



KT-22 DIFFERENTIAL FLOTATION

	%Cu	%Pb	%Zn	Ag ppm	%$BaSO_4$	1CC's Recovery
1CC Cu Conc.	18.4	6.9	21.0	19900		87.1 Cu, 90.3 Ag
1CC Pb Conc.	0.42	75.0	7.0	573		80.6 Pb, 2.4 Ag
1CC Zn Conc.	0.34	2.4	58.6	350		78.8 Zn, 3.0 Ag
Combined Barytes Conc.	0.01	0.03	$<$0.01	4.9	78.4	96.2 Ba, 0.8 Ag
Bulk Sulphides	0.40	1.37	0.32	282	Fe-23.0	75.0 Fe, 3.3 Ag

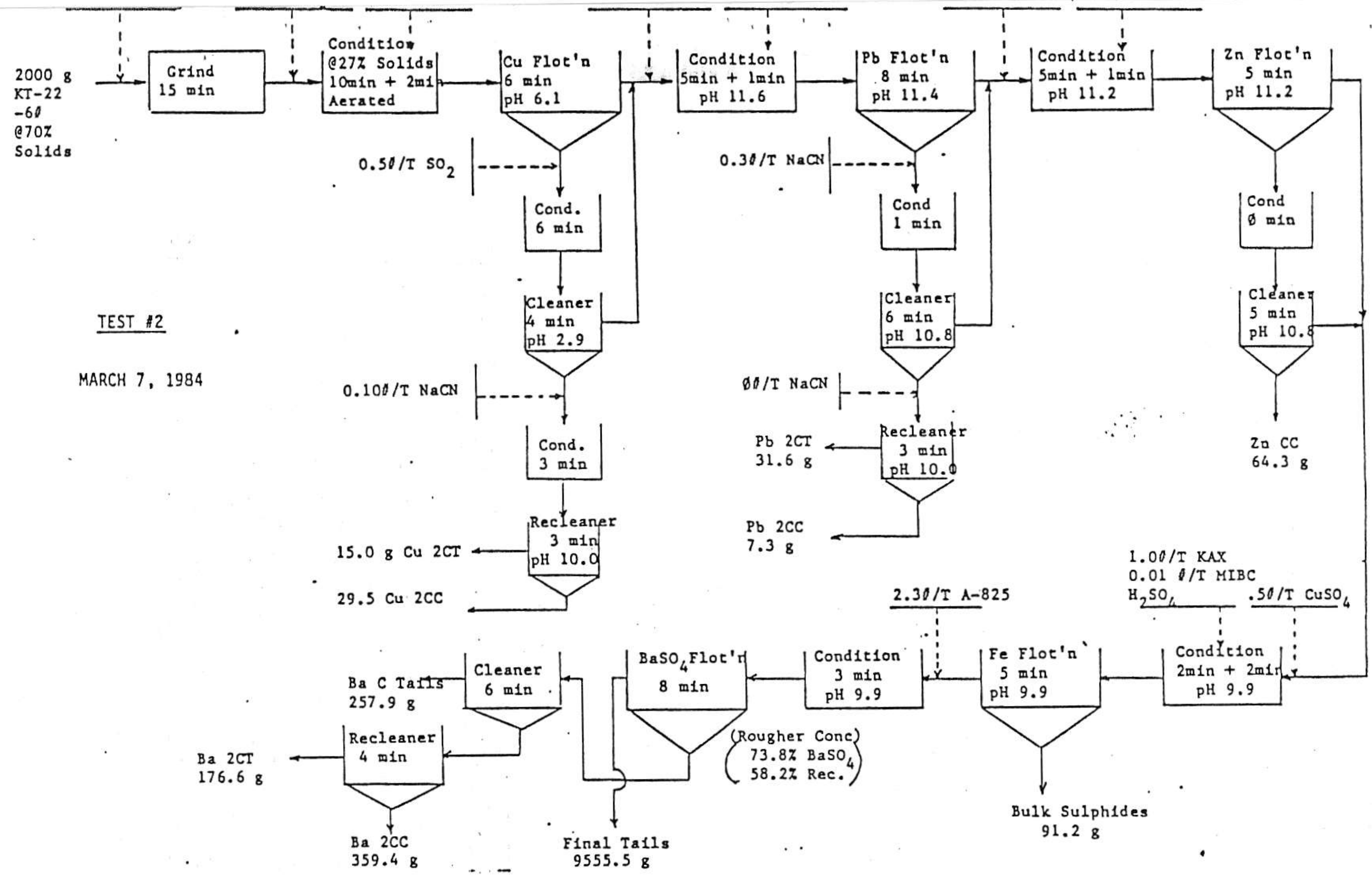
	Grade	Recovery
$BaSO_4$ -	C1-87.4	44.7
	C2-84.7	31.8
	C3-80.9	13.5
	C4-60.5	4.3
	C5-18.6	1.9

MILL DISCHARGE SIZE

Mesh	% Ret.	% Pass
65	0.1	99.9
100	0.3	99.7
150	0.7	99.3
200	3.3	96.7
325	21.1	78.9

Product	Weight (gm)	Assay Cu%	Pb%	Zn%	Ag ppm	$BaSO_4$	Fe%	Units Cu	Pb	Zn	Ag	Ba	Distribution % Cu	Pb	Zn	Ag	$BaSO_4$
Cu 2nd Cl. Conc	29.5	21.05	16.0	16.2	20600	N/A	2.95	621	472	478	607700	N/A	87.1	15.1	8.8	88.0	N/A
Cu 2nd Cl. Tail	15.0	.69	4.7	24.3	860		14.6	10.4	70.5	365	12900		1.5	2.3	6.7	1.8	
"Cu Cl. Conc"	44.5	14.2	12.2	18.9	13950	~5	6.9	631	543	843	120600	~220	88.6	17.4	15.5	89.8	~0.2
Pb 2nd Cl. Conc	31.6	.355	69.2	10.1	530	N/A	2.6	11.2	2159	319	16900	N/A	1.6	69.2	5.9	2.4	
Pb 2nd Cl. Tail	7.3	1.70	21.3	14.5	2045		1.85	12.4	156	106	14900		1.7	5.1	2.0	2.2	
"Pb Cl. Conc"	38.9	.61	59.5	10.9	817	~0	2.5	23.6	2315	425	31800	ø	3.3	74.3	7.9	4.6	ø
Zn Cl. Conc	64.3	.14	1.15	59.6	190	~2	2.05	9.0	73.9	3832	12200	~120	1.3	2.4	70.8	1.8	~0.1
Bulk Sulphides	91.2	.27	1.48	2.88	195	~7	21.2	24.6	135	263	17800	~640	3.4	4.3	4.9	2.6	~0.6
Barytes																	
2nd Cl. Conc	359.4	N/A	N/A	N/A	N/A	92.0	N/A										
1st Cl. Tail	257.9					70.9											
2nd Cl. Tail	176.6					76.0											
"Barytes Ro. Conc"	793.9					81.6		~15	~40	~3	~6500	64770	2.1	1.3	0.1	0.9	~61.2
Final Tails	955.5	.01	.01	.045	2	42	.80	9.6	9.6	43.0	1911	40130	1.3	0.3	0.8	0.3	~37.9

TEST #2 Reagent	Stage Addition (lbs/Feed Ton) Condition Time (min)	Grind	Cu Ro. Condi	Cu Cl. Flot'n	Cu 2nd CL Flot'n	Pb Ro. Cond.	Pb Cl. Flot'n	Pb 2nd Cl Flot'n	Zn Ro. Cond.	Bulk Sulphides	Barytes Ro. Fd.	Barytes Cl. Fd.	Barytes 2nd Cl Fd
Na_2SO_3	15	1.0	Air										
SO_2	10, 6		2.0	0.5									
Z-200	2, 1		0.07						0.05				
NaCN	3, 5, 1				0.1	0.3	0.3	ø					
NaOH	1					3.7							
Na IPX	1					0.06							
MIBC	2, 1, .2		.03			0.03				0.01			
$CuSO_4.5H_2O$	5, 2								0.5	0.5			
KAX	2									1.0			
Aero-Promoter 825	3										2.3		
Quebracho	2												1.0
Starch	2												



KT-22 DIFFERENTIAL FLOTATION

	%Cu	%Pb	%Zn	Ag ppm	$\%BaSO_4$	1CC's Recovery
2CC Cu Conc.	21.1	16.0	16.2	20600		88.6 Cu, 89.8 Ag
2CC Pb Conc.	0.36	69.2	10.1	530		74.3 Pb, 4.6 Ag
1CC Zn Conc.	0.14	1.2	59.6	190		70.8 Zn, 4.6 Ag
2CC Barytes Conc.					83.3	61.2 Ba
Bulk Sulphides	0.27	1.5	2.9	195	Fe-21.2	64.4 Fe, 2.6 Ag

MILL DISCHARGE SIZE

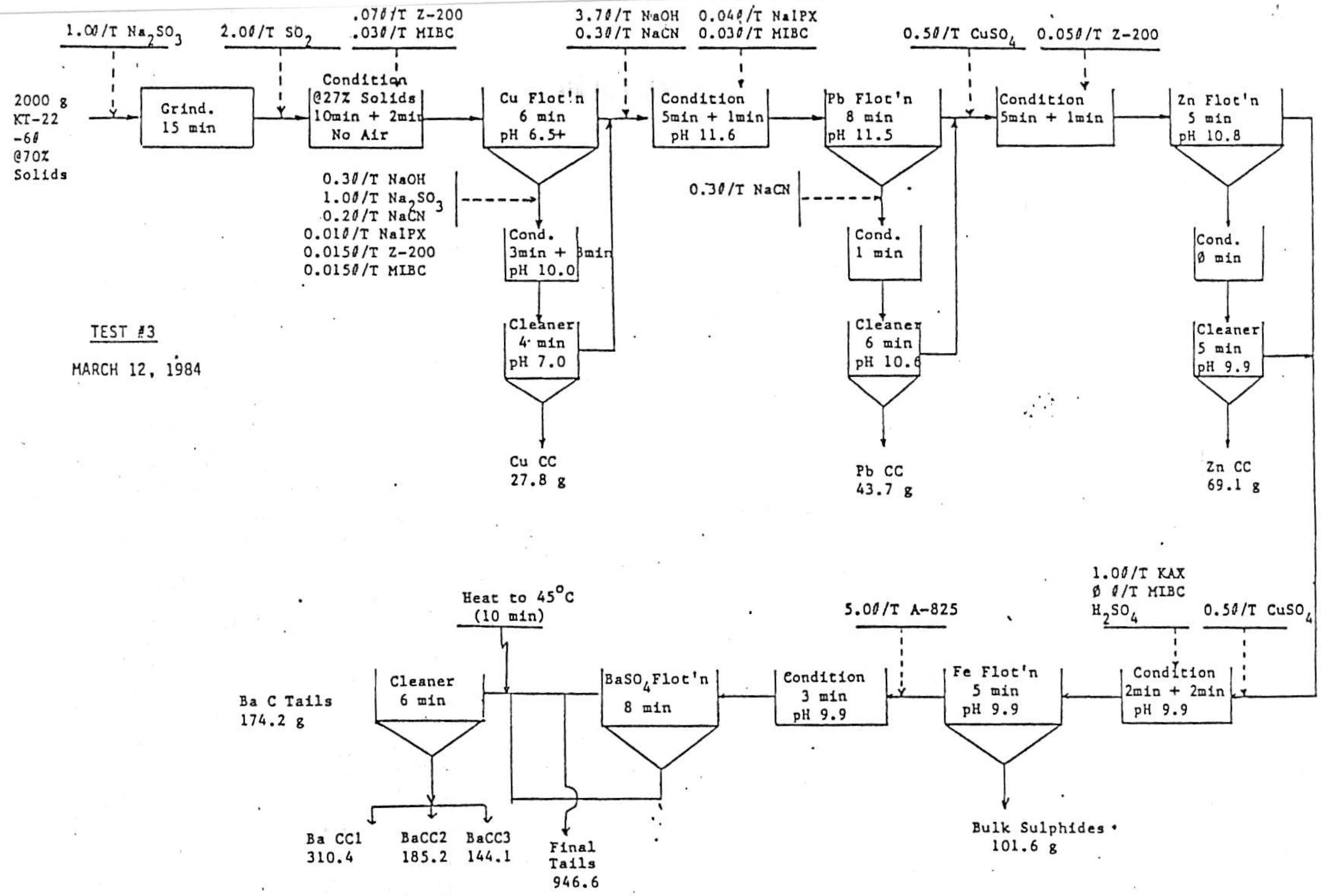
Mesh	% Ret.	% Pass
65	0.1	99.9
100	0.3	99.7
150	0.7	99.3
200	3.3	96.7
325	21.1	78.9

TEST #3

MASS BALANCE

Product	Weight (gm)	Assay						Units					Distribution %				
		Cu%	Pb%	Zn%	Ag ppm	$BaSO_4$	Fe%	Cu	Pb	Zn	Ag	Ba	Cu	Pb	Zn	Ag	$BaSO_4$
Cu 1st Cl. Conc	27.8	24.8	6.6	12.1	26900	~5	3.2	689	183	336	747820	150	87.1	5.6	6.2	89.0	0.2
Pb 1st Cl. Conc	43.7	.93	62.9	10.1	1150	~0	2.7	40.6	2749	441	50260	Ø	5.1	84.5	8.1	6.0	Ø
Zn 1st Cl. Conc	69.1	.23	1.3	58.5	210	~2	1.9	15.9	89.8	4042	14510	140	2.0	2.8	74.5	1.7	0.2
Bulk Sulphides	101.6	.28	1.6	5.3	200	~10	26.0	28.4	162.6	538	20320	1020	3.6	5.0	9.9	2.4	1.0
Barytes																	
Cl. Conc 1	310.4	.01	.02	.02	13.5	87.2	.57					27070					24.8
Cl. Conc 2	185.2	.01	.02	.02	2.0	89.3	.52					16540					15.1
Cl. Conc 3	144.1	.01	.04	.04	2.4	84.5	.58					12180					11.1
Barytes Cl.Tails	174.2	.01	.03	.03	2.0	59.5	.61					10360					9.5
"Barytes Ro. Conc"	813.9	.01	.03	.03	6.5	81.3	.57	8.1	24.4	24.4	5260	66150	1.0	0.8	0.4	0.6	60.5
Final Tails	946.6	.01	.045	.045	2.5	44.2	1.10	9.5	42.6	42.6	2370	41840	1.2	1.3	0.8	0.3	38.1

TEST #3 Reagent	Stage Addition (lbs/Feed Ton)												
	Condition Time (min)	Grind	Cu Ro. Condi	Cu Cl. Floc'n	Cu 2nd CL Floc'n	Pb Ro. Cond.	Pb Cl. Floc'n	Pb 2nd Cl Floc'n	Zn Ro. Cond.	Bulk Sulphides	Barytes Ro. Fd.	Barytes Cl. Fd.	Barytes 2nd Cl Fd
Na_2SO_3	15	1.0	No Air	1.0									
SO_2	10		2.0										
Z-200	2, 1, 1		.07	.015					0.05				
NaCN	3, 5, 1			0.2		0.3	0.3						
NaOH	3, 1			0.3		3.7							
Na IPX	1, 1			0.01		0.04							
MIBC	2, 1, 1		0.03	0.015		0.03							
$CuSO_4.5\ H_2O$	5, 2								0.5	0.5			
KAX	2									1.0			
Aero-Promoter 825	3										5.0		
Heat	10											45°C	



KT-22 DIFFERENTIAL FLOTATION

	%Cu	%Pb	%Zn	Ag ppm	$\%BaSO_4$	1CC's Recovery
Cu Conc.	24.8	6.6	12.1	269000		87.1 Cu, 89.0 Ag
Pb Conc.	0.93	62.9	10.1	1150		84.5 Pb, 6.0 Ag
Zn Conc.	0.23	1.3	58.5	210		74.5 Zn, 1.7 Ag
Barytes Conc.					81.3	60.5 $BaSO_4$
Bulk Sulphides	0.28	1.6	5.3	200	Fe-26.0	59.0 Fe, 2.4 Ag

MASS BALANCE

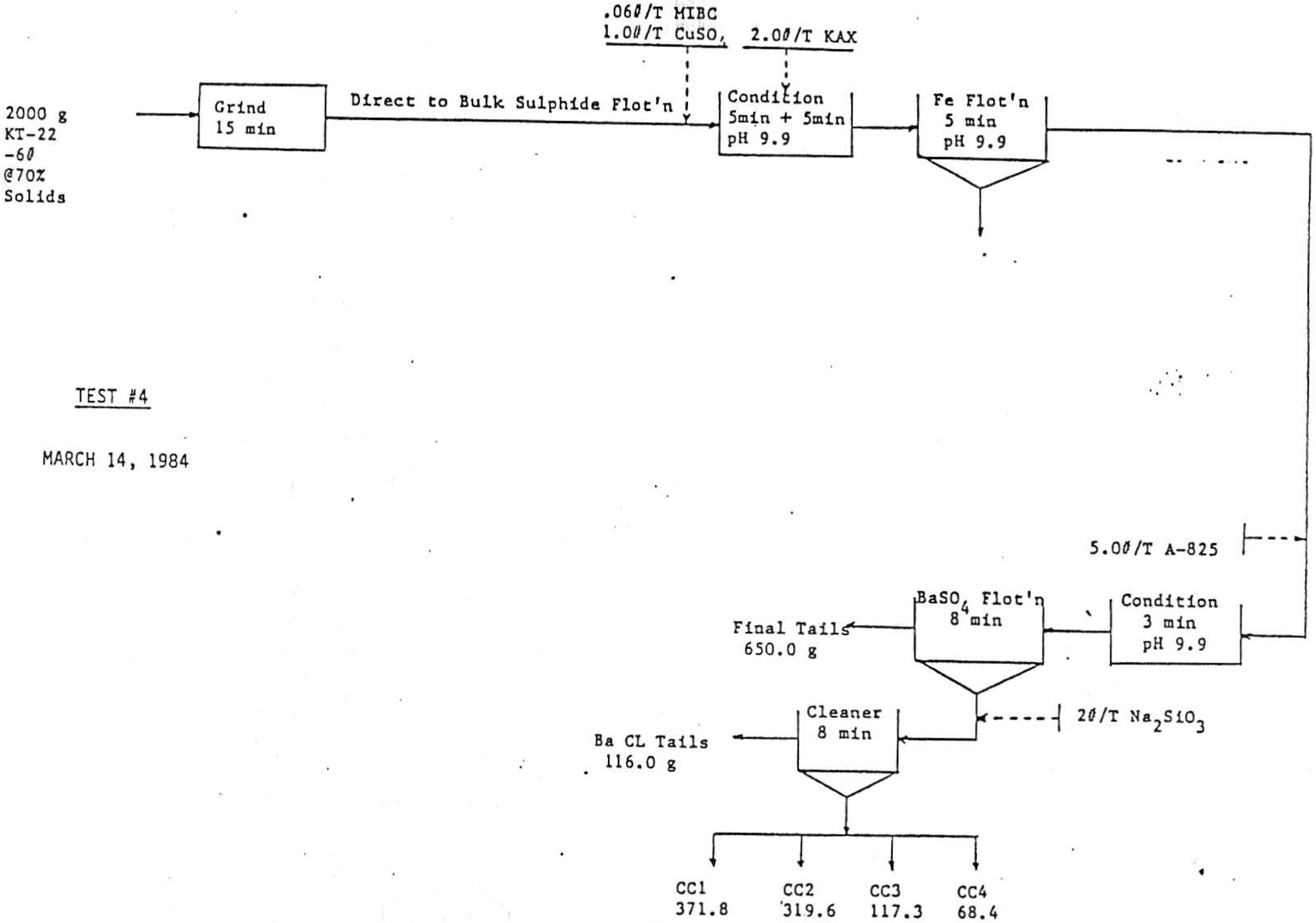
TEST #4

Product	Weight (gm)	Assay						Units					Distribution %				
		Cu%	Pb%	Zn%	Ag ppm	$BaSO_4$	Fe%	Cu	Pb	Zn	Ag	Ba	Cu	Pb	Zn	Ag	$BaSO_4$
Cu Conc	Ø																
Pb Conc	Ø																
Zn Conc	Ø																
Bulk Sulphides	354.9					6.3						2236					2.1
Barytes																	
Cl. Conc 1	371.8					91.0						33830					32.3
Cl. Conc 2	319.6					92.9						29690					28.4
Cl. Conc 3	117.3					89.3						10470					10.0
Cl. Conc 4	68.4					78.2						5349					5.1
Cl. Tails	116.0					16.2						1879					1.8
"Barytes Ro Conc"	993.1					81.8						81220					77.6
Final Tails	650.0					32.5											20.2

TEST #4 Reagent	Stage Addition (lbs/Feed Ton)												
	Condition Time (min)	Grind	Cu Ro. Condi	Cu Cl. Floc'n	Cu 2nd CL Floc'n	Pb Ro. Cond.	Pb Cl. Floc'n	Pb 2nd Cl Floc'n	Zn Ro. Cond.	Bulk Sulphides	Barytes Ro. Fd.	Barytes Cl. Fd.	Barytes 2nd Cl Fd
MIBC	5									0.06			
$CuSO_4 \cdot 5H_2O$	5									1.0			
KAX	5									2.0			
Aero-Promoter 825	3										5.0		
Sodium Silicate	2											2.0	

TEST #4

MARCH 14, 1984



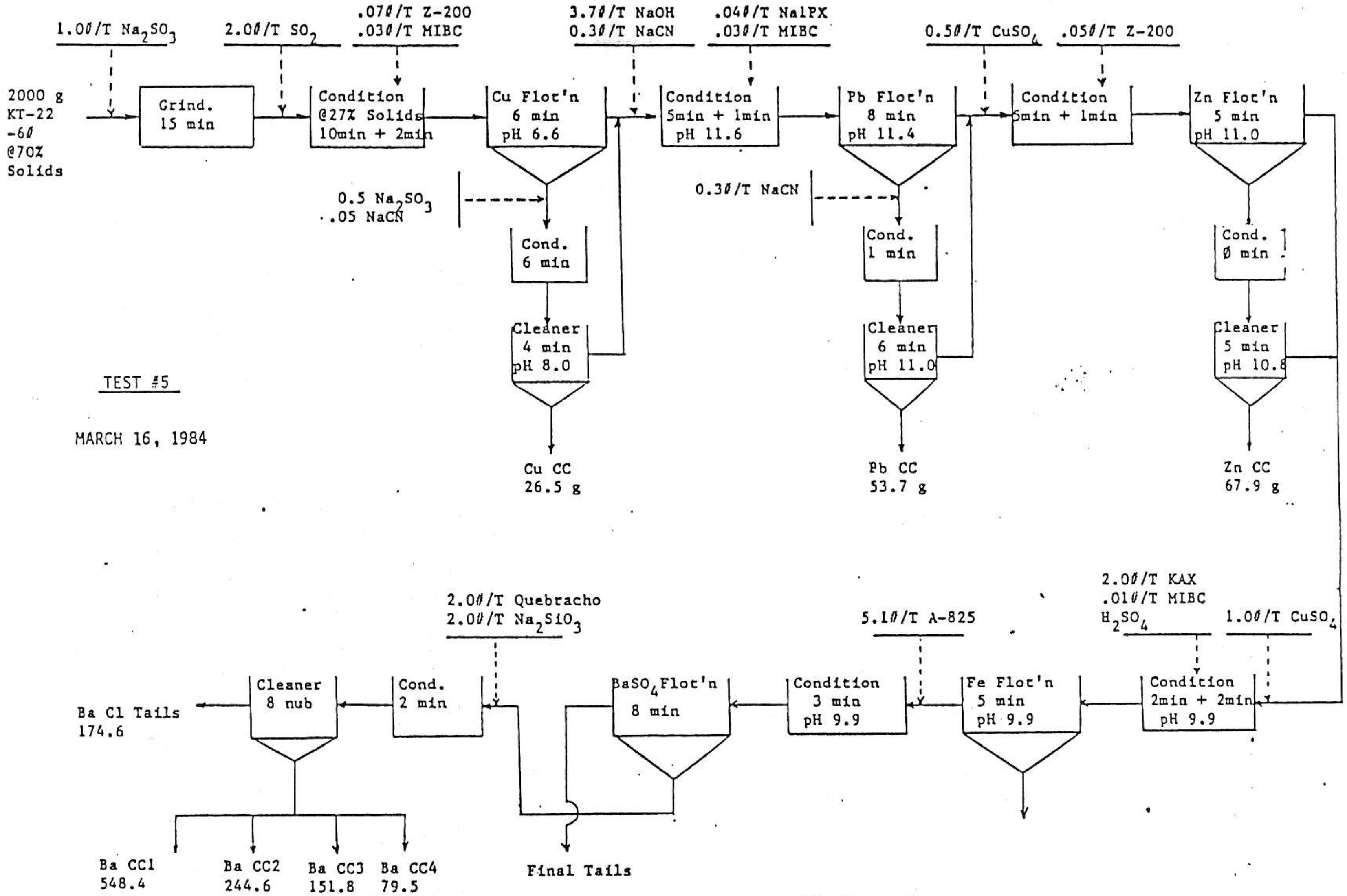
KT-22 DIFFERENTIAL FLOTATION

	%Cu	%Pb	%Zn	Ag ppm	$\%BaSO_4$	Recovery %
Cu Conc.						
Pb Conc..						
Zn Conc.						
Barytes Ro Conc.					81.8	Ba-77.6
Bulk Sulphides						

TEST #5

Product	Weight (gm)	Assay Cu%	Assay Pb%	Assay Zn%	Assay Ag ppm	Assay $BaSO_4$	Assay Fe%	Units Cu	Units Pb	Units Zn	Units Ag	Units Ba	Distribution % Cu	Distribution % Pb	Distribution % Zn	Distribution % Ag	Distribution % $BaSO_4$
Cu Conc	26.5	25.85	4.85	11.15	29000	~4	3.7	685	129	295	768500	106	82.9	3.7	5.4	84.7	0.1
Pb Conc	53.7	0.90	55.85	8.9	1215	~1	2.3	48	2999	478	65246	54	5.8	87.0	8.7	7.2	0.1
Zn Conc	67.9	0.52	1.45	59.5	640	~2	0.56	35	98	4040	43456	136	4.2	2.8	73.7	4.8	0.1
Bulk Sulphides	127.8	0.32	1.25	4.70	184	11.8	24.5	41	160	601	23515	1508	5.0	4.6	11.0	2.6	1.5
Barytes																	
Cl. Conc 1	548.4					90.1											82.3
Cl. Conc 2	244.6					86.2											
Cl. Conc 3	151.8					83.3											
Cl. Conc 4	79.5					10.2											
Cl. Tails	174.6					22.1											3.8
Barytes Ro. Conc.	1198.9	0.01	0.03	0.03	3.5	73.3	0.6	12	36	36	4200	87879	1.5	1.0	0.7	0.5	86.1
Final Tails	508.5	0.01	0.054	0.061	4.1	24.9	1.25	5	27	31	2084	12662	0.6	0.8	0.6	0.2	12.4

TEST #5 Reagent	Condition Time (min)	Grind	Cu Ro. Condi	Cu Cl. Flot'n	Cu 2nd Cl Flot'n	Pb Ro. Cond.	Pb Cl. Flot'n	Pb 2nd Cl Flot'n	Zn Ro. Cond.	Bulk Sulphides	Barytes Ro. Fd.	Barytes Cl. Fd.	Barytes 2nd Cl Fd
	Stage Addition (lbs/Feed Ton)												
Na_2SO_3	15, 3	1.0	No Air	0.5									
SO_2	10		2.0										
Z-200	2, 1		.07						0.05				
NaCN	3, 5, 1			0.05			0.3	0.3					
NaOH	1					3.7							
Na IPX	1					0.04							
MIBC	2, 1, 2		0.03			0.03				0.01			
$CuSO_4.5H_2O$	5, 2								0.5	1.0			
KAX	2									2.0			
Aero-Promoter 825	3										5.1		
Quebracho	2											2.0	
Sodium Silicate	2											2.0	



KT-22 DIFFERENTIAL FLOTATION

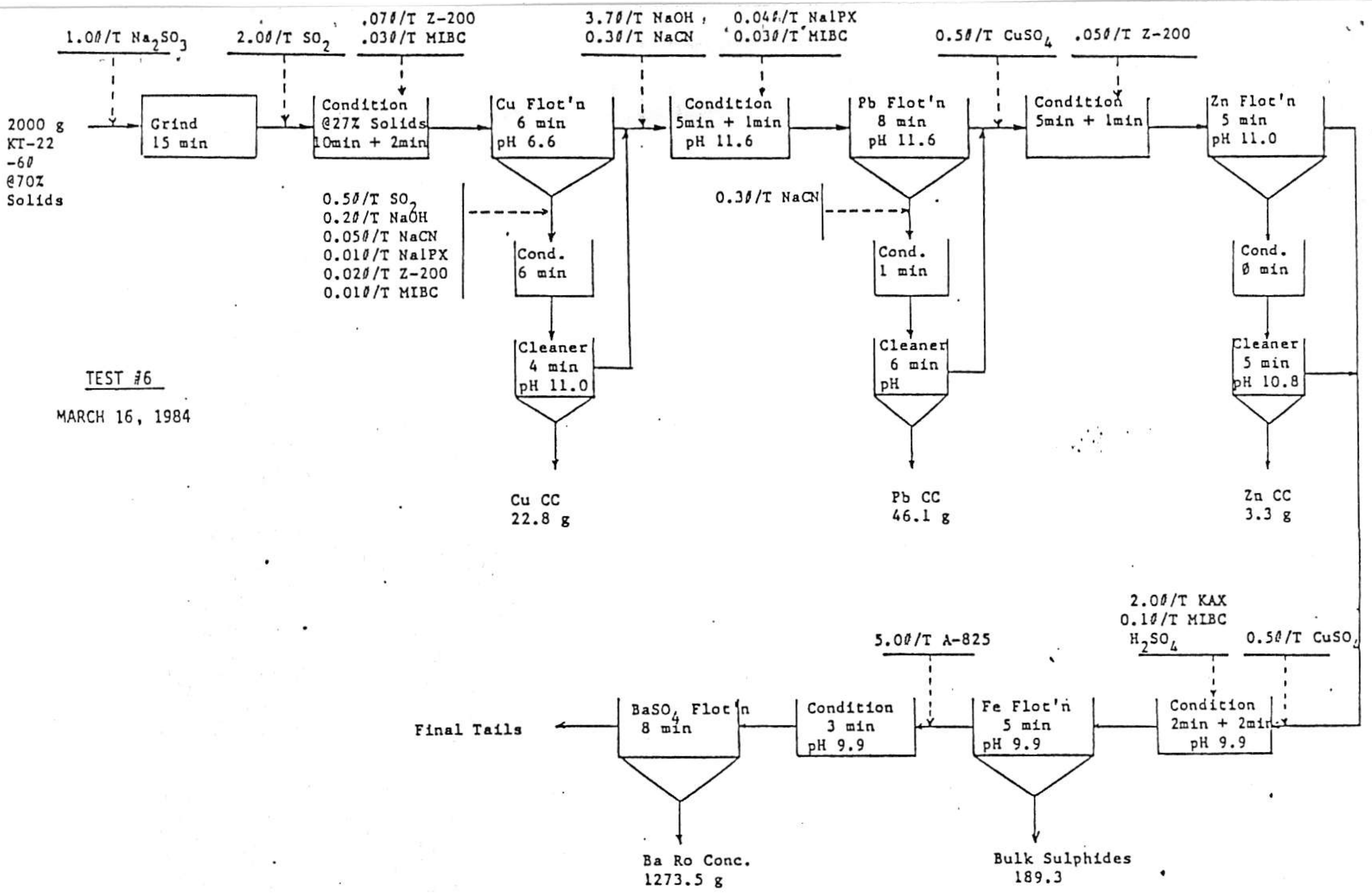
	%Cu	%Pb	%Zn	Ag ppm	%$BaSO_4$	Recovery
Cu Conc.	25.85	4.85	11.15	29000		82.9 Cu, 84.7 Ag
Pb Conc.	0.90	55.85	8.9	1215		87.0 Pb 7.2 Ag
Zn Conc.	0.52	1.45	59.5	640		73.7 Zn, 4.8 Ag
Barytes Ro. Conc.					73.3	86.1 $BaSO_4$
Bulk Sulphides	0.32	1.25	4.7	184	Fe-24.5	66.0 Fe, 2.6 Ag

MASS BALANCE

TEST #6

Product	Weight (gm)	Assay Cu%	Assay Pb%	Assay Zn%	Assay Ag ppm	Assay $BaSO_4$	Assay Fe%	Units Cu	Units Pb	Units Zn	Units Ag	Units Ba	Distribution % Cu	Distribution % Pb	Distribution % Zn	Distribution % Ag	Distribution % $BaSO_4$
Cu Conc	22.8	17.65	9.95	8.05	20800	~5	7.0	402	227	184	474240	114	50.3	6.7	3.3	52.0	0.1
Pb Conc	46.1	1.60	63.2	9.75	2085	~1	2.8	74	2914	449	96119	46	9.3	86.1	8.2	10.5	0
Zn Conc	3.3	N/A	N/A	N/A	N/A	N/A	N/A	-	-	-	-	-	-	-	-	-	-
Bulk Sulphides	189.3	1.61	1.05	25.5	1770	3.4	13.9	305	199	4827	335060	644	38.2	5.9	87.6	36.8	0.6
Barytes Ro Conc	1273.5	0.01	0.03	0.03	3.5	79.3	4.0	13	38	38	4457	100989	1.6	1.1	0.7	0.5	94.5
Final Tails	458.6	0.01	0.02	0.02	4.0	11.0		5	8	8	1800	5049	0.6	0.2	0.1	0.2	4.7

TEST #6 Reagent	Stage Addition (lbs/Feed Ton)												
	Condition Time (min)	Grind	Cu Ro. Condi	Cu Cl. Floc'n	Cu 2nd CL Floc'n	Pb Ro. Cond.	Pb Cl. Floc'n	Pb 2nd Cl Floc'n	Zn Ro. Cond.	Bulk Sulphides	Barytes Ro. Fd.	Barytes Cl. Fd.	Barytes 2nd Cl Fd
Na_2SO_3	15	1.0											
SO_2	10, 5		2.0	0.5									
Z-200	2, 1		0.07	0.02					0.05				
NaCN	2, 5, 1			0.05			0.3	0.3					
NaOH	1, 1			0.2		3.7							
Na IPX	1, 1			0.01		0.04							
MIBC	2, 1, 1, 2		0.03	0.01		0.03				0.01			
$CuSO_4.5\ H_2O$	5, 2								0.5	0.5			
KAX	2									2.0			
Aero-Promoter 825	3										5.0		



KT-22 DIFFERENTIAL FLOTATION

	%Cu	%Pb	%Zn	Ag ppm	$\%BaSO_4$	Recovery %
Cu Conc.	17.65	9.95	8.05	20800		50.3 Cu, 52.0 Ag
Pb Conc.	1.60	63.2	9.75	2085		86.1 Pb, 10.5 Ag
Zn Conc.	N/A	N/A	N/A	N/A		N/A
Barytes Conc.					79.3	94.5 $BaSO_4$
Bulk Sulphides	1.61	1.05	25.5	1770		87.6 Zn, 36.8 Ag

SIMPLE LINEAR REGRESSION

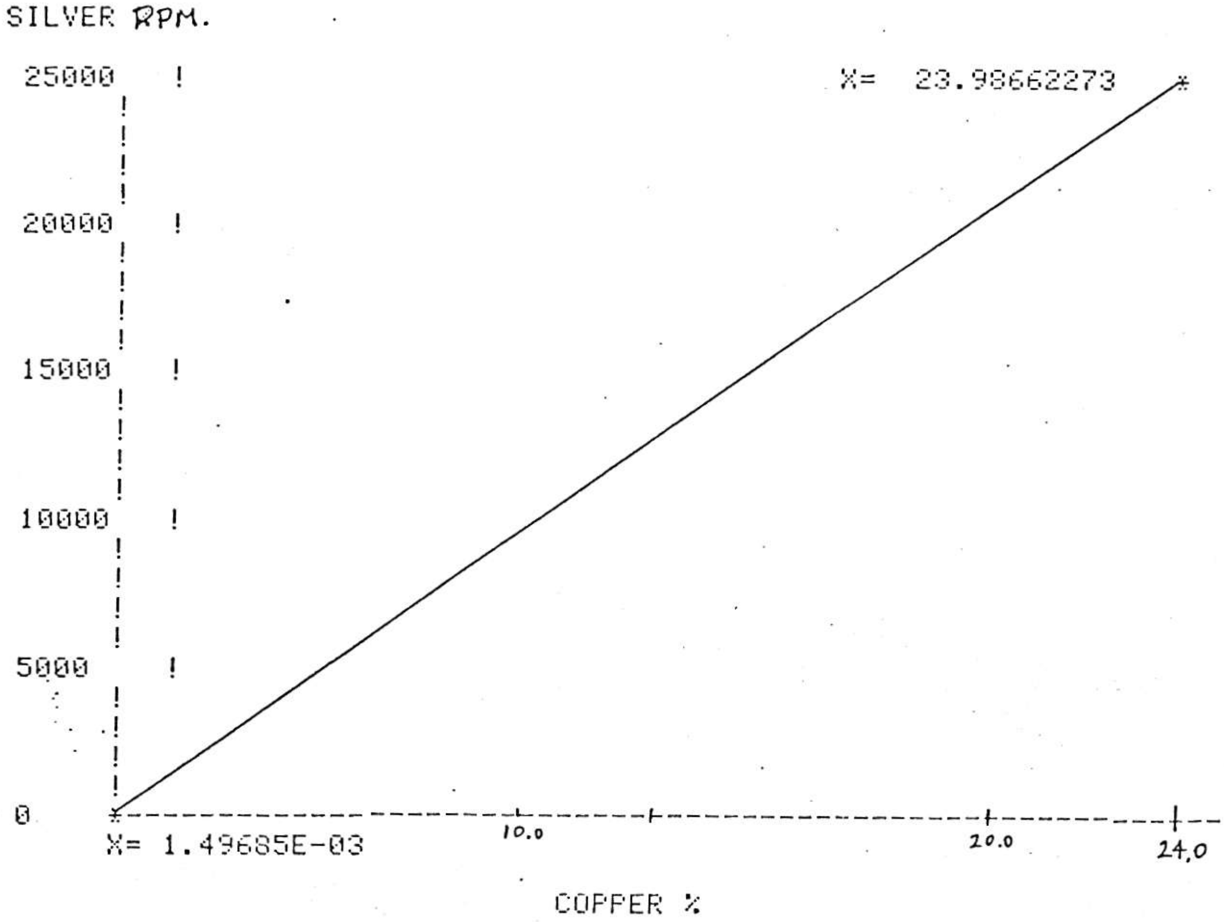
COPPER % VS. SILVER PPM

INTERCEPT B =-1.560188996 =Y WHEN X=0
GRADIENT M = 1042.312645
WHEN Y=0 X = 1.49685E-03
DEGREES OF FREEDOM = 12

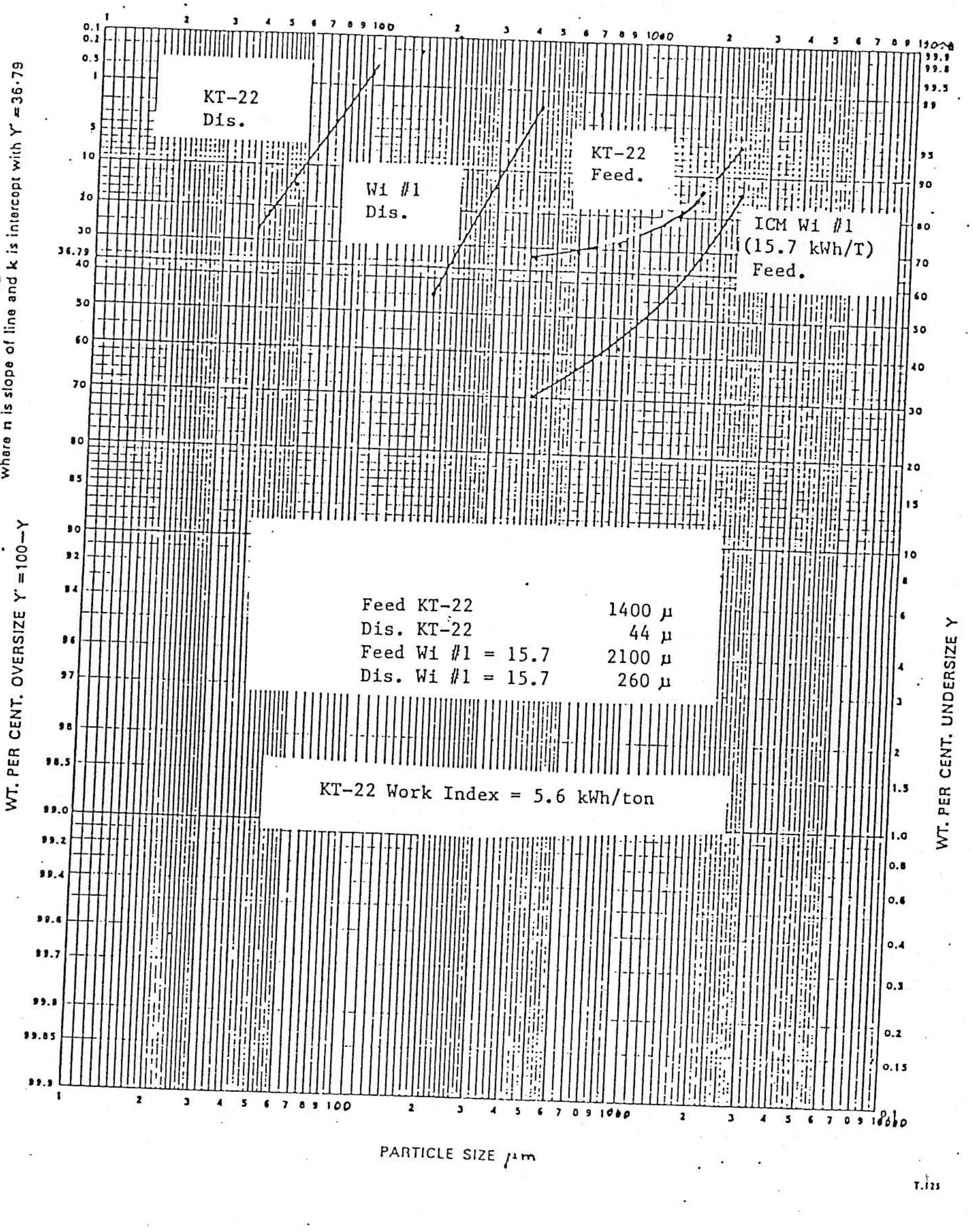
I.E.
EQN. IS:- SILVERPPM=(1042.312645 * COPPER %) + (-1.560188996)

CORRELATION FACTOR = 0.998213037
C.F. WITHIN 1% LEVEL OF SIGNIFICANCE

GRAPH -COPPER % VS. SILVER PPM



APPENDIX BARITE MARKET



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T.123

APPENDIX

BARITE
MARKET

Barite Market

Barite value represents a significant portion of the revenue expected for the Homestake ore. A brief study was made of the Barite market.

Barite consumption and production in Canada has been fluctuating dramatically in the last four years.

	1980	1981	1982	1983
Production (x 10^3 tons)	105	88	33	25

(Source E & MJ, March, 1984)

Barite is an industrial mineral which is used largely (90%) in the oil and gas well-drilling industry.

Barite imports for B.C. and Alberta from 1973 to 1980 were as follows:

Year	Quantity (tonnes)
1973	14,470
74	6,344
75	3,761
76	19,943
77	12,020
78	30,989
79	14,282
80	39,503

(Source CIM Bulletin, September, 1982)

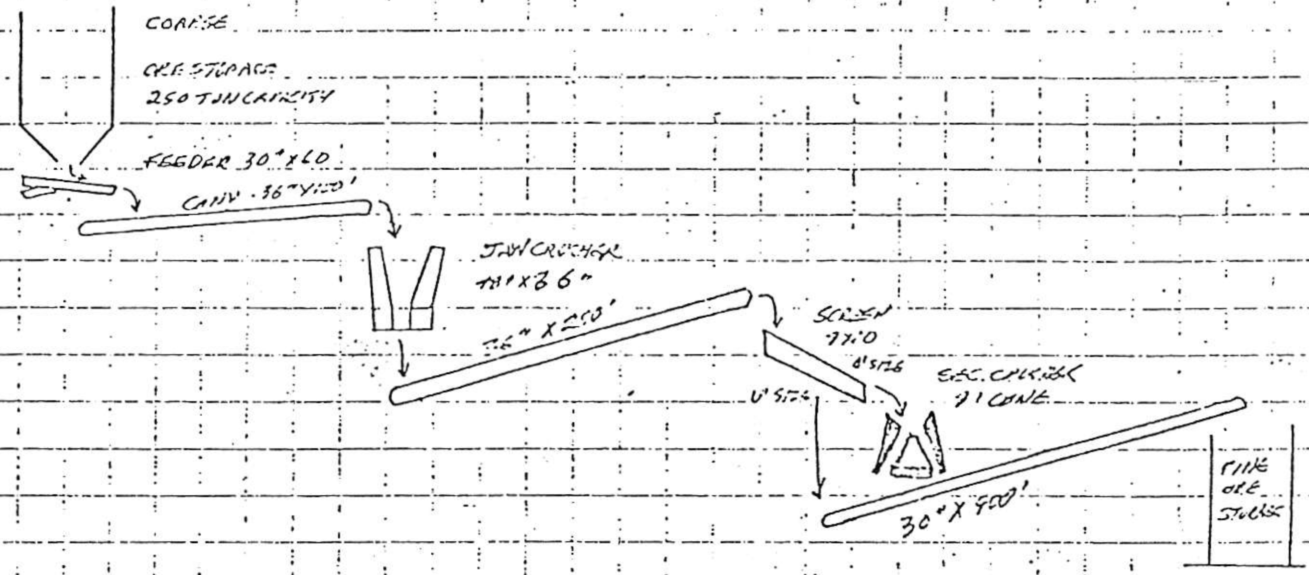
Three possible market areas for Barite to consider are:

a) Western Canadian Sedimentary Basin
b) The Beaufort Sea
c) British Columbia Offshore

The Beaufort Sea is expected to consume large quantities of Barite, however, the Yukon will most likely be the supply source.

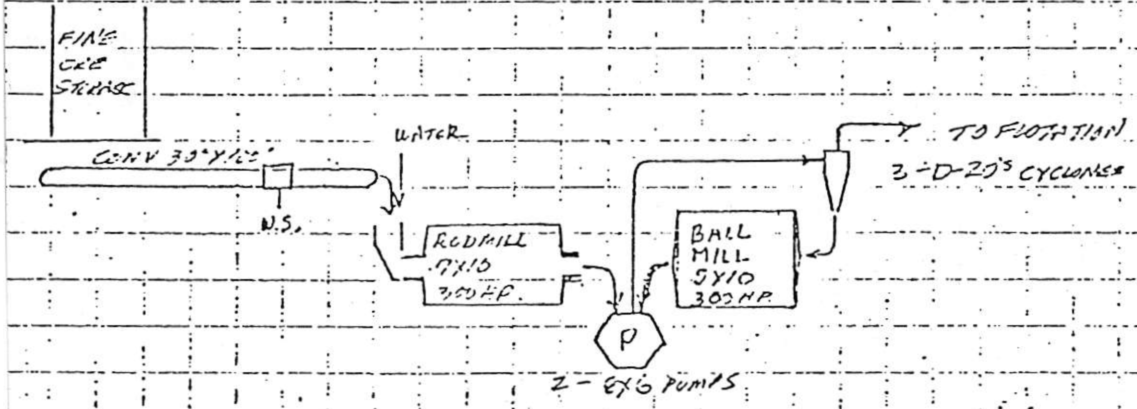
Due to the market uncertainties and offshore drilling unknowns, the large quantities of Barite produced by the Homestake mine will probably be extremely difficult to market.

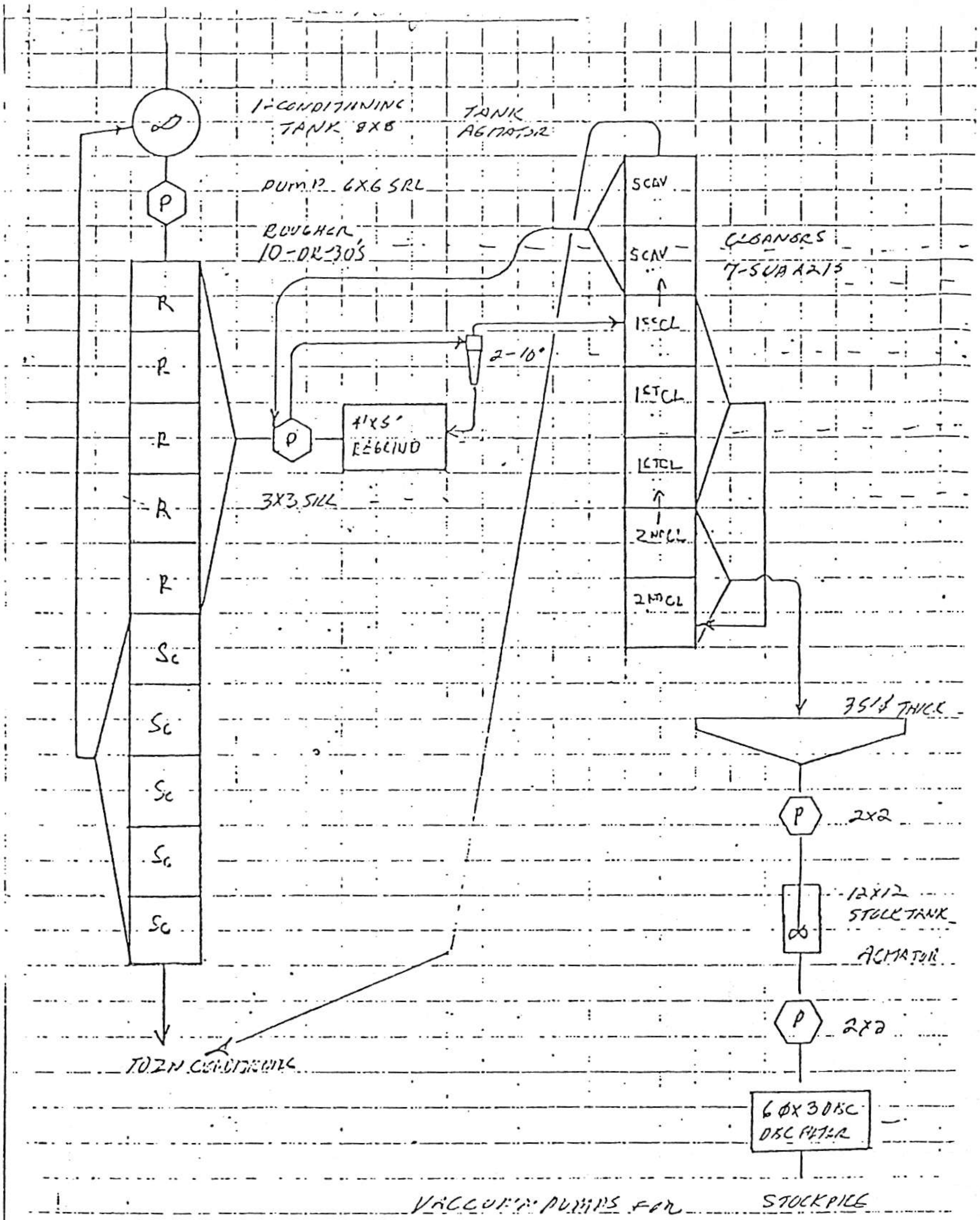
CRUSHING



GRINDING FLOWSHEET

PRELIMINARY



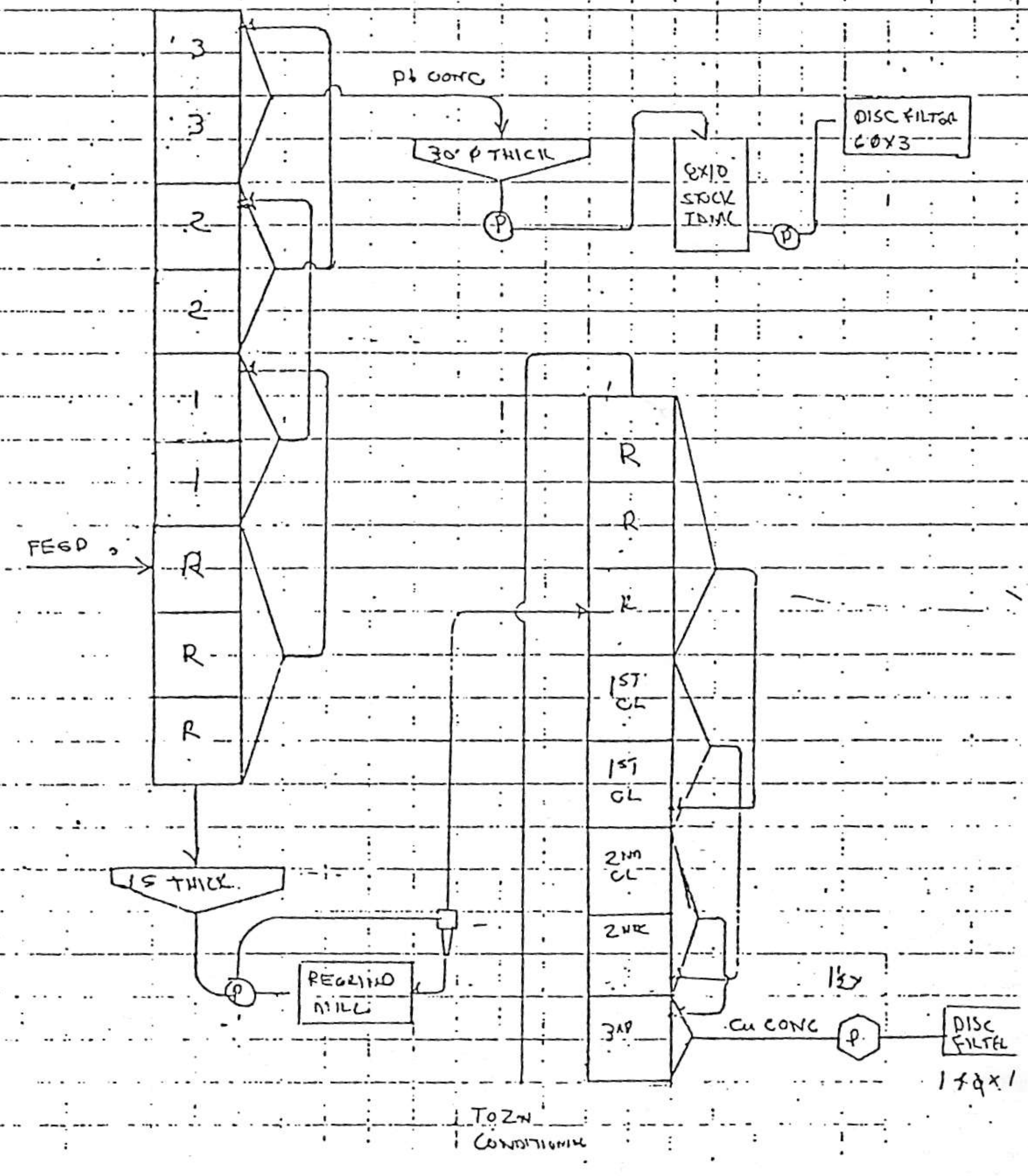

1-CONDITIONING TANK 8X8
TANK AGITATOR
PUMP 6X6 SRL
ROUGHER 10-DR-30'S
R
R
R
R
R
Sc
Sc
Sc
Sc
Sc
3X3 SRL
4'X5' REGRIND
2-10"
SCAV
SCAV
1ST CL
1ST CL
1ST CL
2ND CL
2ND CL
CLEANERS 7-SUB A 21'S
P
35' THICK
2X2
12X12 STOCK TANK AGITATOR
2X2
6 ØX 3 DISC DISC FILTER
STOCKPILE
TO ZN CONDITIONING
VACCUM PUMPS FOR ALL FILTERS - ONE OPERATIONAL - ONE STANDBY

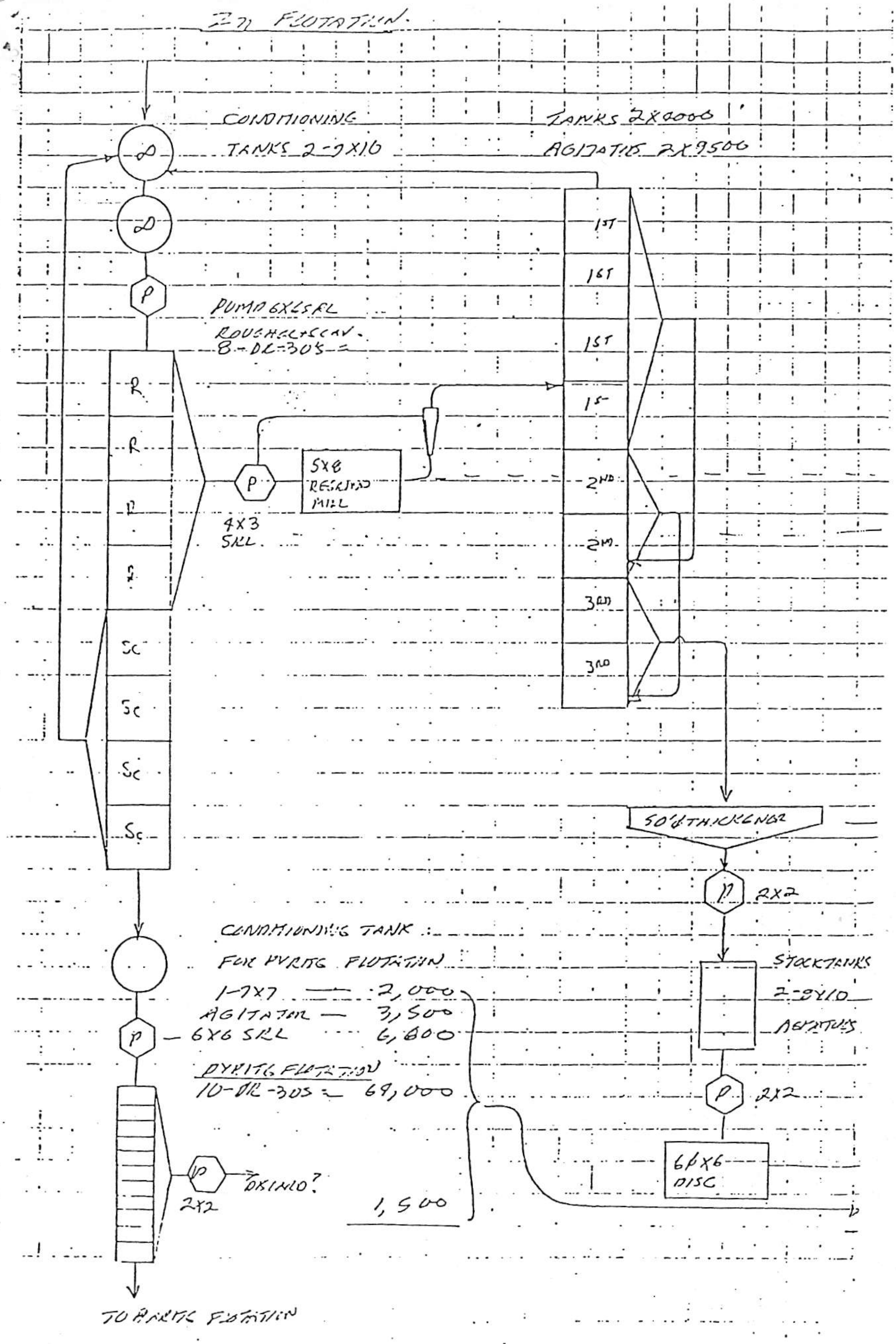
Cu Pb SEPARATION IF POSSIBLE

Rougher Cu Pb conc

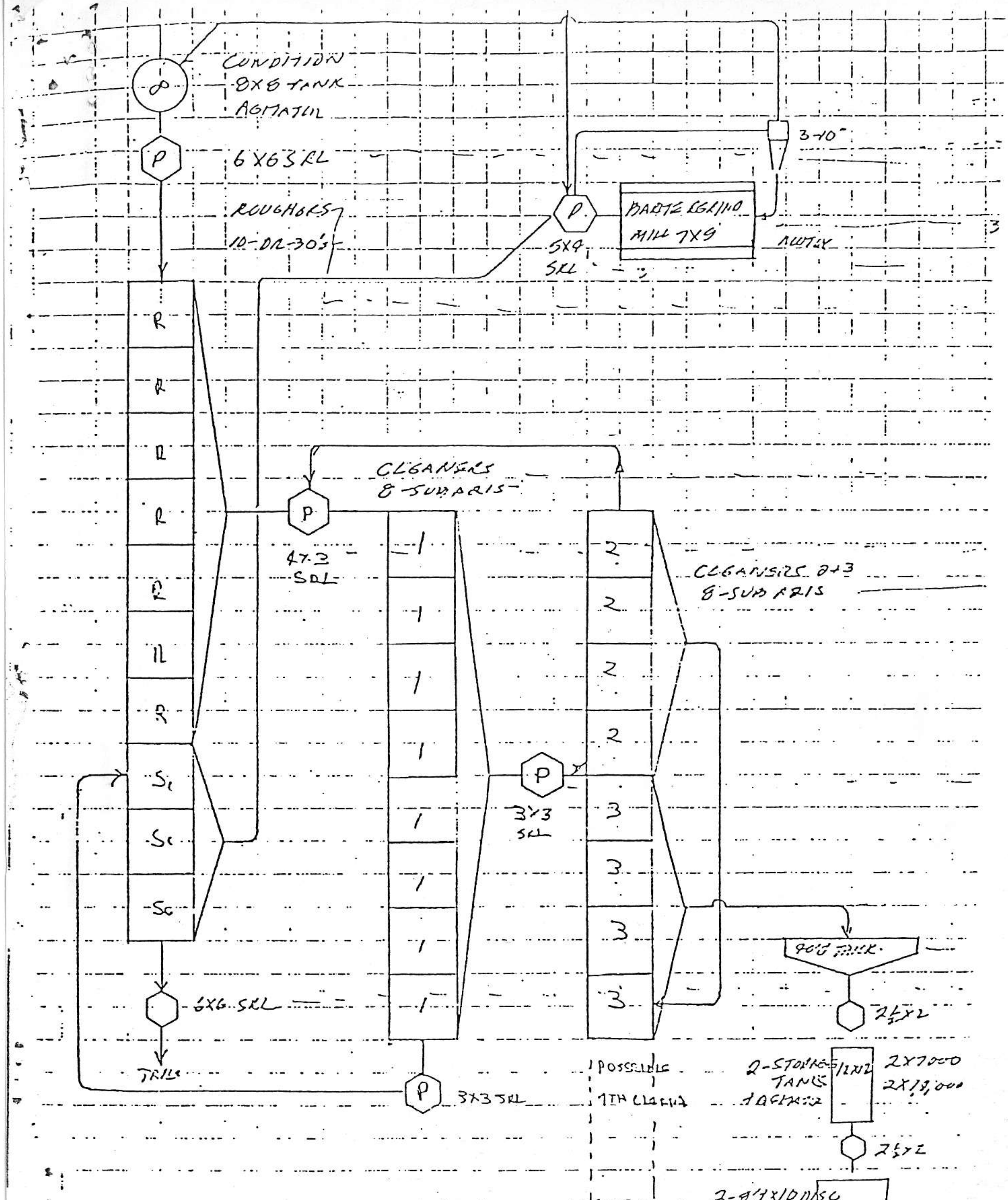
From Cu Pb Ag Circuit of

10-DR-305





Zn FLOTATION
CONDITIONING TANKS 2-7X10
TANKS 2X4000
AGITATORS 2X9500
PUMP 6X6 SRL
ROUGHER+SCAV. 8-DR-30'S =
R
R
R
R
Sc
Sc
Sc
Sc
P
4X3 SRL
5X8 REGRIND MILL
1ST
1ST
1ST
1ST
2ND
2ND
3RD
3RD
50' THICKENER
P 2X2
STOCK TANKS 2-8X10 AGITATORS
P 2X2
6X6 DISC
CONDITIONING TANK :
FOR PYRITE FLOTATION
1-7X7 — 2,000
AGITATOR — 3,500
P — 6X6 SRL 6,800
PYRITE FLOTATION
10-DR-30S = 69,000
P 2X2
DRINO?
1,500
TO PYRITE FLOTATION


CONDITION
8X8 TANK
6X6 SRL
ROUGHERS
10-DR-30's
3-10"
MILL 7X9
5X4
SRL
CLEANERS
4X3
SRL
CLEANERS 2+3
3X3
SRL
6X6 SRL
3X3 SRL
POSSIBLE
2-STORAGE
TANKS
2X7000
2½X2
2½X2
FILTERS