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HUCKLEBERRY PROJECT

Preliminary Feasibility Study

THE GRANBY MINING COMPANY LIMITED
VANCOUVER, B.C.

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I

INTRODUCTION

LOCATION AND ACCESS

The Huckleberry property in west-central British Columbia, at 127° 10' longitude and 53° 41' latitude, is situated on the southern flank of Huckleberry Mountain north of Tahtsa Reach, approximately 55 miles south-west of Houston, which is 193 miles west of Prince George and is served by Highway 16 and the Canadian National Railway. Access to the property is via 70 miles of gravelled public forest access road and 5 miles of private gravel road. The nearest railway is at Houston, although a corridor study has recently been conducted in the Owen Lake region which is approximately 35 miles nearer to the property than Houston. The closest access to tidewater is at Prince Rupert, approximately 350 road miles from the property.

CLIMATE AND TOPOGRAPHY

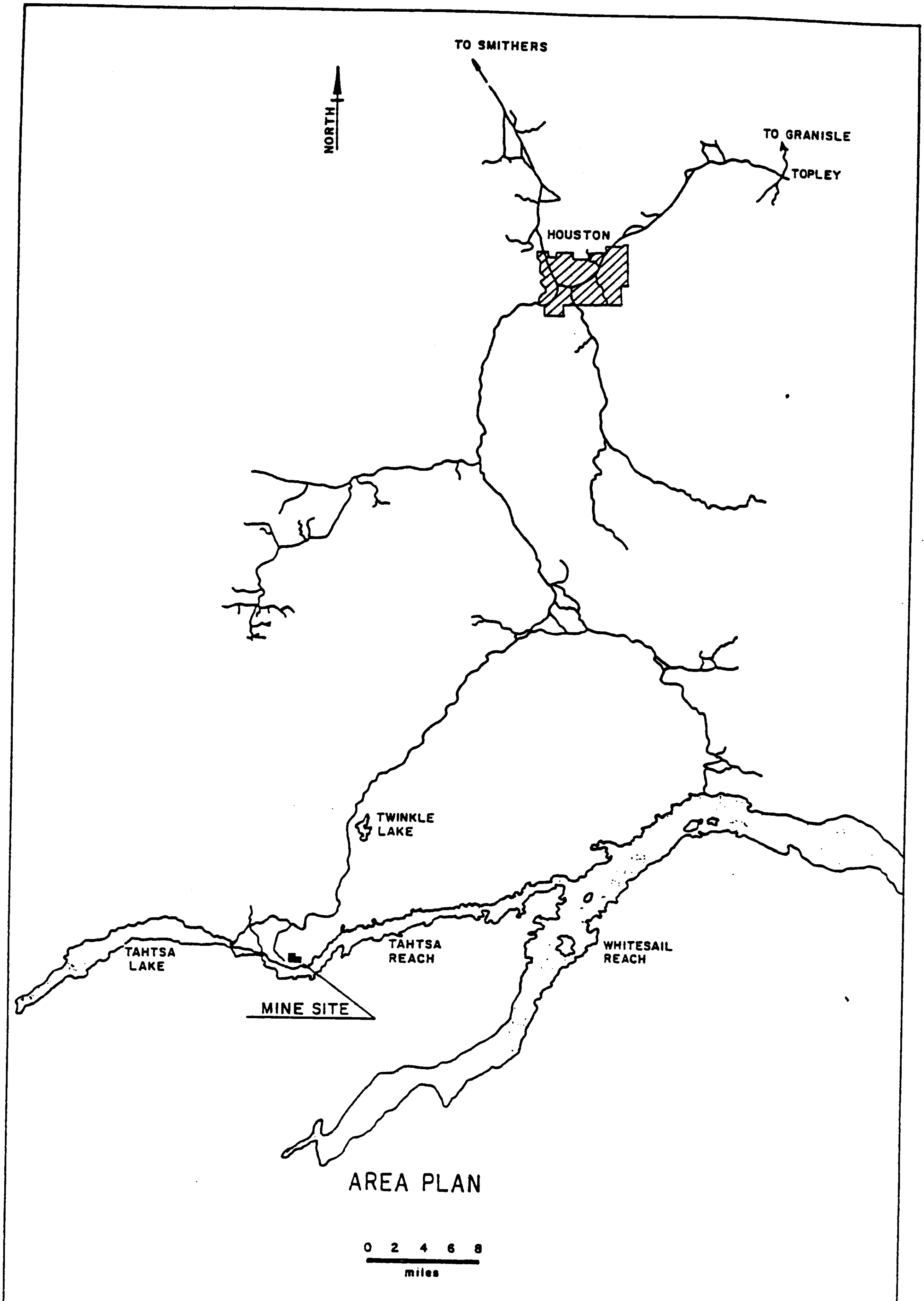
The property lies on the western extremity of the Nechako Plateau at the fringe of the Coast Range mountains. The variability of average rainfall over short distances in this region makes estimation of precipitation at the property rather imprecise. Average precipitation is in the range of 60 inches per year, and average maximum snowpack is 6 to 8 feet. High temperatures in summer are 60 degrees to 85 degrees F. and low temperatures in winter are -40 to -50 degrees F.

The deposit is at an average surface elevation of 3,400 feet, between Huckleberry Mountain at 5,061 feet and Tahtsa Reach which has a maximum surface elevation of 2,820 feet. The Coast Ranges to the west rise to peaks exceeding 7,000 feet.

In the immediate area of the property relief is moderate to steep, the southern slope of Huckleberry Mountain having a gradient of 25% in the north fringe of the proposed pit area. Road access can be provided at relatively flat grades and the topography provides few obstacles to plant and tailings impoundment construction.

LOCATION MAP

Figure I-1



TO SMITHERS
NORTH
TO GRANISLE
TOPLEY
HOUSTON
TWINKLE LAKE
TAHTSA LAKE
TAHTSA REACH
WHITESAIL REACH
MINE SITE
AREA PLAN
0 2 4 6 8
miles

Figure I-2

HISTORY OF PROPERTY

Kennco Work

The discovery resulted from surface prospecting following up anomalous copper values in stream silt in the course of regional geochemical exploration. In 1962 the property was staked by Kennco Explorations (Western) Limited, and some preliminary drilling totalling 953 feet was carried out in 6 small diameter holes. In 1964, standard equipment was used to drill 9 holes totalling 4,647 feet. The property was then left until 1970 when a further 9 holes totalling 4,557 feet were drilled, topographic and geophysical surveys were conducted and considerable surface trenching done. In 1971 work was continued with drilling of 5 holes totalling 2,854 feet. Kennco then decided to farm the property out.

Granby Work

Early in 1972, Granby's offer to develop the property was accepted. At least two substantial competitors also made offers. In 1972 Granby drilled 18 holes totalling 9,282 feet, exploring the most promising portion of the known mineralization to a depth of 500 feet. Granby also had a claim survey done by a B. C. land surveyor. The drilling results were considered encouraging and in 1973 a total of 43,824 feet in 47 holes were drilled to further explore the area laterally to an average depth of 1,000 feet. The road extension providing access to the property was improved and gravelled and topographic surveys were completed.

Most of the 1973 holes were vertical and at 200' grid spacing; the results of this and prior drilling form the basis for the feasibility study.

TERMS OF AGREEMENT

The fundamental provision of the agreement between Granby and Kennco is that Granby may acquire a 50% interest in the Huckleberry property by spending $1,500,000 on further exploration. Of this amount, $500,000 must be spent before December 31, 1974 and the rest by December 31, 1989. Granby's accountable expenditures to date are approximately $1,000,000. In the 15 year period

December 31, 1974 to December 31, 1989, there are no fixed expenditure schedules, but if Granby does not carry out work and Kennco considers work should be done; Kennco may propose programs which Granby must carry out or relinquish all interest upon Kennco executing the program.

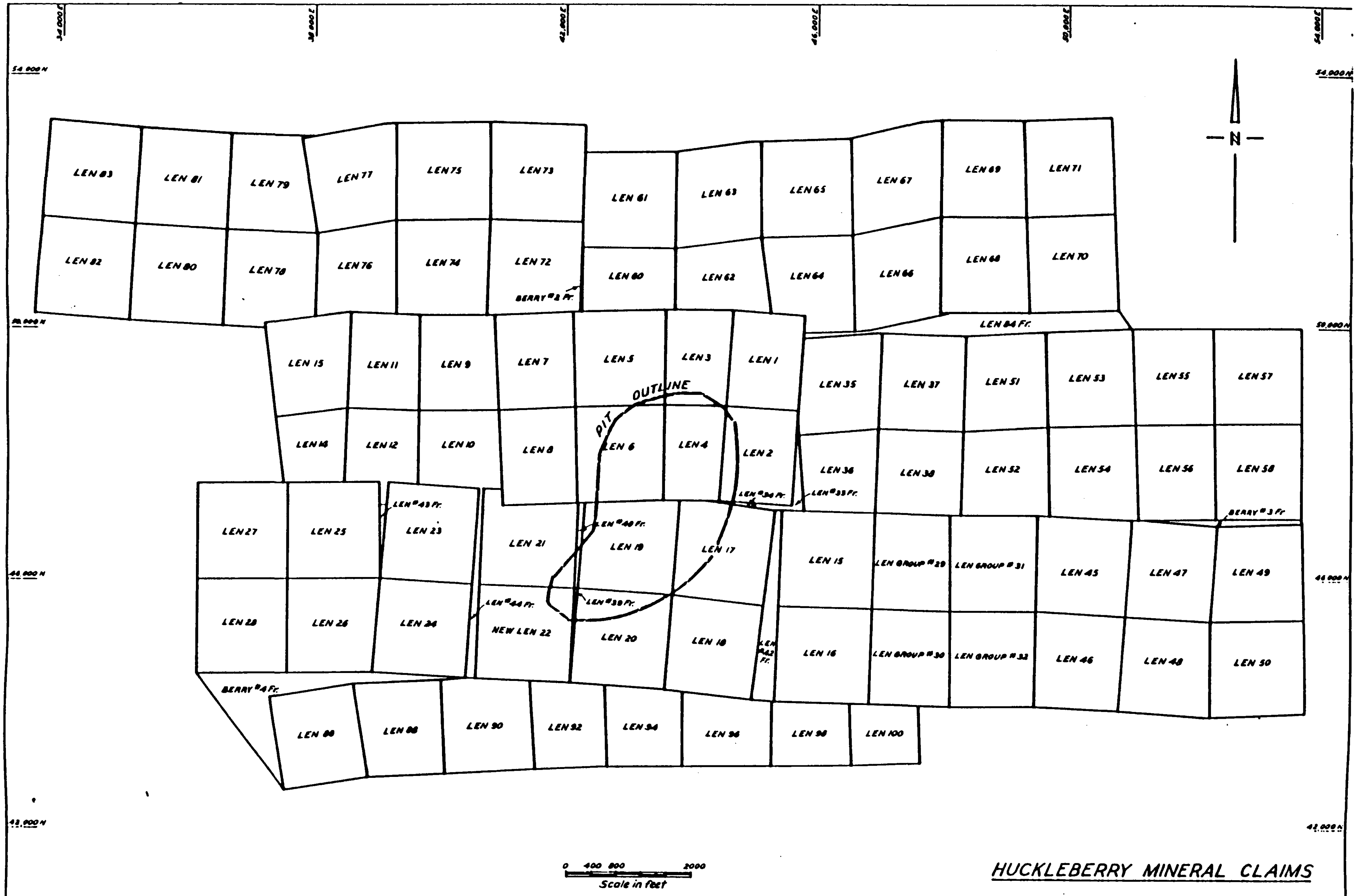
When Granby has earned its 50% interest it may continue to propose and manage work to further develop the property and place it in production on a joint venture basis. Kennco must either pay half the cost or allow its interest to decline to an irreducible minimum of 30%. If Kennco does not contribute Granby will earn the additional 20% interest when it has spent a further $2,000,000. If Granby does not take the initiative to place the property in production, Kennco may do so. Granby can either contribute or retain a 30% interest.

All money spent by either party after the point at which Granby has earned the half interest is treated as advances, repayable out of cash flow after any institutional borrowing but before any distribution to the interest holders. The agreement contemplates the possibility, by mutual agreement, of transferring the property to a company, rather than operating it as a joint venture. In this event the interests would be represented by common stock and returns would come as dividends.

MINERAL CLAIMS

The Huckleberry property consists of 94 located mineral claims in the Omineca Mining District as shown in Figure I-3. The claims are owned by Kennco Explorations (Western) Limited and are held under option by the Granby Mining Company Limited. The option agreement currently covers the following mineral claims:

CLAIM	YEAR LOCATED
LEN 1-21; 23-32	1962
LEN 33-44	1963
LEN 45-58; 60-84	1969
LEN 86,88,90,92,94,96,98 & 100	1971
NEW LEN 22; BERRY 2-4	1972



HUCKLEBERRY MINERAL CLAIMS

Sufficient work has been recorded to keep all claims except one in good standing until 1990 or later. Work recorded on the BERRY 4 fractional mineral claim will keep it valid until 1982.

ACKNOWLEDGEMENTS

This study is based on data accumulated over 12 years by personnel associated with Kennco Explorations (Western) Limited and, since 1972, the Granby Mining Company Limited. The recent work was primarily under the supervision of D. H. James, who has provided much of the geological input. Granby field supervision was principally by J. Paxton and later D. G. McIntosh. The plant capital cost study, from which the bulk of the capital costs were drawn, was compiled by Kilborn Engineering (B.C.) Limited. Specialized input for several areas of the study was provided by numerous individuals, the most significant contributors being D. G. McIntosh, M. J. McGarry and K. C. Fahrni. B. Raymor was responsible for typing and initial proofing.

II

SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

1. The Granby Mining Company Limited, under the terms of an agreement dated June 1, 1972, has the right to acquire an interest in approximately 4,000 acres of mining properties consisting of 94 located mineral claims. The property, which has been optioned from Kennco Explorations (Western) Limited, a subsidiary of Kennecott Copper Corporation, is located on Tahtsa Reach in north-central British Columbia, approximately 75 road miles south of the town of Houston, B. C.

2. The agreement requires Granby to spend $1,500,000 by 1989 of which approximately $1,000,000 has been spent to date. Upon Granby having fulfilled their expenditure obligation, a joint venture can be formed with each partner contributing their 50% share of future expenditures. If either party fails to maintain their contribution, their interest will be diluted to a minimum of 30%. The agreement also provides that a company could be formed by mutual agreement of the two parties.

3. The purpose of this preliminary feasibility study is to evaluate the data acquired to date, to assess the economic potential of the property, and to make recommendations for onward work.

4. The mineral claims cover deposits containing copper and minor amounts of molybdenum, gold and silver. Exploration activities to date have consisted of 66,117 feet of diamond drilling including 43,824 feet drilled by Granby in 1973.

5. The drilling has indicated the presence of a copper deposit approximately 2,500 feet long by 1,500 feet wide and containing a higher grade zone approximately 1,000 feet long by 500 feet wide. Reserve estimates indicate that the following tonnages of material could be mined by conventional open pit mining methods.

Reserve Tonnage (Millions)	Reserve Grade (% Cu)	Cutoff Grade (% Cu)	Stripping Ratio (W:0)
27.9	0.501	0.40	1.28
42.6	0.353	0.25	1.28
15.1	0.347	0.30	--
85.6	0.401	0.30	1.11

Due to the distribution of mineralization within the deposit, it is improbable that the deposit could be mined on a more selective basis.

The presence of mineralization in some of the outlying drill holes suggests that larger tonnages of low grade material may be present at higher than average stripping ratios. In our opinion, this material could increase the mine life but is unlikely to significantly improve the economic viability of the property.

6. Investigations of drill core samples indicate that the copper mineralization can be recovered by grinding to 60% minus 200 mesh, followed by flotation. The studies suggest that the northern and eastern portions of the deposit, which include most of the material below 0.40% Cu, are appreciably harder grinding than the remainder of the deposit. Ball Mill Work Indices, an indication of grindability, ranged from 10.3 to 19.2 and averaged 14.4 for the ten samples tested. Studies of the flotation characteristics concluded that a concentrate containing 27% copper could be attained with total copper recoveries of 93 - 95%, depending upon the feed grade. Molybdenite recovery appears to be in the order of 50%; however, because of its minor importance, it has not been fully investigated and is not considered in the evaluation.

7. Estimates of capital and operating costs were prepared for plant capacities of 10,000 and 15,000 tons per day and for various mining schedules. Due to the significantly lower capital costs per ton of plant capacity, a 15,000 ton per day plant offers economic benefits in comparison to a lesser mining rate.

At a treatment rate of 15,000 tons per day (5.475 million tons per year), the plant could operate for approximately five years on the higher grade material and nine years or more on the lower grade material. Larger plant sizes have not been investigated in this preliminary study as they would significantly reduce the total mine life and the period of operation on the higher grade material.

8. The widespread and rapid inflation currently being experienced in British Columbia and throughout the Western world, introduces considerable uncertainty into estimates of future costs and revenues. Consequently, all estimates have been based upon January - March, 1974, information, without taking into account either future escalation or any short term disparities which might have existed at that time.

9. The capital and operating cost estimates have been based upon the implementation of a mining operation which will minimize labor content and operating costs. Certain operating concepts incorporated in this report may differ from accepted practice at some mining operations of comparable size. The effective operating costs and consequently the profitability of the project, are dependent upon adherence to these concepts in the onward development of the project.

10. It has been estimated that an investment of $69 million would be required for a 15,000 ton per day concentrator, associated mining and surface facilities, townsite, power, permanent road, inventory, working capital and a 10% contingency. Operating costs, assuming an operating stripping ratio of 1.28:1, were estimated to be $2.30 per ton milled. At 10,000 tons per day and 1.25 stripping ratio, the estimates were $64 million capital investment and operating costs of $2.66 per ton milled. Manpower requirements have been estimated at 254 and 222 respectively.

11. Economic projections have been based upon copper prices of $0.80 to $1.00 per pound, a gold price of $150.00 per Troy ounce, a silver price of $5.00 per Troy ounce, and typical current smelting and refining charges for Japanese smelters. The increase in treatment charges over the last five years due to a combination of inflation, pollution control costs,

and together with the recent increases in energy costs, limits the utility of historical data in predicting the current long term average copper price. Nevertheless, an attempt has been made to consider these factors, and from this study we believe that a long term average copper price will lie between $0.75 and $0.85 per pound, and that a price of $0.80 per pound (February, 1974, base) should be utilized for decision making purposes.

12. The Government of British Columbia has introduced legislation which, if enacted, requires the payment of a royalty of 5% of the Net Minesite Revenue, plus a "super" royalty of 50% of the amount by which the metal price received by the producer exceeds 120% of the average price received over the preceding five years. It is expected that provision will be made to account for inflation, so that the effective royalty would be a simple 5%, except for the peak years in the metal price cycle. The effect of the "super" royalty on profitability cannot be accurately predicted; consequently, the evaluation has been based on a 5% royalty. Management should subjectively raise their minimum profitability criteria to account for the "super" royalty coming into effect once during the debt retirement period.

13. The economic projections for a 15,000 ton per day plant indicate that the internal rate of return would be approximately 8.5% at a copper price of $0.80/lb.; 11.0% at $0.85/lb.; 13.5% at $0.90/lb.; and 18.1% at $1.00/lb. The 10,000 ton per day plant alternative would require a copper price of $1.00/lb. to generate an 11.1% rate of return.

14. The profitability of the project is highly sensitive to metal prices and capital costs. Within the price range of $0.80 to $0.90 per pound copper, a change of $0.05 (5.9%) in the price of copper changes the internal rate of return by 2.5 units or approximately 22%. Similarly, changing the direct capital costs by 10% ($6.4 million), changes the internal rate of return 1.6 units or approximately 15%.

15. If the project is financed by $15 million equity capital and 54 million debt at 9% interest and repayable from first available cash flow, the debt could be retired within five years from the start of production, providing that the copper price were to average $0.85 per pound. The internal

rate of return on the equity investment would be 12.5% over the projected mine life. Other financing alternatives have not been investigated.

16. The Huckleberry Project would be sub-economic at the forecast costs and a copper price of $0.80 per pound, but could be marginally attractive if the long term copper price averages $0.85 per pound; or if the grade or tonnage of the higher grade zone can be increased; or if the capital investment in the townsite and power supply were borne by the public sector.

17. It is recommended that until the areas of uncertainty with respect to royalties and infrastructure development are resolved, development work on the project be continued on a modest scale in order to obtain additional background information on the proposed plant and townsite areas, and to increase the reliability of certain estimates through further engineering studies. The cost of such a program would be in the order of $150,000. The necessity of additional drilling is being investigated and, if warranted, would cost an additional $200,000.

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III

GEOLOGY

REGIONAL GEOLOGY

Regional geology of the Huckleberry area is described in GSC Memoir 299, Whitesail Lake Map-Area and the accompanying map at 1" to 4 miles published in 1959. Mapping in the area predates discovery of the Huckleberry deposit.

The deposit occurs in rocks of the Hazelton Group of Jurassic age around a quartz diorite stock dated at 80 million years. This stock is one of a number of small Eocene intrusives occurring east of the Coast Batholith in central British Columbia. Several other showings of a similar nature occur in the Smithers area. The Granisle mine and Bell Copper mine are somewhat younger but have features in common with the Huckleberry deposit.

LOCAL GEOLOGY

Geology in the vicinity of the deposit was mapped by Kennco geologists and has been reported on briefly in annual reports of the Minister of Mines for 1964 and 1970. Granby staff have not re-mapped the area, but have attempted correlations between previous and recent drilling, and have studied a number of thin sections, slabs and polished specimens.

Intrusive Rocks

Porphyritic Quartz Diorite:

Mineralization in the area is controlled by an intrusive stock approximately 1200 feet wide and 2500 feet long, elongated in a N 60 E direction. The intrusive is a grey rock composed of white zoned plagioclase phenocrysts as much as 1 cm. long, quartz phenocrysts 1-2 mm. across and biotite flakes 1-5 mm. across, in a matrix made up of the same three minerals with varying amounts of orthoclase. It is likely that the orthoclase is a later alteration mineral. Biotite is partly altered to chlorite. Alteration intensity increases at the margins; the rock appears correspondingly darker in hand specimens. More quartz and some orthoclase are introduced into the groundmass and biotite is more altered than in the interior of the stock. Rock resembling this altered phase is intersected in some drill holes away from the main stock, indicating probable dyke extensions from it.

Dykes:

Two types of post-mineral dykes occur, referred to in logs as lamprophyre and diabase. Thin sections studied indicate only diabase, composed of ophitic plagioclase laths and interstitial biotite, magnetite and carbonate. The lamprophyre appears to be an inequigranular phase of the same mineralogical composition.

Volcanic Rocks

No fresh volcanic rocks have been encountered. All specimens have been hornfelsed by the intrusive. Megascopically the volcanics appear principally fragmental with some micro porphyritic specimens and a few sections of aphanitic or micro granular material. Fragmental specimens consist of rock fragments less than 5 mm. across, principally composed of the same minerals as the surrounding groundmass. There are also numerous crystal fragments of zoned plagioclase and rarely of quartz. Where only the plagioclase is visible the rock appears porphyritic. The groundmass is an extremely fine grained mosaic of quartz, plagioclase and biotite. It is usually seen corroding the margins of plagioclase phenocrysts and is assumed to be replacing them. Plagioclase in the groundmass is interpreted to be the only remains of the original mineral in the rock. Occasionally the quartz biotite replacement is accompanied by orthoclase, but this mineral is usually restricted to fractures and margins thereof.

Aphanitic specimens have the same composition as the fragmental. Granular material contains more plagioclase than quartz. In three of the thin sections the plagioclase is somewhat ophitic and may indicate an original flow or dyke rather than tuff. Two contain a fibrous, rather low birefringent mineral, probably tremolite-actinolite. Most rocks are dark, due to the presence of fine-grained biotite and magnetite. Magnetite also occurs in fractures as does (rarely) hematite.

Mineralization

Mineralization consists of chalcopyrite and minor molybdenite in fractures, principally in the hornfelsed volcanics but also in the quartz diorite stock. The fractures are usually thin, frequently hairline and rarely more than 2-3 mm. wide. The minerals accompanying chalcopyrite appear to be quartz, orthoclase and pyrite. Calcite, gypsum and zeolite are likely later. Sericite may accompany quartz. Magnetite occasionally occurs in the same fracture as chalcopyrite. Chalcopyrite is also occasionally seen disseminated in grains probably replacing biotite, but only a small proportion of it occurs in this manner. Molybdenite is usually on hairline fractures associated with quartz.

Alteration

Alteration of the hornfels appears to be associated with the mineralization. An alteration of brown biotite progressively to green biotite and eventually chlorite is the most widespread effect; a few specimens are broadly pervaded by orthoclase. Most of the alteration is adjacent to fractures, where biotite and chlorite are altered to quartz and sericite, producing "bleached borders". Orthoclase often replace plagioclase. Very highly altered sections of rock are almost entirely quartz and sericite. Similar alteration effects were observed in portions of the intrusive.

Structure

Mineralization is found around the intrusive contact, but apparently has no systematic detailed relation to it. The mineralization may extend only a few tens of feet from the intrusive in some places and several hundred feet from it in others. The distribution of molybdenite has not been studied due to its comparatively minor value, but it appears to be concentrated close to the intrusive contact and for some distance inside the stock in sections where chalcopyrite is also present. Margins of the intrusive are steep, but may be as flat as 60° to the east end where current work is being done. Alteration appears to extend farthest in this direction also. Faults are intersected in many holes but are not easily correlated. One substantial fault is parallel and close to the southeast side of the intrusive and another may well lie on the northwest side.

An effort has been made to relate the distribution of chalcopyrite as indicated by the detailed drilling to structure and lithology. Zones of higher and lower grade appear to dip at shallow angles south and east, but no associated variations in rock type are discernable.

IV

MINERAL RESERVES

INTRODUCTION

The Huckleberry mineral reserve estimate is based on assays of split AQ and NQ core from diamond drilling done for Kennco in 1964 and 1970 and for Granby in 1972 and 1973. Earlier small diameter drilling results were not used because of poor core recovery. Sample preparation and assaying were done at several different laboratories with check assays by a separate laboratory on every tenth pulp.

RELIABILITY OF DATA

The level of confidence in a given statement of mineral reserves is affected by two classes of error. These are:

a) the error due to the extrapolation of the grade of a sample to a designated zone of influence;

b) a sampling error which has three components:

   (i) sampling proper - drilling methods and core recovery

   (ii) sample preparation - core splitting and splitting of crushed material

   (iii) assaying.

Extrapolation Error

The error of extrapolation is related to variability of the mineralization, and the geometry of the zone of influence. These errors, or variances, are impossible to measure by using standard statistical techniques, because of spatial considerations. Geostatistical techniques may be used to quantify confidence limits of geologic and mineable reserves, and may also be used to indicate further work required to reduce the error of estimation to some acceptable limit. No effort to employ these statistics has been made to date.

Sampling Error

Because core recovery was consistently above 95%, variance due to sampling technique at the collar should be accepted as minimal.

The possibility of additional error due to core splitting was investigated. Portions of halved drill core were assayed twice, once for each half, and variance tests were calculated under the assumption that the assay pairs were independent. These calculations showed that no differences existed between the two sets of samples at the 95% confidence level. Therefore, the core splitting did not affect subsequent assaying.

A similar check scheme was not used with respect to the splitting of crushed samples (sample rejects were not assayed), and direct inferences cannot be drawn. However, it has been assumed that no significant error has resulted by splitting.

Independent assays for every tenth pulp sample were obtained as a check on the assay results of the primary assaying scheme. Variance tests were performed on the data groups as paired data, and it was found that these assays had the same mean at the 95% confidence limit. The conclusion is that the assays have no significant error affecting the reliability reserve calculations.

In general, while no numerical estimate of error due to sampling, expressed as a variance of the mean is submitted, the drill hole sampling scheme employed can be considered reliable.

BLOCK GRADE INTERPOLATION

The mineral reserve is estimated by dividing the mineralized volume into 40' horizontal benches and 50' by 50' area elements within each bench. Each 50' x 50' x 40' block is assigned a grade by an averaging technique, using all drill hole assay data in the vicinity of the block. The resultant three dimensional grade matrix is the model from which geological reserves are calculated and which in turn form the basis for determining pit designs and mineable reserves.

The calculation procedure commences with the determination of the block positions and the calculation of the arithmetic average grade for each block that contains assay data. (The drill hole assays are first averaged into 40' vertical bench composites and these values are used in interpolation.)

The centroid of each block is considered the point location of the corresponding grade element and all distances used in weighting are measured from this point. A test is made to see if the block is surrounded by drill holes, i.e. if it is an internal or external block with respect to the drill hole pattern. For internal blocks a search radius of 300' and a vertical constraint of two benches above and two benches below the block are used to limit the number of bench composites used in determining a block grade. For external blocks the search radius is 150'. The block grade value is determined by weighted averaging of all drill hole bench composite grades within a sphere of the given search radius surrounding the block with composite grades weighted by the inverse square of the distance from the block. Given a series of composites G_1, G_2, G_3 .... with corresponding radial distances d_1, d_2, d_3 .... from the block centroid in question, the block grade is:

$$G = \frac{\sum_{i=1}^{n} \left(\frac{G_i}{d_i^2} \right)}{\sum_{i=1}^{n} \left(\frac{1}{d_i^2} \right)}$$

The block grades can be printed out on bench plans and are used in stored form within a computer system as input to the pit design program. They can also be used to produce tonnage and grade figures for a geologic reserve at any given cut off.

The distance constraints on interpolation were determined by the drill hole configuration and the practical limits imposed by volume of data. The 300' horizontal internal search radius ensured that every internal block within the drilled out zone was assigned a grade value; the 150' horizontal external search radius limits extrapolation of values into untested ground to half the distance of maximum interpolation. Ideally, lacking a definite geologic bias, the interpolation technique should be spherical, using the same search radius in all directions. This would require including values up to seven benches above and seven benches below each block. The volume of data handled for each block grade interpolation in a true spherical search would make the program very cumbersome and expensive to run, so a practical limit of two benches above and below was placed on the vertical dimension, making the search volume a truncated sphere. A true spherical interpolation was done on one bench to compare to the truncated spherical method; the comparison showed some extension of low grade assay boundaries and dilution of high grade core zones by the true spherical method. However, the effective difference in the two results was marginal.

Geologic Reserves

Geologic reserves were calculated using the computerized interpolation model described elsewhere. The volume for interpolation was bounded as follows:

	FROM	TO
Latitude	45,000N	49,000N
Departure	42,000E	45,000E
Elevation	3,600	2,680

In addition, only those 50' x 50' x 40' blocks that satisfied the location constraints outlined in the description of the interpolation model would have an interpolated value. Thus, the limits of interpolation are not geologic, but geometric. The tonnage factor was assumed to be 12.0 cubic feet per ton; a block would thus carry 8,333 tons. For purposes of tonnage calculation, no partial blocks were considered in the model. Table IV-1 shows the bench geologic reserves.

PIT DESIGN

Design Concept

The design of an optimum pit is not strictly necessary for a maximum return on investment. Ultimate pit limits are substituted as an approximation to an optimum pit, and these limits provide the boundaries within which production planning takes place. Production planning affects the return on investment to a greater degree than the selection of ultimate pit limits, because the mining of higher grade material in the early years increases the net present value. The approach used to calculate pit limits, from which a production schedule

Table IV-1

GEOLOGIC RESERVES

CUTOFF	.25		.30		.35		.40		.45	
BENCH	TONS	GRADE	TONS	GRADE	TONS	GRADE	TONS	GRADE	TONS	GRADE
3560	33,332	.285	8,333	.320						
3520	108,329	.283	24,999	.337	8,333	.370				
3480	374,985	.284	133,328	.314						
3440	2,258,243	.337	1,324,947	.386	566,644	.475	258,323	.608	158,327	.73
3400	5,149,794	.376	3,874,845	.411	2,566,564	.459	1,649,934	.509	1,066,624	.55
3360	5,983,094	.375	4,424,823	.412	3,191,539	.448	2,074,917	.491	1,366,612	.527
3320	6,749,730	.382	5,258,123	.413	3,833,180	.449	2,524,899	.490	1,816,594	.518
3280	7,458,035	.392	6,166,420	.418	4,549,818	.451	3,066,544	.490	1,824,927	.54
3240	7,974,681	.403	6,866,392	.424	5,191,459	.457	3,491,527	.501	2,416,570	.53
3200	8,308,001	.402	7,033,052	.425	5,383,118	.458	3,774,849	.497	2,449,902	.539
3160	8,616,322	.396	6,874,725	.427	5,133,128	.463	3,716,518	.499	2,649,894	.53
3120	8,316,334	.393	6,633,068	.423	4,966,468	.458	3,541,525	.494	2,533,232	.529
3080	7,999,680	.389	6,283,082	.422	4,924,803	.450	3,249,870	.490	2,158,247	.526
3040	7,383,038	.382	5,758,103	.414	4,333,160	.445	2,899,884	.484	1,733,264	.526
3000	6,716,398	.376	4,983,134	.413	3,766,516	.444	2,558,231	.479	1,466,608	.52
2960	6,299,748	.366	4,741,477	.397	3,316,534	.431	2,108,249	.466	1,058,291	.515
2920	6,308,081	.351	4,616,482	.381	2,924,883	.418	1,566,604	.461	816,634	.50
2880	6,133,088	.346	4,608,149	.370	3,749,890	.403	1,291,615	.442	449,982	.48
2840	5,608,109	.347	4,099,836	.376	2,558,231	.410	1,308,281	.450	516,646	.50
2800	4,691,479	.363	3,574,857	.392	2,591,563	.418	1,274,949	.469	683,306	.514
2760	4,124,835	.367	3,199,872	.395	2,399,904	.420	1,283,282	.465	524,979	.538
2720	3,674,853	.364	2,958,215	.386	1,983,254	.419	1,016,626	.468	424,983	.544
2680	3,424,863	.355	2,533,232	.383	1,683,266	.416	758,303	.473	424,983	.516
TOTAL	123,695,052	.377	95,979,494	.408	68,622,255	.443	43,414,930	.483	26,540,605	.529

can be drawn, was to assume constant price and cost parameters, and to generate different pit surfaces by changing the design cutoff grade. A number of objectives are achieved in this manner.

1. The sensitivity of both the average grade and the stripping ratio to changes in design cutoff can be observed and used as a guideline along with total tonnages, grades, and stripping ratios, in the selection of an ultimate pit.

2. The above figures can be used in conjunction with basic calculations of marginal cutoff grades. These calculations are elementary and should thus be considered simply as a framework for production planning.

3. The incremental changes in grades and stripping ratios resulting from changes in design cutoff will indicate how critical the selection of an intermediate pit is. The more highly defined a central high grade zone is, the more important it is that the best intermediate, or payback pit be correctly defined.

4. The increasingly larger pit surfaces generated by successively lowering the design cutoff grade will outline a mining sequence. Production scheduling is then only a matter of incorporating time constraints and possibly the smoothing of stripping ratios.

Pit Limits Program

Parameters:

The grade matrix, the unit which was used to calculate geologic reserves, was also used in the pit limits program. This program, which uses only whole blocks, is based on the moving cone technique, and will calculate a pit outline under a given set of prices and costs. Input to the program consists of topography, prices, costs, pit slopes, a design cutoff, and location constraints used in the search for economic blocks.

Topography: Topography input may be the original surface or some intermediate surface such as a payback pit.

Prices: The copper price used is the gross selling price adjusted downward to compensate for mill recovery, and net of smelting, refining, and transportation charges.

Costs: The only costs used were the cost per ton mined, and the processing cost per ton milled, which includes administrative overhead.

Pit Slopes: Only one slope can be used. The slope of the moving cone is specified, and this fixes the slope of the pit wall.

Location Constraints: It is possible to delineate specific areas for coning, such as high grade core areas, rather than the whole matrix, by blocking out portions as desired.

Design Cutoff: The design cutoff grade is the grade of a block below which a cone will not be calculated using that block as a base. The concept of design cutoff is distinct from that of operating cutoff in that it merely determines an ultimate volume within which reserve tonnages at various operating cutoffs can be determined. The design cutoff must not be lower than the lowest feasible operating cutoff, since otherwise the stripping ratio is higher than it need be.

Program Method:

Once the input parameters have been set, the program will generate automatically, without a manual interface, a unique solution. The sequencing of events is the following: commencing with the uppermost bench specified, the grade blocks are scanned and sorted by grade with respect to the design cutoff. A cone having the required slope will be constructed above each block whose grade is above the design cutoff; each new cone will define an incremental volume outside of previously constructed cones and this represents the removal increment necessary to mine the particular block. Only those blocks whose centroids are within the cone are included in the increment. The net worth of the increment is calculated to see if it is economic. The grades for all the blocks above the design cutoff are summed and a total copper value is calculated. The cost of processing these blocks and the cost of mining all the blocks in the cone are subtracted from the total value, and since the result is based on one ton per block the net value is multiplied by the block tonnage factor. If the net worth is positive, the increment is removed, and the surface topography is changed accordingly. This process is repeated on the next bench using the modified surface. In this manner, it is certain that any block will not carry more waste than it has to, and that any block above the design cutoff that can show a positive net worth while supporting its cone increment will be mined. The moving cone sequence will either seek its own bottom, or will bottom at a specified elevation.

The iteration does not produce an "optimum" pit, but a series of pits can be generated by changing any of the input parameters. Unique project requirements can be more easily accommodated by this method, rather than by some more rigid or sophisticated mathematical algorithm. The use of whole blocks does not significantly affect the precision of tonnage calculations, providing the total volume is reasonably large, but pit walls usually require manual smoothing the degree of which may affect the accuracy of of a pit outline. The ultimate value of the design will decrease accordingly. In addition, the time dimension is ignored - it is possible that some cone whose base is very far below surface is economic, but it would have little impact on the profitability of the pit, because the discounted present value would be insignificant.

MINEABLE RESERVES

Details of structural geology were not studied, and the mechanical properties of rock, such as cohesion and compressive strength, suitable for rock mechanics studies, were not determined. In the absence of such information, an arbitrary pit slope selection of 45° was made. This is usually regarded as a conservative slope which should provide a reasonable factor of safety. It should be noted that no provision for pit roads was made at the pit design stage, but the final slope, including roads, will be 45°.

The gross copper price used was $0.60 per pound. After applying a 90% mill recovery and $0.11 smelting and refining charges the net price of copper applied to the pit design program was \$0.43 per pound. The mining cost used was \$0.40 per ton, and processing cost used was \$1.60 per ton milled. These particular values are used for the purpose of defining pit limits only and have no onward economic significance. The expression for the value of a cone constructed over a particular block would thus be for a stated design cutoff grade:

$$V_C = (S_G \times .01 \times 20 \times .43 - 1.60 \times N_O - 0.40 \times N_T)\ 8333$$
$$= (.086\ S_G - 1.6\ N_O - .4\ N_T)\ 8333 \text{ where}$$

V_C = Value of cone

S_G = Sum of the grades in the cone above the design cutoff

N_O = Number of blocks whose grades are above the design cutoff

N_T = Total number of blocks in the cone.

The design cutoff was varied by 0.05% copper decrements in the range from 0.50 design cutoff to 0.30 design cutoff. Thus five pit surfaces were outlined. The 0.50 pit was calculated with the original topography as the first surface, and successive pits, each having a lower design cutoff than the last, were calculated using the previous pit outline as the input surface.

Mineable reserves for each of the five pits are summarized in Table IV-2; this is the basis of the outline of a production schedule. An ultimate pit, and an intermediate or payback pit, were selected from Table IV-2 by calculating a design cutoff for each using previously established cost and price criteria and assuming a maximum allowable stripping ratio of 2:1.

<u>Calculation of Design Cutoff Grades</u>

$$G = \frac{T_C + C_m}{20 \times R_S \times R_C \times P} \text{ where}$$

G = Design cutoff grade

T_C = Total operating cost, including S & R charges, per ton

C_m = A minimum desired capital cost recovery per ton of mill feed

R_S = Smelter recovery = (concentrate grade less 1%) /concentrate grade = 29/30 = .967

Table IV-2

MINEABLE RESERVES FOR CHANGING DESIGN CUTOFF GRADES

OPERATING CUTOFF			.45	.40	.35	.30	.25
PIT NO.	DESIGN CUTOFF						
2	.50	ORE	15,866,032	20,824,167	25,982,294	29,707,145	32,623,695
		GRADE	.547	.516	.487	.467	.449
		WASTE	24,315,694	19,357,559	14,199,432	10,474,581	7,558,031
		S.R.	1.533	.930	.547	.353	.232
3	.45	ORE	17,907,617	23,932,376	30,298,788	34,856,939	38,523,459
		GRADE	.541	.510	.481	.460	.442
		WASTE	32,107,049	26,082,290	19,715,878	15,157,727	11,491,207
		S.R.	1.793	1.090	.651	.435	.298
4	.40	ORE	19,390,891	27,907,217	36,373,545	42,964,948	48,189,739
		GRADE	.536	.501	.470	.447	.428
		WASTE	46,139,821	37,623,495	29,157,167	22,565,764	17,340,973
		S.R.	2.379	1.348	.802	.525	.360
5	.35	ORE	20,265,856	30,373,785	41,673,333	50,656,307	57,989,347
		GRADE	.535	.496	.462	.437	.416
		WASTE	63,964,108	53,856,179	42,556,632	33,573,657	26,240,617
		S.R.	3.156	1.773	1.021	.663	.453
7	.30	ORE	23,315,734	36,831,860	56,906,057	76,480,274	90,779,702
		GRADE	.534	.491	.448	.415	.393
		WASTE	142,569,300	129,053,171	108,978,973	89,404,757	75,105,329
		S.R.	6.115	3.504	1.915	1.169	.827

R_C = Concentrator recovery = .9

P = Gross copper price, per pound = 60¢

$$
\begin{array}{lll}
\text{Processing} & = 1.60 & \\
\text{S \& R} & = \underline{0.70} & = 0.11/\text{lb. copper} \\
T_C & = 3.10 &
\end{array}
$$

ULTIMATE PIT - $C_m = 0 \quad G = \dfrac{3.10}{20 \times .967 \times .90 \times .6} = 0.30$

INTERMEDIATE PIT - $C_m = 1 \quad G = \dfrac{3.10 + 1.00}{20 \times .967 \times .90 \times .6} = 0.39$ SAY 0.40

Ultimate Pit

revenues

The ultimate pit is that pit in which marginal costs equal marginal reserves; capital cost requirements, and hence, capital cost recovery should have no bearing on its design. Using this principle, the design cutoff grade was calculated to be 0.30% copper. The corresponding pit, PIT 7, has mineable reserves of 90,780,000 tons averaging 0.393% copper at a 0.25% copper operating cutoff, with 75,105,000 tons of waste including overburden; the resulting stripping ratio is 0.827:1. These figures are unadjusted for pit smoothing and internal roadways. Bench reserve summaries for PIT 7 are shown in Table IV-3 and Table IV-4; Figure IV-1 is a smoothed pit ouline. The total mineable reserves shown in the production schedules differ from these because of changing operating cutoff during the planned mine life.

Payback Pit

While an ultimate pit is used to define mineable reserves, an intermediate pit is required for payback of capital. A payback pit may be defined as that pit in which mill feed at the cutoff provides a given minimum amount of capital return per ton. For a minimum capital return of $1.00 per ton of mill feed at the cutoff, the design cutoff grade is calculated as 0.40% copper. This defines the payback pit as PIT 4, which contains 27,907,220 tons averaging 0.501% copper at a 0.40% copper operating cutoff, with 37,623,500 tons of waste, giving a stripping ratio of 1.348:1. Tables IV-5 and IV-6 show bench

TABLE IV-3

RESERVES (PIT 7)   DESIGN CUTOFF = 0.30

BLOCK RESERVE SUMMARY BY BENCH **

BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3960	16666.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	16666.	16666.	16666.	16666.	16666.
		S.R	.000	.000	.000	.000	.000
3920	41665.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	41665.	41665.	41665.	41665.	41665.
		S.R	.000	.000	.000	.000	.000
3880	124995.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	124995.	124995.	124995.	124995.	124995.
		S.R	.000	.000	.000	.000	.000
3840	483314.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	483314.	483314.	483314.	483314.	483314.
		S.R	.000	.000	.000	.000	.000
3800	1074957.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	1074957.	1074957.	1074957.	1074957.	1074957.
		S.R	.000	.000	.000	.000	.000
3760	1449942.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	1449942.	1449942.	1449942.	1449942.	1449942.
		S.R	.000	.000	.000	.000	.000
3720	1774929.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	1774929.	1774929.	1774929.	1774929.	1774929.
		S.R	.000	.000	.000	.000	.000
3680	2116582.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	2116582.	2116582.	2116582.	2116582.	2116582.
		S.R	.000	.000	.000	.000	.000
3640	2483234.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	2483234.	2483234.	2483234.	2483234.	2483234.
		S.R	.000	.000	.000	.000	.000
3600	2816554.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	2816554.	2816554.	2816554.	2816554.	2816554.
		S.R	.000	.000	.000	.000	.000
3560	3233204.	ORE	33332.	8333.	0.	0.	0.
		GRADE	.285	.320	.000	.000	.000
		WASTE	3199872.	3224871.	3233204.	3233204.	3233204.
		S.R	96.000	387.000	.000	.000	.000
3520	3841513.	ORE	108329.	24999.	8333.	0.	0.
		GRADE	.283	.337	.370	.000	.000
		WASTE	3733184.	3816514.	3833180.	3841513.	3841513.
		S.R	34.462	152.667	460.000	.000	.000
3480	5766436.	ORE	374985.	133328.	0.	0.	0.
		GRADE	.284	.314	.000	.000	.000
		WASTE	5391451.	5633108.	5766436.	5766436.	5766436.
		S.R	14.378	42.250	.000	.000	.000
3440	9891271.	ORE	2233244.	1308281.	566644.	258323.	158327.
		GRADE	.337	.386	.475	.608	.731
		WASTE	7658027.	8582990.	9324627.	9632948.	9732944.
		S.R	3.429	6.561	16.456	37.290	61.474
3400	15141061.	ORE	5124795.	3858179.	2566564.	1649934.	1066624.
		GRADE	.376	.412	.459	.509	.557
		WASTE	10016266.	11282882.	12574497.	13491127.	14074437.
		S.R	1.954	2.924	4.899	8.177	13.195
3360	13949442.	ORE	5933096.	4399824.	3166540.	2074917.	1366612.
		GRADE	.375	.413	.449	.491	.527
		WASTE	8016346.	9549618.	10782902.	11874525.	12582830.
		S.R	1.351	2.170	3.405	5.723	9.207

TABLE IV-3 (CONTINUED)

RESERVES (PIT 7) DESIGN CUTOFF = 0.30

BLOCK RESERVE SUMMARY BY BENCH **

BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3320	12899484.	ORE	6563070.	5174793.	3760516.	2491567.	1816594.
		GRADE	.384	.414	.450	.491	.518
		WASTE	6316414.	7724691.	9132968.	10407917.	11082890.
		S.R	.959	1.493	2.425	4.177	6.101
3280	11807861.	ORE	6949722.	5899764.	4433156.	2983214.	1824927.
		GRADE	.398	.420	.453	.492	.541
		WASTE	4858139.	5908097.	7374705.	8824047.	9982934.
		S.R	.699	1.001	1.664	2.958	5.470
3240	10874505.	ORE	7241377.	6416410.	4883138.	3324867.	2358239.
		GRADE	.410	.428	.461	.505	.538
		WASTE	3633188.	4458155.	5991427.	7549698.	8516326.
		S.R	.502	.695	1.227	2.271	3.611
3200	9832940.	ORE	7083050.	6183086.	4816474.	3516526.	2416570.
		GRADE	.412	.433	.466	.502	.540
		WASTE	2749890.	3649854.	5016466.	6316414.	7416370.
		S.R	.388	.590	1.042	1.796	3.069
3160	8932976.	ORE	7083050.	5916430.	4466488.	3266536.	2349906.
		GRADE	.406	.432	.468	.505	.540
		WASTE	1849926.	3016546.	4466488.	5666440.	6583070.
		S.R	.261	.510	1.000	1.735	2.801
3120	7991347.	ORE	6558071.	5583110.	4233164.	3008213.	2133248.
		GRADE	.403	.425	.459	.495	.527
		WASTE	1433276.	2408237.	3758183.	4983134.	5858099.
		S.R	.219	.431	.888	1.657	2.746
3080	7041385.	ORE	5933096.	5074797.	4058171.	2583230.	1683266.
		GRADE	.400	.422	.447	.490	.529
		WASTE	1108289.	1966588.	2983214.	4458155.	5358119.
		S.R	.187	.388	.735	1.726	3.183
3040	6291415.	ORE	5199792.	4433156.	3416530.	2258243.	1341613.
		GRADE	.396	.418	.446	.486	.531
		WASTE	1091623.	1858259.	2874885.	4033172.	4949802.
		S.R	.210	.419	.841	1.766	3.689
3000	5408117.	ORE	4608149.	3958175.	3033212.	2049918.	1141621.
		GRADE	.395	.416	.445	.482	.531
		WASTE	799968.	1449942.	2374905.	3358199.	4266496.
		S.R	.174	.366	.783	1.638	3.737
2960	4058147.	ORE	4133168.	3016522.	2049894.	1624935.	783302.
		GRADE	.385	.401	.431	.469	.527
		WASTE	524979.	1041625.	2008253.	3033212.	3874845.
		S.R	.127	.288	.758	1.867	4.947
2920	3941509.	ORE	3733184.	3274869.	2291575.	1208285.	633308.
		GRADE	.374	.388	.417	.461	.501
		WASTE	208325.	666640.	1649934.	2733224.	3308201.
		S.R	.056	.204	.720	2.262	5.224
2880	3241537.	ORE	3174873.	2966548.	2116582.	1024959.	349986.
		GRADE	.374	.381	.403	.439	.479
		WASTE	66664.	274989.	1124955.	2216578.	2891551.
		S.R	.021	.093	.531	2.163	8.262
2840	2749890.	ORE	2733224.	2591563.	1908257.	1008293.	383318.
		GRADE	.383	.388	.411	.448	.502
		WASTE	16666.	158327.	841633.	1741597.	2366572.
		S.R	.006	.061	.441	1.727	6.174
2800	2266576.	ORE	2224911.	2083250.	1683266.	866632.	566644.
		GRADE	.401	.409	.429	.485	.520
		WASTE	41665.	183326.	583310.	1399944.	1699932.
		S.R	.019	.088	.347	1.615	3.000
2760	1749930.	ORE	1741597.	1649934.	1341613.	791635.	433316.
		GRADE	.411	.418	.440	.490	.552
		WASTE	8333.	99996.	408317.	958295.	1316614.
		S.R	.005	.061	.304	1.211	3.038
2720	1283282.	ORE	1283282.	1216618.	974961.	566644.	316654.
		GRADE	.412	.419	.443	.499	.562
		WASTE	0.	66664.	308321.	716638.	966628.
		S.R	.000	.055	.316	1.265	3.053
2680	708305.	ORE	708305.	708305.	524979.	274989.	191659.
		GRADE	.409	.409	.439	.501	.533
		WASTE	0.	0.	183326.	433316.	516646.
		S.R	.000	.000	.349	1.576	2.696

TABLE IV-4

RESERVES (PIT 4) · DESIGN CUTOFF = 0.40

BLOCK RESERVE SUMMARY BY BENCH **

BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3960	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3920	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3880	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3840	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3800	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3760	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3720	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3680	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3640	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
3600	8333.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	8333.	8333.	8333.	8333.	8333.
		S.R	.000	.000	.000	.000	.000
3560	158327.	ORE	33332.	8333.	0.	0.	0.
		GRADE	.285	.320	.000	.000	.000
		WASTE	124995.	149994.	158327.	158327.	158327.
		S.R	3.750	18.000	.000	.000	.000
3520	466648.	ORE	83330.	24999.	8333.	0.	0.
		GRADE	.290	.337	.370	.000	.000
		WASTE	383318.	441649.	458315.	466648.	466648.
		S.R	4.600	17.667	55.000	.000	.000
3480	1566604.	ORE	166660.	74997.	0.	0.	0.
		GRADE	.290	.313	.000	.000	.000
		WASTE	1399944.	1491607.	1566604.	1566604.	1566604.
		S.R	8.400	19.889	.000	.000	.000
3440	5174793.	ORE	1816594.	1191619.	558311.	249990.	158327.
		GRADE	.350	.393	.476	.615	.731
		WASTE	3358199.	3983174.	4616482.	4924803.	5016466.
		S.R	1.849	3.343	8.269	19.700	31.684
3400	6932976.	ORE	4508153.	3549858.	2566564.	1649934.	1066624.
		GRADE	.388	.420	.459	.509	.557
		WASTE	4424823.	5383118.	6366412.	7283042.	7866352.
		S.R	.982	1.516	2.481	4.414	7.375
3360	6041345.	ORE	4983134.	3958175.	3083210.	2074917.	1366612.
		GRADE	.390	.422	.451	.491	.527
		WASTE	3058211.	4083170.	4958135.	5966428.	6074733.
		S.R	.614	1.032	1.508	2.876	4.684

TABLE IV-4 (CONTINUED)

RESERVES (PIT 4) DESIGN CUTOFF = 0.40

BLOCK RESERVE SUMMARY BY BENCH **

BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3320	7224711.	ORE	5308121.	4474821.	3499860.	2449902.	1808261.
		GRADE	.402	.426	.456	.493	.519
		WASTE	1916590.	2749890.	3724851.	4774809.	5416450.
		S.R	.361	.615	1.064	1.949	2.995
3280	6333080.	ORE	5066464.	4691479.	3949842.	2916550.	1816594.
		GRADE	.428	.441	.463	.494	.542
		WASTE	1266616.	1641601.	2383236.	3416530.	4516486.
		S.R	.250	.350	.603	1.171	2.486
3240	5524779.	ORE	4808141.	4449822.	3941509.	3133208.	2299908.
		GRADE	.448	.462	.481	.509	.540
		WASTE	716638.	1074957.	1583270.	2391571.	3224871.
		S.R	.149	.242	.402	.763	1.402
3200	4749810.	ORE	4383158.	4049838.	3608189.	2841553.	2116582.
		GRADE	.451	.466	.484	.515	.548
		WASTE	366652.	699972.	1141621.	1908257.	2633228.
		S.R	.084	.173	.315	.672	1.244
3160	4033172.	ORE	3858179.	3633188.	3208205.	2558231.	1899924.
		GRADE	.451	.462	.481	.509	.542
		WASTE	174993.	399984.	824967.	1474941.	2133248.
		S.R	.045	.110	.257	.577	1.123
3120	3399864.	ORE	3283202.	3149874.	2766556.	2208245.	1633268.
		GRADE	.447	.454	.472	.498	.526
		WASTE	116662.	249990.	633308.	1191619.	1766596.
		S.R	.036	.079	.229	.540	1.082
3080	2699892.	ORE	2674893.	2574897.	2333240.	1983254.	1341613.
		GRADE	.455	.462	.476	.495	.531
		WASTE	24999.	124995.	366652.	716638.	1358279.
		S.R	.009	.049	.157	.361	1.012
3040	2183246.	ORE	2183246.	2108249.	1999920.	1624935.	1149954.
		GRADE	.464	.471	.479	.504	.537
		WASTE	0.	74997.	183326.	558311.	1033292.
		S.R	.000	.036	.092	.344	.899
3000	1774929.	ORE	1774929.	1766596.	1716598.	1449942.	1024959.
		GRADE	.478	.479	.484	.504	.537
		WASTE	0.	8333.	58331.	324987.	749970.
		S.R	.000	.005	.034	.224	.732
2960	1366612.	ORE	1366612.	1366612.	1308281.	1191619.	766636.
		GRADE	.472	.472	.479	.488	.527
		WASTE	0.	0.	58331.	174993.	599976.
		S.R	.000	.000	.045	.147	.783
2920	974961.	ORE	974961.	974961.	958295.	849966.	599976.
		GRADE	.465	.465	.467	.479	.504
		WASTE	0.	0.	16666.	124995.	374985.
		S.R	.000	.000	.017	.147	.625
2880	599976.	ORE	599976.	599976.	566644.	483314.	224991.
		GRADE	.434	.434	.440	.452	.485
		WASTE	0.	0.	33332.	116662.	374985.
		S.R	.000	.000	.059	.241	1.667
2840	233324.	ORE	233324.	233324.	224991.	183326.	91663.
		GRADE	.440	.440	.445	.462	.515
		WASTE	0.	0.	8333.	49998.	141661.
		S.R	.000	.000	.037	.273	1.545
2800	74997.	ORE	74997.	74997.	66664.	49998.	24999.
		GRADE	.422	.422	.433	.453	.487
		WASTE	0.	0.	8333.	24999.	49998.
		S.R	.000	.000	.125	.500	2.000
2760	8333.	ORE	8333.	8333.	8333.	8333.	0.
		GRADE	.410	.410	.410	.410	.000
		WASTE	0.	0.	0.	0.	8333.
		S.R	.000	.000	.000	.000	.000
	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000
	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	0.	0.	0.	0.	0.
		S.R	.000	.000	.000	.000	.000

TABLE IV-5

RESERVES (PIT 7) DESIGN CUTOFF = 0.30

BLOCK RESERVE -CUMULATIVE SUMMARY*

BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3960	16666.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	16666.	16666.	16666.	16666.	16666.
		S.R	.000	.000	.000	.000	.000
3920	58331.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	58331.	58331.	58331.	58331.	58331.
		S.R	.000	.000	.000	.000	.000
3880	183326.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	183326.	183326.	183326.	183326.	183326.
		S.R	.000	.000	.000	.000	.000
3840	666640.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	666640.	666640.	666640.	666640.	666640.
		S.R	.000	.000	.000	.000	.000
3800	1741597.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	1741597.	1741597.	1741597.	1741597.	1741597.
		S.R	.000	.000	.000	.000	.000
3760	3191539.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	3191539.	3191539.	3191539.	3191539.	3191539.
		S.R	.000	.000	.000	.000	.000
3720	4966468.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	4966468.	4966468.	4966468.	4966468.	4966468.
		S.R	.000	.000	.000	.000	.000
3680	7083050.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	7083050.	7083050.	7083050.	7083050.	7083050.
		S.R	.000	.000	.000	.000	.000
3640	9566284.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	9566284.	9566284.	9566284.	9566284.	9566284.
		S.R	.000	.000	.000	.000	.000
3600	12382838.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	12382838.	12382838.	12382838.	12382838.	12382838.
		S.R	.000	.000	.000	.000	.000
3560	15616042.	ORE	33332.	8333.	0.	0.	0.
		GRADE	.285	.320	.000	.000	.000
		WASTE	15582710.	15607709.	15616042.	15616042.	15616042.
		S.R	467.500	1873.000	.000	.000	.000
3520	19457555.	ORE	141661.	33332.	8333.	0.	0.
		GRADE	.284	.333	.370	.000	.000
		WASTE	19315894.	19424223.	19449222.	19457555.	19457555.
		S.R	136.353	582.750	2334.000	.000	.000
3480	25223991.	ORE	516646.	166660.	8333.	0.	0.
		GRADE	.284	.318	.370	.000	.000
		WASTE	24707345.	25057331.	25215658.	25223991.	25223991.
		S.R	47.823	150.350	3026.000	.000	.000
3440	35115262.	ORE	2749890.	1474941.	574977.	258323.	158327.
		GRADE	.327	.378	.474	.608	.731
		WASTE	32365372.	33640321.	34540285.	34856939.	34956935.
		S.R	11.770	22.808	60.072	134.935	220.789
3400	50256323.	ORE	7874685.	5333120.	3141541.	1908257.	1224951.
		GRADE	.359	.402	.461	.522	.580
		WASTE	42381638.	44923203.	47114782.	48348066.	49031372.
		S.R	5.382	8.423	14.997	25.336	40.027
3360	64205765.	ORE	13807781.	9732944.	6308081.	3983174.	2591563.
		GRADE	.366	.407	.455	.506	.552
		WASTE	50397984.	54472821.	57897684.	60222591.	61614202.
		S.R	3.650	5.597	9.178	15.119	23.775

TABLE IV-5 (CONTINUED)

RESERVES (PIT 7) DESIGN CUTOFF = 0.30

BLOCK RESERVE -CUMULATIVE SUMMARY*

BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3320	77105249.	ORE	20390851.	14907737.	10074597.	6474741.	4408157.
		GRADE	.372	.409	.453	.500	.538
		WASTE	56714398.	62197512.	67030652.	70630508.	72697092.
		S.R	2.781	4.172	6.653	10.909	16.491
3280	88913110.	ORE	27340573.	20807501.	14507753.	9457955.	6233084.
		GRADE	.378	.412	.453	.498	.539
		WASTE	61572537.	68105609.	74405357.	79455155.	82680026.
		S.R	2.252	3.273	5.129	8.401	13.265
3240	99787675.	ORE	34581950.	27223911.	19390891.	12782822.	8591323.
		GRADE	.385	.416	.455	.500	.539
		WASTE	65205725.	72563764.	80396784.	87004853.	91196352.
		S.R	1.886	2.665	4.146	6.806	10.615
3200	109620614.	ORE	41665000.	33406997.	24207365.	16299348.	11007893.
		GRADE	.390	.419	.457	.500	.539
		WASTE	67955615.	76213618.	85413250.	93321267.	98612722.
		S.R	1.631	2.281	3.528	5.725	8.958
3160	118553591.	ORE	48748050.	39323428.	28673853.	19565884.	13357799.
		GRADE	.392	.421	.459	.501	.539
		WASTE	69805541.	79230164.	89879738.	98987707.	105195792.
		S.R	1.432	2.015	3.135	5.059	7.875
3120	126544937.	ORE	55306121.	44906537.	32907017.	22574097.	15491047.
		GRADE	.393	.422	.459	.500	.537
		WASTE	71238817.	81638401.	93637921.	103970841.	111053891.
		S.R	1.288	1.818	2.846	4.606	7.169
3080	133586323.	ORE	61239217.	49981334.	36965188.	25157327.	17174313.
		GRADE	.394	.422	.458	.499	.537
		WASTE	72347106.	83604989.	96621135.	108428995.	116412009.
		S.R	1.181	1.673	2.614	4.310	6.778
3040	139877740.	ORE	66439009.	54414490.	40381718.	27415570.	18515926.
		GRADE	.394	.421	.457	.498	.536
		WASTE	73438729.	85463248.	99496020.	112462167.	121361811.
		S.R	1.105	1.571	2.464	4.102	6.554
3000	145285850.	ORE	71047158.	58372665.	43414930.	29465486.	19657547.
		GRADE	.394	.421	.456	.497	.536
		WASTE	74238697.	86913190.	101870924.	115820366.	125628307.
		S.R	1.045	1.489	2.346	3.931	6.391
2960	149944000.	ORE	75180326.	61989187.	46064824.	31090423.	20440849.
		GRADE	.394	.420	.454	.495	.536
		WASTE	74763676.	87954815.	103879176.	118853579.	129503153.
		S.R	.994	1.419	2.255	3.823	6.336
2920	153885510.	ORE	78913510.	65264056.	48356399.	32298706.	21074157.
		GRADE	.393	.418	.453	.494	.535
		WASTE	74972001.	88621455.	105529112.	121586803.	132811354.
		S.R	.950	1.358	2.182	3.764	6.302
2880	157127050.	ORE	82088383.	68230604.	50472981.	33323667.	21424143.
		GRADE	.392	.417	.450	.492	.534
		WASTE	75038665.	88896444.	106654067.	123803382.	135702910.
		S.R	.914	1.303	2.113	3.715	6.334
2840	159876940.	ORE	84821607.	70822167.	52381238.	34331960.	21807461.
		GRADE	.392	.416	.449	.491	.533
		WASTE	75055331.	89054771.	107495700.	125544977.	138069480.
		S.R	.885	1.257	2.052	3.657	6.331
2800	162143510.	ORE	87046518.	72905417.	54064504.	35198592.	22374105.
		GRADE	.392	.415	.448	.491	.533
		WASTE	75096996.	89238097.	108079010.	126944922.	139769410.
		S.R	.863	1.224	1.999	3.607	6.247
2760	163893450.	ORE	88788115.	74555351.	55406117.	35990227.	22807421.
		GRADE	.392	.415	.448	.491	.533
		WASTE	75105329.	89338093.	108487327.	127903217.	141086030.
		S.R	.846	1.198	1.958	3.554	6.186
2720	165176730.	ORE	90071397.	75771969.	56381078.	36556871.	23124075.
		GRADE	.393	.415	.448	.491	.534
		WASTE	75105329.	89404757.	108795647.	128619856.	142052650.
		S.R	.834	1.180	1.930	3.518	6.143
2680	165885030.	ORE	90779702.	76480274.	56908057.	36831860.	23315734.
		GRADE	.393	.415	.448	.491	.534
		WASTE	75105329.	89404757.	108976973.	129053171.	142569300.
		S.R	.827	1.169	1.915	3.504	6.115

TABLE IV-6

RESERVES (PIT 4)   DESIGN CUTOFF = 0.40

BLOCK RESERVE -CUMULATIVE SUMMARY*

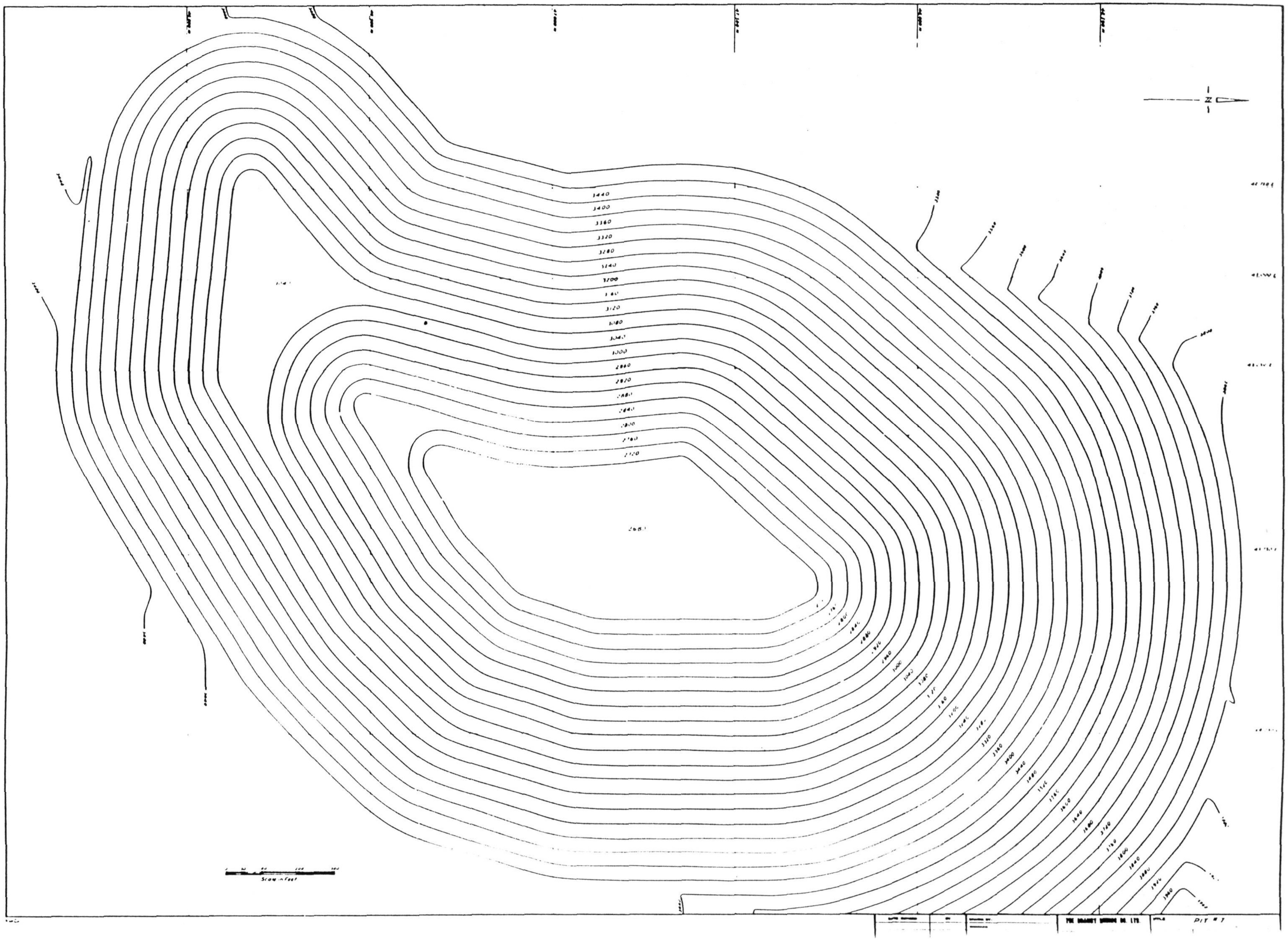
BENCH	TOTAL TONS		CUT-OFF GRADES .250	.300	.350	.400	.450
3960	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3920	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3880	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3840	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3800	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3760	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3720	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3680	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3640	0.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	-0.	-0.	-0.	-0.	-0.
		S.R	.000	.000	.000	.000	.000
3600	8333.	ORE	0.	0.	0.	0.	0.
		GRADE	.000	.000	.000	.000	.000
		WASTE	8333.	8333.	8333.	8333.	8333.
		S.R	.000	.000	.000	.000	.000
3560	166660.	ORE	33332.	8333.	0.	0.	0.
		GRADE	.265	.320	.000	.000	.000
		WASTE	133328.	158327.	166660.	166660.	166660.
		S.R	4.000	19.000	.000	.000	.000
3520	633308.	ORE	116662.	33332.	8333.	0.	0.
		GRADE	.289	.333	.370	.000	.000
		WASTE	516646.	599976.	624975.	633308.	633308.
		S.R	4.429	18.000	75.000	.000	.000
3480	2199912.	ORE	283322.	108329.	8333.	0.	0.
		GRADE	.289	.319	.370	.000	.000
		WASTE	1916590.	2091583.	2191579.	2199912.	2199912.
		S.R	6.765	19.308	263.000	.000	.000
3440	7374705.	ORE	2099916.	1299948.	566644.	249990.	158327.
		GRADE	.342	.386	.475	.615	.731
		WASTE	5274789.	6074757.	6808061.	7124715.	7216378.
		S.R	2.512	4.673	12.015	28.500	45.579
3400	16307681.	ORE	6608069.	4849806.	3133208.	1899924.	1224951.
		GRADE	.373	.411	.461	.523	.580
		WASTE	9699612.	11457875.	13174473.	14407757.	15082730.
		S.R	1.468	2.363	4.205	7.583	12.313
3360	24349026.	ORE	11591203.	8807981.	6216418.	3974841.	2591563.
		GRADE	.381	.416	.456	.506	.552
		WASTE	12757823.	15541045.	18132608.	20374185.	21757463.
		S.R	1.101	1.764	2.917	5.126	8.395

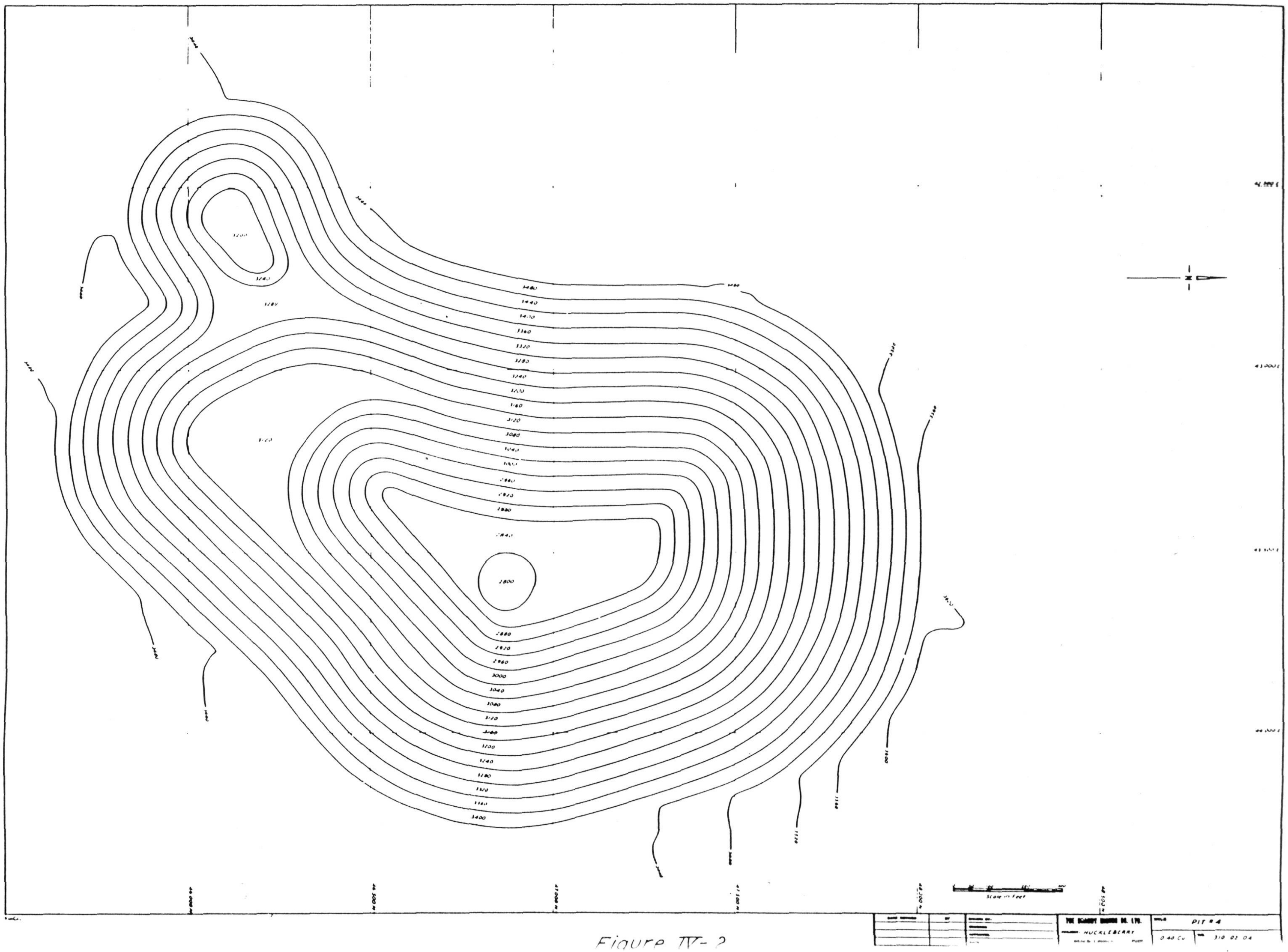
TABLE IV-6 (CONTINUED)

RESERVES (PIT 4) DESIGN CUTOFF = 0.40

BLOCK RESERVE -CUMULATIVE SUMMARY*

BENCH	TOTAL TONS		CUT-OFF GRADES				
			.250	.300	.350	.400	.450
3320	31573737.	ORE	16899324.	13282802.	9716278.	6424743.	4399824.
		GRADE	.387	.419	.456	.501	.538
		WASTE	14674413.	18290935.	21857459.	25148994.	27173913.
		S.R	.866	1.377	2.250	3.914	6.176
3280	37906817.	ORE	21965788.	17974281.	13666120.	9341293.	6216418.
		GRADE	.397	.425	.458	.499	.539
		WASTE	15941029.	19932536.	24240697.	28565524.	31690399.
		S.R	.726	1.109	1.774	3.058	5.098
3240	43431596.	ORE	26773929.	22424103.	17607629.	12474501.	8516326.
		GRADE	.406	.432	.463	.501	.539
		WASTE	16657667.	21007493.	25823967.	30957095.	34915270.
		S.R	.622	.937	1.467	2.482	4.100
3200	48181406.	ORE	31157087.	26473941.	21215818.	15316054.	10632908.
		GRADE	.412	.438	.467	.504	.541
		WASTE	17024319.	21707465.	26965588.	32865352.	37548498.
		S.R	.546	.820	1.271	2.146	3.531
3160	52214578.	ORE	35015266.	30107129.	24424023.	17874285.	12532832.
		GRADE	.417	.440	.469	.505	.541
		WASTE	17199312.	22107449.	27790555.	34340293.	39681746.
		S.R	.491	.734	1.138	1.921	3.166
3120	55614442.	ORE	38298468.	33257003.	27190579.	20082530.	14166100.
		GRADE	.419	.442	.469	.504	.540
		WASTE	17315974.	22357439.	28423863.	35531912.	41448342.
		S.R	.452	.672	1.045	1.769	2.926
3080	58314334.	ORE	40973361.	35831900.	29523819.	22065784.	15507713.
		GRADE	.422	.443	.469	.503	.539
		WASTE	17340973.	22482434.	28790515.	36248550.	42806621.
		S.R	.423	.627	.975	1.643	2.760
3040	60497580.	ORE	43156607.	37940149.	31523739.	23690719.	16657667.
		GRADE	.424	.445	.470	.503	.539
		WASTE	17340973.	22557431.	28973841.	36806861.	43839913.
		S.R	.402	.595	.919	1.554	2.632
3000	62272509.	ORE	44931536.	39706745.	33240337.	25140661.	17682626.
		GRADE	.426	.446	.471	.503	.539
		WASTE	17340973.	22565764.	29032172.	37131848.	44589883.
		S.R	.386	.568	.873	1.477	2.522
2960	63639121.	ORE	46298148.	41073357.	34548618.	26332280.	18449262.
		GRADE	.427	.447	.471	.503	.538
		WASTE	17340973.	22565764.	29090503.	37306841.	45189859.
		S.R	.375	.549	.842	1.417	2.449
2920	64614082.	ORE	47273109.	42048318.	35506913.	27182246.	19049238.
		GRADE	.428	.448	.471	.502	.537
		WASTE	17340973.	22565764.	29107169.	37431836.	45564844.
		S.R	.367	.537	.820	1.377	2.392
2880	65214058.	ORE	47873085.	42648295.	36073557.	27665560.	19274229.
		GRADE	.428	.447	.470	.501	.536
		WASTE	17340973.	22565764.	29140501.	37548498.	45939829.
		S.R	.362	.529	.808	1.357	2.383
2840	65447382.	ORE	48106410.	42881618.	36298548.	27848886.	19365892.
		GRADE	.428	.447	.470	.501	.536
		WASTE	17340973.	22565764.	29148834.	37598496.	46081490.
		S.R	.360	.526	.803	1.350	2.380
2800	65522379.	ORE	48181406.	42956615.	36365213.	27898884.	19390891.
		GRADE	.428	.447	.470	.501	.536
		WASTE	17340973.	22565764.	29157167.	37623495.	46131488.
		S.R	.360	.525	.802	1.349	2.379
2760	65530712.	ORE	48189739.	42964948.	36373545.	27907217.	19390891.
		GRADE	.428	.447	.470	.501	.536
		WASTE	17340973.	22565764.	29157167.	37623495.	46139821.
		S.R	.360	.525	.802	1.348	2.379
2720	65530712.	ORE	48189739.	42964948.	36373545.	27907217.	19390891.
		GRADE	.428	.447	.47	.501	4y536
		WASTE	17340973.	22565764.	29157167.	37623495.	46139821.
		S.R	.360	.525	.802	1.348	2.379


3440
3400
3360
3320
3280
3240
3200
Scale in Feet
PIT # 7



PIT # 4
HUCKLEBERRY
0.40 Cu
310 02 04
Scale in Feet
3400
3360
3320
3280
3240
3200
3160
3120
3080
3040
3000
2960
2920
2880
2840
2800

Figure IV-2

reserve summaries for PIT 4; Figure IV-2 is the corresponding smoothed outline. Tonnage increments in the high cutoff ranges as shown in Table IV-2 are not large. This indicates that the choice of a design cutoff for the payback pit is not critical to the profitability of the project.

DRILLING DENSITY

Systematic drilling, once the magnitude of the mineralized zone had been indicated, was done to provide vertical holes on a 200' square grid pattern. Most of the early holes did not fall on grid corner locations. Three of these were used to make comparisons between calculated grade values from surrounding drill holes and actual assay values at the same locations, (Table IV-7). The calculation of grades was done using bench composites from the closest surrounding holes within 300', up to a maximum of eight holes and the calculated bench grade was the weighted average of the surrounding bench grades; the weighting factor being the inverse square of the distance.

Statistical tests applied to these data all require some basic assumption, such as assigning levels of confidence or type of distributions. Therefore; as decision making criteria they only mask the subjective nature of the data interpretation. The tabulated data suggest that the calculated values are not significantly comparable to the real values, hence, the drilling density is not high enough. However, a valid test of the drilling density can only be made when the margins of the mineralized zone are defined and additional internal drilling is evaluated on a cumulative basis; i.e. reserves are re-calculated as one or more internal holes are drilled. If additional internal drilling significantly changes the tonnage and/or the grade of the reserves then further internal drilling is warranted. If the tonnage and/or grade of the reserves is not changed; then the drilling density is adequate. An alternative test of the drilling density can be made by subtracting one or more holes, which are not at 200' grid corner locations, from the total data and recalculating the reserves without these holes. The effect of these holes on tonnages and grades is seen by comparing the recalculated reserve figures with the current reserve estimate.

Table IV-7

ACTUAL AND CALCULATED BENCH COMPOSITE GRADES - %CU

BENCH	73-3		72-1		DDH-14	
TOE ELEV.	ACTUAL	CALCULATED	ACTUAL	CALCULATED	ACTUAL	CALCULATED
3480.0	.232					
3440.0	.184					
3400.0	.148	.183	.265	.166		
3360.0	.058	.280	.384	.247		
3320.0	.377	.328	.207	.306	.208	.371
3280.0	.530	.316	.137	.459	.209	.426
3240.0	.509	.316	.745	.499	.313	.355
3200.0	.386	.299	.416	.410	.356	.444
3160.0	.278	.238	.269	.397	.422	.430
3120.0	.549	.219	.197	.404	.385	.463
3080.0	.439	.348	.314	.379	.246	.529
3040.0	.471	.400	.361	.332	.256	.346
3000.0	.461	.384	.375	.319	.491	.384
2960.0	.553	.301	.451	.300	.298	.292
2920.0	.344	.298	.236	.375	.230	.365
2880.0	.400	.267	.634	.321	.164	.349
2840.0	.354	.268	.455	.355	.323	.328
2800.0	.585	.243	.580	.402	.315	.471
2760.0			.481	.376	.309	.425
2720.0			.425	.286	.310	.351
2680.0			.542	.284	.319	.318
2640.0			.279	.223	.346	.546
2600.0					.432	.226
2560.0					.431	.259
2520.0					.625	.350
AVERAGE GRADE	.401	.293	.388	.342	.328	.382

FURTHER WORK

Recalculation of reserves to test the drilling density can be done prior to additional marginal drilling, and this will be an integral part of continuing development. The first aim of additional drilling will be to define the margins of the mineral reserve where they are unknown. A series of peripheral holes, with some additional internal definition drilling, is proposed at a cost in the order of $200,000.

V

METALLURGICAL PROJECTIONS

SCOPE OF STUDIES

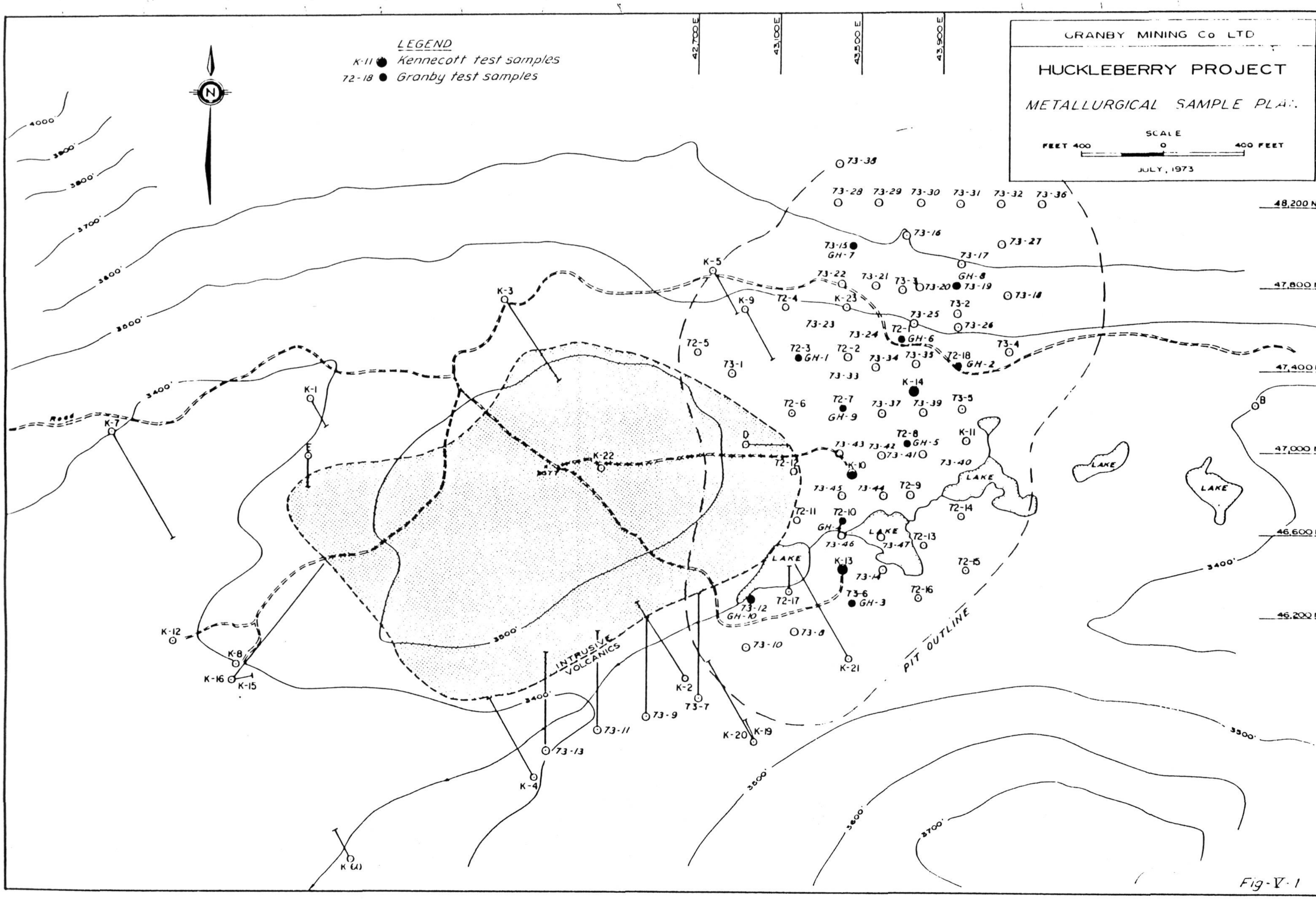
Commencing in July 1973, metallurgical test work was carried out on ten core samples designated GH 1 to GH 10. The locations of these drill holes are shown on the Drill Hole Plan, Figure V-1.

The purposes of the investigations were to establish the grinding characteristics, copper recovery and concentrate grades that may be expected in an operating plant. Flotation tests were conducted on samples GH 1 to GH 4, and grindability estimates and comparisons were carried out on all ten samples in the laboratories of Granisle Copper. A quantity of "gouge" material was made available to determine its effects upon flotation performance. The Galigher Company in Salt Lake City performed a limited amount of metallurgical testing on portions of samples GH 3 and GH 4, commencing in November, 1973. Allis-Chalmers Research Centre determined Bond Grindability Indices for rod milling and ball milling, and Abrasion Indices on samples GH 5 and GH 6. Davis Tube tests were carried out by Britton Research Limited, Vancouver, on representative flotation tailings from samples GH 1 to GH 4 to ascertain the percentages of magnetite. Additionally, Technical Report 71-10 prepared by Kennecott Research Centre in Salt Lake City, covering metallurgical studies on the ore in 1970, served as a useful reference.

The investigation of molybdenite recovery was not undertaken. The projections reported by Kennecott are used for approximating molybdenite performance, and these are supported in part by recoveries in copper flotation calculated from assays for typical products of the Granisle tests.

MINERALOGY

The host rock is hornfelsed volcanic composed of a very fine aggregate of quartz, biotite, feldspar, amphibole, and variable amounts of magnetite, the last being finely disseminated and in aggregates up to a half inch in size.


GRANBY MINING Co LTD
HUCKLEBERRY PROJECT
METALLURGICAL SAMPLE PLAN
SCALE
FEET 400 0 400 FEET
JULY, 1973
LEGEND
K-11 Kennecott test samples
72-18 Granby test samples
42,700 E
43,100 E
43,500 E
43,900 E
48,200 N
47,800 N
47,400 N
47,000 N
46,600 N
46,200 N
4000'
3900'
3800'
3700'
3600'
3500'
3400'
3300'
3600'
3700'
Road
INTRUSIVE
VOLCANICS
PIT OUTLINE
LAKE
LAKE
LAKE
LAKE
LAKE
K-1
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K-3
K-4
K-5
K-7
K-8
K-9
K-10
K-11
K-12
K-13
K-14
K-15
K-16
K-19
K-20
K-21
K-22
K-23
72-1
72-2
72-3
72-4
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72-18
GH-1
GH-2
GH-3
GH-5
GH-6
GH-7
GH-8
GH-9
GH-10
73-1
73-2
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73-4
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73-47

Fig-V-1

The chief copper mineral is chalcopyrite, which occurs principally in small fractures with quartz, and to a much lesser extent disseminated in host rock. Molybdenite also occurs in fractures, apparently mostly of later deposition than chalcopyrite.

Microscopic examination of concentrates produced from GH 1 and GH 2 revealed the following opaque minerals in descending order of abundance: chalcopyrite, molybdenite, pyrite, magnetite, sphalerite, pyrrhotite, bornite, chalcocite, marcasite and ilmenite. Transparent gangue minerals, also in descending order of occurrence, included quartz, chlorite, biotite, sericite, actinolite, calcite and plagioclase.

In the concentrates, chalcopyrite was largely free. Quartz was the major diluent, followed by molybdenite, pyrite, magnetite, sphalerite, chalcocite and biotite. There was considerable interlocking of quartz and chalcopyrite above 400 mesh, and a few inclusions of chalcopyrite, pyrite and molybdenite about 10 microns in size were observed in quartz.

METALLURGICAL PROJECTIONS

The predicted metal balance for an average 0.417 mill feed is as follows:

Product	% Weight	% Copper	% Distribution
Heads	100.00	0.41	100.0
Concentrates	1.43	27.0	94.1
Tailings	98.57	.025	5.9

The indicated recoveries for heads of 0.35% and 0.47% copper are 93.5% and 94.7%, respectively

TESTING PROGRAM

SUMMARY OF RESULTS

The results of the Granisle locked-cycle flotation tests on each of the four samples are as follows:

Sample No.	Calculated Test Head	Concentrate Grade	Tailings Assay	Calculated Recovery	Fineness % -200
GH 1	0.443	27.28	.025	94.45	57.6
GH 2	0.298	26.46	.018	94.03	67.3
GH 3	0.252	26.44	.020	92.13	52.8
GH 4	0.696	27.14	.031	95.66	56.6

The projections from the locked-cycle tests were determined by using the calculated test heads and the average of the concentrates and tailings from the last 3 cycles of a 6-cycle test. It should be noted that a finer grind was employed in test GH 2. In order to compare copper recoveries at a similar fineness of grind, it was estimated that the 94.03% recovery in test GH 2 at 67.3% -200 mesh would decline to 92.8% at 55% -200 mesh. This percentage is used in the recovery-head grade relationship shown graphically in Figure V-2. On this basis, cycle tests at a fineness of 55% -200 mesh indicated an average overall performance of 94.1% copper recovery in concentrates grading 26.8% from a 0.42% mill feed. Typical concentrates from each of the four lots averaged 0.059% oz. gold and 1.94 oz. silver per ton, with the gold ranging from 0.030 to 0.084 and silver 1.54 to 2.40 ozs./ton.

In general these flotation results were supported by the test work conducted by the Galigher Company and Kennecott Research Centre.

With the exception of mercury, which was reported as 2 ppm in concentrates from sample GH 2, but less than 1 ppm in concentrates from the other samples, there were no deleterious constituents reported in semi-quantitative spectrographic analyses.

Owing to the considerable variation in grindability, most of the attention in the test work was focused upon estimating power requirements and optimizing a fineness-recovery relationship. This important matter is dealt with further in this section. The calculated power consumption for comminution in a conventional rod-ball mill circuit from a feed size of 1/2" - 5/8" to 60% -200 mesh is 12.59 KWH/ton.

COPPER RECOVERY VS. HEAD GRADE

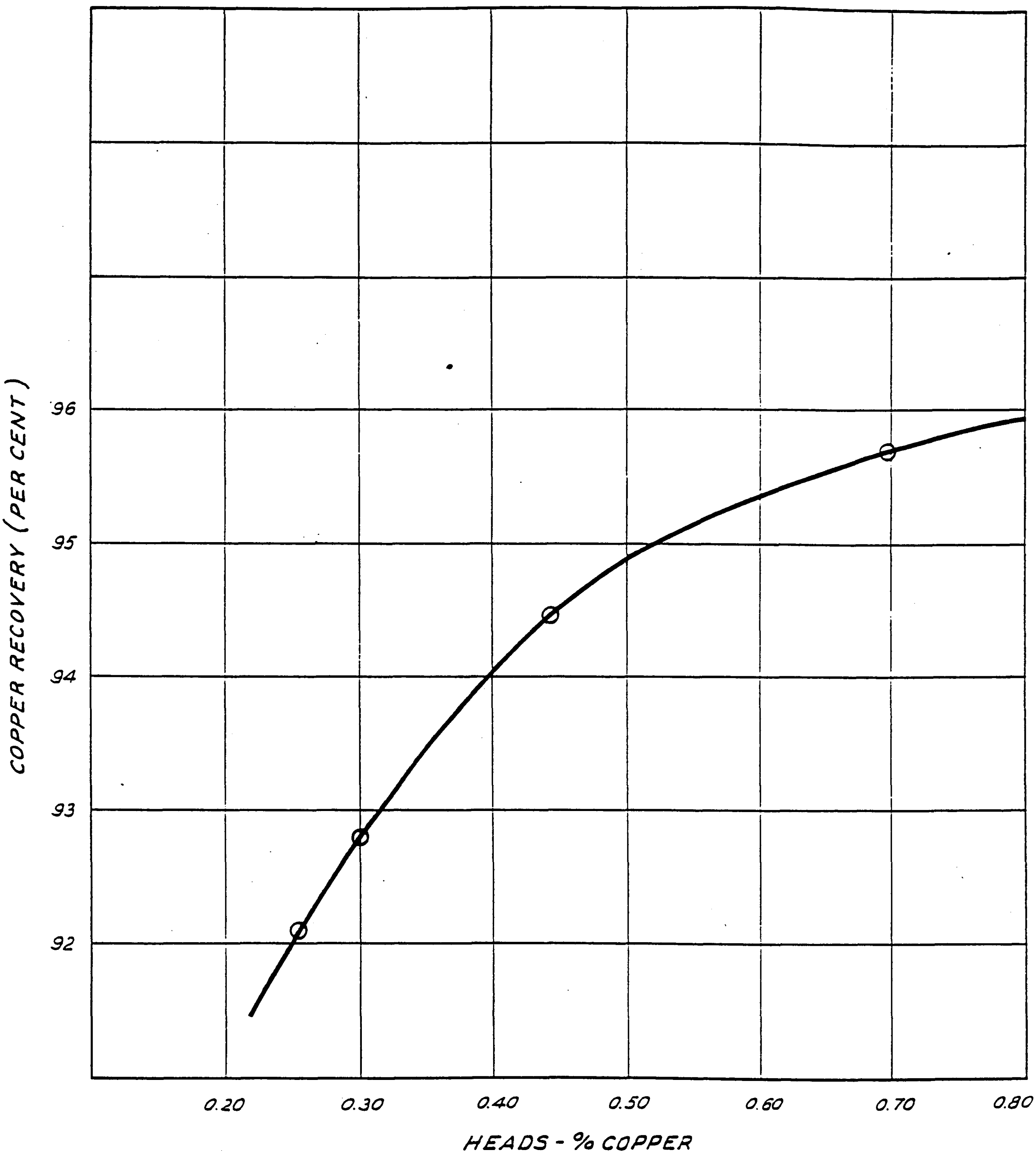


Figure V-2

DISCUSSION OF PROGRAM

Description of Metallurgical Samples

Following is a list of the various samples used for flotation and grindability tests:

Sample Description	Drill Hole No.	Interval Footage	Material (Footnote 1)	Calculated % Cu	Assay % Cu	Weight Lbs.
GH 1	72-3	180-450	Rejects	0.51	0.47	200
GH 2	72-18	160-310	"	0.32	0.27	100
GH 3	73-6	200-500	"	0.25	0.24	217
GH 4	72-10	200-500	"	0.65	0.66	260
GH 5	72-8	300-470	"	0.53		124
GH 6	72-1	150-320	"	0.37		110
GH 7	73-15	200-300	"	0.44		84
GH 8	73-19	240-340	"	0.53		84
GH 9	72-7	200-300	"	0.41		86
GH 10	73-12	200-300	"	0.41		86
GAL-A(2)	73-6	200-500	"	0.25		117
GAL-B(2)	72-10	200-500	"	0.65	0.25	149
AC-A(3)	72-8	300-470	Core	0.53		
AC-B(3)	72-1	150-320	Core	0.37		100
Gouge Material	73-17	100-140	Rejects	0.08		--
Gouge Material	73-19	190-240	"	0.20		--

The percentage of non-sulphide copper in the samples used for bench tests (GH 1 - GH 4) is negligible, 0.005 - 0.007.

NOTES:

1. Rejects comprise -4 mesh crushed core after removing a portion for assay.

   Core is half split core, uncrushed. Copper content has been calculated from core assays.

2. GAL-A and GAL-B comprise one half of samples GH 3 and GH 4, respectively, sent to the Galigher Company in Salt Lake City.

3. AC-A and AC-B are uncrushed split core samples, sent to Allis-Chalmers, U.S., for grindability tests. These are from the same drill holes and same footages as samples GH 5 and GH 6, respectively.

Description of Test Procedures

After some exploratory tests to determine the sensitivity of the different samples to various collectors and depressants, the following general procedure was adopted:

> Primary grinding with lime and Dow Z-200 to produce a flotation feed of pH 10.6, followed by roughing and scavenging with stage feeding of frother and collector.
>
> Cleaner flotation in three stages with lime.
>
> Regrinding of scavenger concentrates and cleaner middlings for reflotation in a separate circuit. In locked-cycle tests the reflotation concentrates were combined with rougher concentrates for cleaning.

The overall reagent consumption averaged:

Lime (hydrated)	1.5 lbs./ton
Dow Z-200	0.03 lbs./ton
Amyl Xanthate	0.04 lbs./ton
Frother (PPG)	0.02 lbs./ton

Cyanide was used sparingly (0.01 - 0.02 lb./ton) to aid pyrite depression on sample GH 3.

Effect of Fineness of Grind on Copper Recovery

The effect of the fineness of grind on copper recovery was investigated at Granisle on Samples GH 1 - GH4. The results are shown in Table V-1. The average increase in copper recovery with increasing fineness of grind was: 50% to 55% -200 mesh, 0.95% Cu; 55% to 60% -200 mesh, 0.65% Cu; 60% to 65% -200 mesh, 0.48% Cu. This effect is shown graphically in Figure V-3.

Optimization of Fineness of Grind

Basic Assumptions:

Rod mill feed K_{80} (80% passing) 16,000 μ (microns) 1/2" - 5/8"
Rod mill discharge K_{80} 1,400 μ
Flotation feed: K_{80} 180 μ (50% -200 M)
K_{80} 165 μ (55% -200 M)
K_{80} 138 μ (60% -200 M)
K_{80} 110 μ (65% -200 M)

Rod Mill Bond Work Index 22.2 (average of the two values reported by Allis-Chalmers Research Center = 18.5; adjusted by their recommended contingency factor of 1.2, 1.2 x 18.5 = 22.2).

Ball Mill Bond Work Index 14.4 (average of 10 values calculated from Granisle tests, taking into account an acceptable agreement between these data and those reported by Allis-Chalmers and Galigher). The grindability data for the individual samples are shown in Table V-2.

Power requirements calculated by Bond's formula:

$$W = \frac{10\ Wi}{\sqrt{P}} - \frac{10\ Wi}{\sqrt{F}}$$

Where W = KWH/Ton
Wi = Work Index
P = Product size in microns, μ
F = Feed size in microns, μ

Table V-1

EFFECT OF FINENESS OF GRIND
ON COPPER RECOVERY

Lot No.	% Cu.	% -200	% Rougher Recovery	Copper Recovery Difference (%) Increase 50-55% -200	Increase 55-60% -200	Increase 60-65% -200
GH 1	0.47	50	93.2			
		55	94.5	1.3		
		60	95.3		0.8	
		65	95.9			0.6
GH 2	0.27	50	93.9			
		55	94.4	0.5		
		60	95.0		0.6	
		65	95.3			0.3
GH 3	0.24	50	92.0			
		55	92.8	0.8		
		60	93.4		0.6	
		65	93.8			0.4
GH 4	0.66	50	94.6			
		55	95.8	1.2		
		60	96.6		0.8	
		65	97.0			0.4
Average	0.41	50	93.4			
		55	94.4	0.95		
		60	95.1		0.70	
		65	95.5			0.43

NOTE: The recoveries shown in the table apply to roughing and scavenging flotation only, excluding flotation cleaning.

COPPER RECOVERY VS. FINENESS OF GRIND

Basis: Average of samples GH-1 - GH-4

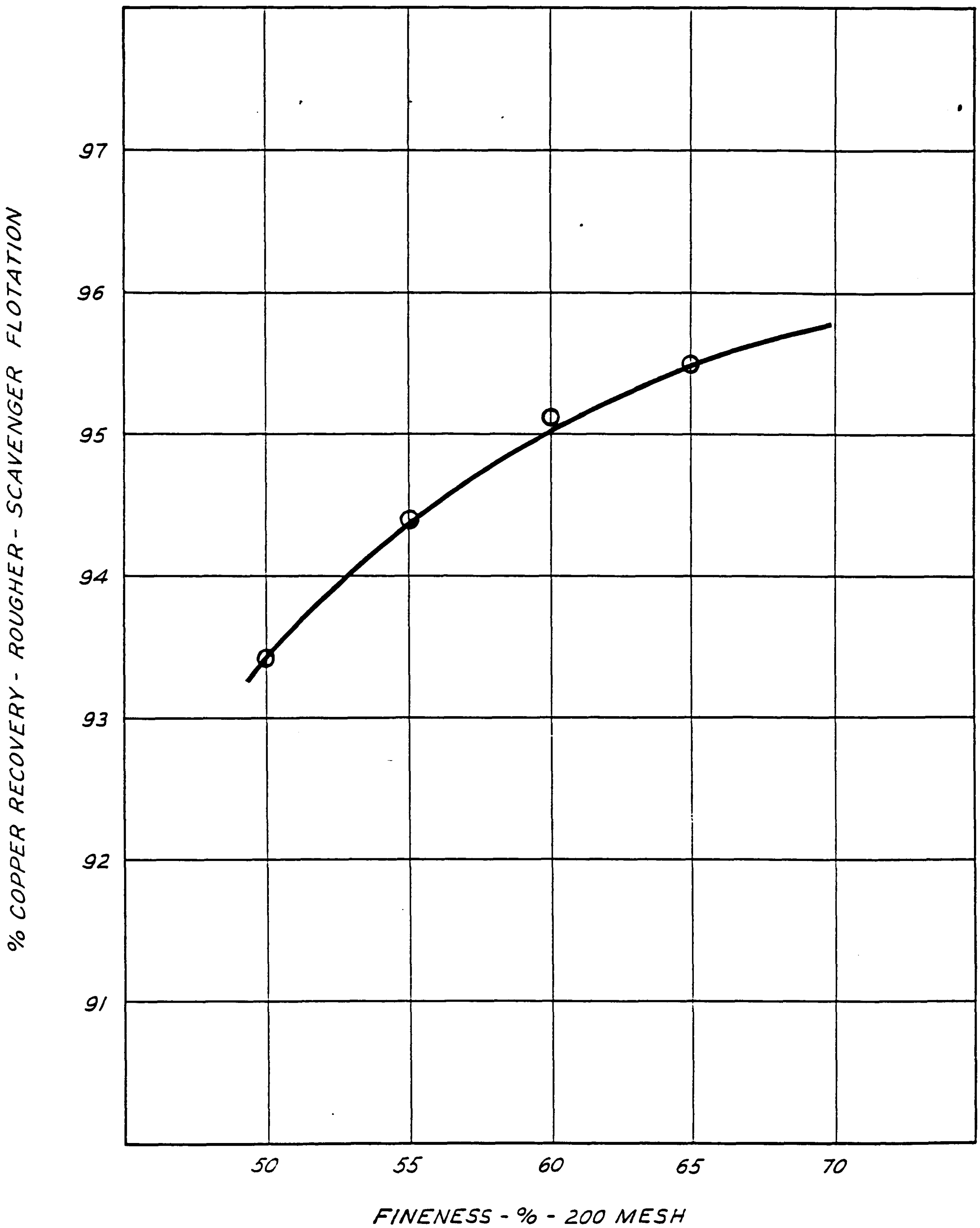


Figure V-3

Table V-2

GRINDABILITY DATA

Sample No.	Drill Hole No.	% Cu	Granisle Calculated B.M. Wi	Galigher Estimated B.M. Wi	Allis-Chalmers R.M. Wi	Allis-Chalmers B.M. Wi	Allis-Chalmers Abrasion Index
GH 1	72-3	.47	12.3				
GH 2	72-18	.27	14.5				
GH 3	73-6	.24	12.0	12.5-14.5			
GH 4	72-10	.66	11.6	11.0-12.5			
GH 5	72-8	.53*	15.5		17.5	15.8	0.1862
GH 6	72-1	.37	19.2		19.5	17.8	0.3472
GH 7	73-15	.44	16.7				
GH 8	73-19	.53	19.2				
GH 9	73-7	.62	10.3				
GH 10	73-12	.41	12.8				
Average			14.41		18.5	16.8	0.2667

* Assays for samples GH 5 to GH 10 were calculated from drill core assays.

Power cost 7.5 mils/KWH

Abrasion Index 0.2667 (average of two values reported by Allis-Chalmers)

Media and liner wear (Allis-Chalmers published data related to Abrasion Index)

Rods 0.265 lb./KWH
Balls 0.22 lb. KWH *
Rod mill liners 0.025 lb./KWH
Ball mill liners 0.020 lb./KWH *

* Since the ball consumption calculated in this manner is substantially higher than experienced in several similar plants, and because the wear rate for Ni-hard liners is known to be less than with manganese alloy upon which the Allis-Chalmers data were based, wear rates are discounted by 25% in these two instances.

Media and liner costs (current estimated cost f.o.b. minesite)

Rods	18¢/lb.
Balls	16¢/lb.
Cr-moly liners	50¢/lb.
Ni-hard liners	35¢/lb.

(Rod mill liners assumed to be half Cr-moly/half Ni-hard @ 42.5¢/lb.)

Average mill heads 0.41% copper = 8.2 lbs. of copper per ton of ore.

Copper price 80¢/lb., less marketing charges of 20¢/lb.

Calculated Data:

Power consumption figures were calculated from Bond's formula:

Rod Milling		4.17 KWH/Ton
Ball Milling	50% -200 Mesh -	6.89 KWH/Ton
	55% -200 Mesh -	7.37 KWH/Ton
	60% -200 Mesh -	8.42 KWH/Ton
	65% -200 Mesh -	9.89 KWH/Ton

Power costs

Rod-ball milling to 50% -200 mesh - (11.06 x .75) = 8.30 cents
55% -200 mesh - (11.54 x .75) = 8.66 cents
60% -200 mesh - (12.59 x .75) = 9.44 cents
65% -200 mesh - (14.06 x .75) =10.55 cents

Media and liner costs -		Cents/Ton
Rod milling - Rods	4.17 KWH x 0.265 = 1.10 lbs @ 18¢	19.9
Liners	4.17 KWH x 0.025 x 42.5¢	4.4
Total		24.3

Ball milling		Cents/Ton
To 50% -200 mesh		
Balls	6.89 KWH x 0.22 x .75 = 1.14 lbs @ 16¢	18.2
Liners	6.89 KWH x 0.020 x .75 x 35¢	3.6
Total		21.8
Total rod-ball milling		46.1
To 55% -200 mesh		
Balls	7.37 KWH x 0.22 x .75 = 1.22 lbs @ 16¢	19.5
Liners	7.37 KWH x 0.02 x .75 x 35¢	3.9
Total		23.4
Total rod-ball milling		47.7
To 60% -200 mesh		
Balls	8.42 KWH x 0.22 x .75 = 1.39 lbs @ 16¢	22.2
Liners	8.42 KWH x 0.02 x .75 x 35¢	4.4
Total		26.6
Total rod-ball milling		50.9

	Cents/Ton
To 65% -200 mesh	
Balls 9.89 KWH x 0.22 x .75 = 1.63 lbs @ 16¢	26.1
Liners 9.89 KWH x 0.02 x .75 = 35¢	5.2
Total	31.3
Total rod-ball milling	55.6

Total power plus steel costs will be:

	Cents/Ton
Rod-ball milling to 50% -200 mesh	54.4
Rod-ball milling to 55% -200 mesh	56.4
Rod-ball milling to 60% -200 mesh	60.3
Rod-ball milling to 65% -200 mesh	66.2

Economic Assessment:

Increasing recovery by 1%, increases revenue by 0.01 x 8.2 x 60¢ = 4.92 cents/ton.

The monetary gains and costs at various finenesses of grind were determined utilizing Table V-1, considering only the steel and power costs.

	Cents/Ton
50 to 55% -200 Mesh:	
Gain .95% recovery @ 4.92¢	4.7
Cost increase 56.4 - 54.4	2.0
Net Gain	2.7
55 to 60% -200 Mesh:	
Gain 0.70% recovery @ 4.92¢	3.4
Cost increase 60.3 - 56.4	3.9
Net Loss	0.5
60 to 65% 200 Mesh:	
Gain 0.43% recovery @ 4.92¢	2.1
Cost increase 66.2 - 60.3	5.9
Net Loss	3.8

For the conditions assumed, the break-even fineness falls between 55% and 60% -200 mesh. However, since plant-ground material contains a higher percentage of the undesirable plus 65 mesh fraction, a target of 60% -200 mesh is used for estimating power and equipment requirements.

Effect of Feed Grade on Copper Recovery

The relationship between the copper content of the test samples and the metal recovered, as determined from the four locked-cycle tests, is shown in Figure V-2. The recovery data indicate that material similar to the sample investigated can be treated in a conventional flotation circuit with copper recoveries of 93% to 95%.

Dewatering of Concentrates

Since the quantity of cleaned concentrates obtained from bench tests was insufficient for definitive thickening and filtering tests, attempts were made to determine the plant requirements by comparison with similar quantities of Granisle concentrates where the filtering and drying rate is 8-10 tons per hour. In general it was concluded that GH 1 and GH 4 thickened and filtered substantially better. Concentrates from GH 2 and GH 3 settled and filtered more slowly than average Granisle material, but no worse than the occasional troublesome concentrates encountered at Granisle. Some turbidity was noted in the supernatant liquid for GH 2 and GH 3, denoting less favourable settling properties, so the plant thickening area should be scaled up as noted under design criteria. A dryer capability similar to that at Granisle should suffice.

Tailings Settling Characteristics

As in the case of concentrates, the settling properties were compared with Granisle tailings. In all cases, the pulp settled more rapidly and with greater clarity than Granisle tailings. A pH in excess of 11.0 was required to achieve a perfectly clear supernatant liquid, but a lower pH was adequate when anionic flocculants were used.

Effect of Gouge Material

Flotation tests on "gouge" material selected from Drill Hole Nos. 73-17 and 73-19 containing 0.08% and 0.20% copper respectively, and on samples of GH 1 to which varying amounts of "gouge" material were added, showed that gouge material had little adverse effect upon performance. Copper recovery was 89% on gouge alone, and higher when mixed with GH 1 sample. Frothing properties were not noticeably impaired.

The term "gouge material" refers to heavily fractured somewhat slickensided rock. This material is much less plastic than the gouge encountered at Granisle where adverse froth effects have been experienced.

Magnetite

The results of Davis Tube magnetic separation tests carried out by Britton Research Limited on flotation tailings from samples GH 1 to GH 4, and on a recent sample of Granisle tailings are as follows:

	Magnetite Concentrate			Calculated Overall Percentage in Tailings	
Sample	% of Feed	% Cu	% Fe	Fe	Magnetite
GH 1 Test 12	24.1	0.027	18.6	4.48	6.2
GH 2 Test 11	8.5	0.009	9.0	0.77	1.1
GH 3 Test 12	28.2	0.010	12.6	3.55	4.9
GH 4 Test 11	18.3	0.021	14.4	2.64	3.7
Average	19.8	0.017	13.7	2.86	4.0
Granisle Tailings	3.52	0.09	36.64	1.40	1.93

While the calculated percentage of magnetite is approximately twice that of Granisle tailings, it is finely disseminated and should not cause difficulties with the tramp iron magnets in the crushing plant.

MOLYBDENITE RECOVERY

The recovery of molybdenum was not investigated. However, from molybdenite assays on flotation heads and typical concentrates and tailings, the indicated distribution of molybdenite in copper flotation was as follows:

Percentage MoS_2

	GH 1	GH 2	GH 3	GH 4	Average
Heads	0.027	0.006	0.014	0.038	0.021
Copper Concentrates	1.83	0.47	1.20	1.51	1.25
Tailings	0.002	0.001	0.002	0.005	0.0025
Calc. Recovery	92.9	83.9	85.7	87.2	87.4

By comparison, the earlier bench tests by Kennecott showed 75.17 recovery in copper flotation. Excerpts from their report stated "tests showed that the molybdenite could be separated from the copper concentrate and recovered as a high grade product, but insufficient work was done to establish an optimum molybdenite recovery process for this mineralization". Their metal balance for the copper-moly separation was as follows:

Product	Weight Percent	Assays % Cu	Assays % MoS_2	% Distribution Cu	% Distribution MoS_2
Copper Concentrate	84.5	26.7	.142	87.50	8.23
Molybdenite Tailings	13.66	22.6	.65	11.98	6.10
Molybdenite Cleaner Tailings	.84	15.1	32.53	.49	18.73
Molybdenite Concentrate	1.05	.77	92.60	.03	66.94
Calculated Head	100.00	25.77	1.454	100.00	100.00

The indicated overall recovery of molybdenite reported in the Kennecott report is about 50% in concentrates grading 92.6% MoS_2. This is based upon Kennecott's 67% molybdenite recovery from copper concentrates containing 75% of the molybdenite in the head sample.

REFERENCES

Detailed test data are contained in the following reports:

Progress Reports 1 to 4 covering Samples GH 1 to GH 4, respectively, and a supplementary report on the Testing of Gouge Material, issued by the Metallurgical Laboratory, Granisle Copper Limited.

Results of Laboratory Testing on Samples from Huckleberry Project in report dated January 25, 1974, by the Galigher Company, Salt Lake City.

Technical Report 71-10, on Amenability Testing of Samples from Kennco's Huckleberry Project, April 22, 1971.

Allis-Chalmers Manufacturing Company Test Report dated November 12, 1973, covering Grindability Tests on two samples.

Letter Report from Britton Research Limited, December 7, 1973, covering Davis Tube Magnetic Separation tests.

Report of Ore Microscopy 310-03-40 by George A. Wilson, P. Eng.

Cyanamid Technical Service Report, Microscopical Examination of Concentrates March, 1974.

VI

MINING

GENERAL

The Huckleberry deposit is to be mined using large scale open pit methods. Topography dictates that most of the mill feed will have to be hauled on adverse road grades, while a large portion of the waste removal will have favourable road grades. After the initial years, mill feed and waste excavation will be distinctly separate operations, with waste handling well above and removed from mill feed mining. Benches will be 40 feet high, and in order to maintain a maximum wall slope of 45°, safety berms 80 feet in width toe to toe will be maintained on every other bench. Maximum road grade will be 10%. Preproduction stripping will require removal of 5,000,000 tons of overburden, and 2,000,000 tons of waste rock. This is adequate to uncover about a six month ore supply at 10,000 per day. It is estimated that about 1,500,000 tons of overburden will be removed on a contract basis. The remaining preproduction material will be removed by the mine operator. This latter material is expected to be suitable for tailings dams construction. Two production operating schedules are considered. The first is for a 10,000 TPD plant, and the second is for a 15,000 TPD plant. Both schedules are based on a five day operating week.

SCHEDULING CRITERIA

An idealized mining schedule, for a maximum return on investment, would incorporate a diminishing cutoff grade over time coupled with an increasing stripping ratio. The application of these criteria must be tempered by practical considerations. While the Huckleberry deposit does have a high grade core, grades of mill feed on an annual basis deviate very little from the average grade for a given operating cutoff grade. Instead of having steadily diminishing cutoff grades; the Huckleberry operation is faced with mining to the average grade of the payback pit, and then treating the lower grade material from the ultimate pit reserves. During the payback period, the stripping of waste should be kept to a minimum, but it is apparent that if this is done unrealistic stripping ratios would result in later years. Without any stockpiling of low grade material, some deferment of waste stripping is possible. However, at a minimum copper price of $0.60 it is preferable to raise the mill feed cutoff grade in the early years while stockpiling low grade, and sustain a higher stripping ratio, than it is to lower the cutoff grade and minimize the stripping ratio. The resulting stockpiling will be limited by mill feed requirements at a constant mining rate and by the spaces available for a stockpile.

PRODUCTION SCHEDULES

Three variables were considered in selecting a production schedule: cutoff grade during payback period, cutoff grade after payback period, and milling rate. Table VI-1 shows the alternatives that were tested. Production schedules were drawn for each alternative using best guess estimates of capital and operating costs as guidelines in selecting mining rates. Cash flows and internal rates of return were calculated and compared at a constant $1.00 per pound copper price and for schedules (e) to (f) at $0.70 per pound copper to ascertain that the relative profitability of these was not affected by copper price. The highest internal rates of return were alternative (b) at 10,000 TPD and alternative (d) at 15,000 TPD. Equipment selection and operating costs were determined for these two cases and these were the base data for the financial analysis. The schedule for alternative (b) is shown on Table VI-2 and for alternative (d) on Table VI-3.

EQUIPMENT SELECTION

With a basic five day work week in the pit, a variety of loading and hauling equipment combinations are possible. By limiting the haulage truck selection to electric drive 100 ton capacity units, the possibilities were reduced to the following:

Case 1 - 10,000 TPD with 3, 7 cu. yd. shovels; assumed stripping ratio 1.7

Case 2 - 10,000 TPD with 3, 11 cu. yd. shovels; assumed stripping ratio 1.7

Table VI-1

MINE OPERATING CHARACTERISTICS

ALTERNATIVE	MILL FEED NOMINAL RATE	INTERMEDIATE PIT CUTOFF GRADE	ULTIMATE PIT CUTOFF GRADE	NOMINAL S.R.
(a)	10,000	.40	.30	1.70
(b)	10,000	.40	.25	1.25
(c)	15,000	.40	.30	1.60
(d)	15,000	.40	.25	1.28
(e)	15,000	.35	.30	1.28
(f)	15,000	.35	.25	1.00

Table VI-2

PRODUCTION SCHEDULE - 10,000TPD

YEAR	MILL FEED TONS	MILL FEED GRADE	WASTE & STOCKPILE TONS	S.R.	STOCKPILE RECLAIM TONS	STOCKPILE RECLAIM GRADE
PREPRODUCTION			7,000,000			
1.	3,650,000	.501	4,576,000	1.25		
2.	3,650,000	.501	4,576,000	1.25		
3.	3,650,000	.501	4,576,000	1.25		
4.	3,650,000	.501	4,576,000	1.25		
5.	3,650,000	.501	4,576,000	1.25		
6.	3,650,000	.501	4,576,000	1.25		
7.	3,650,000	.501	4,576,000	1.25		
8.	3,650,000	.422	4,576,000	1.25		
9.	3,650,000	.353	4,576,000	1.25		
10.	3,650,000	.353	4,576,000	1.25		
11.	3,650,000	.353	4,576,000	1.25		
12.	3,650,000	.353	4,576,000	1.25		
13.	3,650,000	.353	4,576,000	1.25		
14.	3,650,000	.353	4,576,000	1.25		
15.	3,650,000	.353	4,576,000	1.25		
16.	3,650,000	.353	4,576,000	1.25		
17.	3,650,000	.353	4,576,000	1.25		
18.	3,650,000	.353	4,576,000	1.25		
19.	3,650,000	.353	4,576,000	1.25		
20.	1,147,000	.353	1,444,000	1.26	2,503,000	.347
21.					3,650,000	.347
22.					3,650,000	.347
23.					3,650,000	.347
24.					1,605,000	.347
TOTALS	70,497,000	.412	95,388,000	1.35	15,058,000	.347

Table VI-3

PRODUCTION SCHEDULE - 15,000 TPD

YEAR	MILL FEED TONS	MILL FEED GRADE	WASTE & STOCKPILE TONS	S.R.	STOCKPILE TONS	RECLAIM GRADE
PREPRODUCTION			7,000,000			
1.	5,475,000	.501	7,000,000	1.28		
2.	5,475,000	.501	7,000,000	1.28		
3.	5,475,000	.501	7,000,000	1.28		
4.	5,475,000	.501	7,000,000	1.28		
5.	5,475,000	.501	7,000,000	1.28		
6.	5,475,000	.367	7,000,000	1.28		
7.	5,475,000	.353	7,000,000	1.28		
8.	5,475,000	.353	7,000,000	1.28		
9.	5,475,000	.353	7,000,000	1.28		
10.	5,475,000	.353	7,000,000	1.28		
11.	5,465,000	.353	7,000,000	1.28		
12.	5,475,000	.353	7,000,000	1.28		
13.	4,797,000	.353	4,388,000	1.28	668,000	.347
14.					5,475,000	.347
15.					5,475,000	.347
16.					3,440,000	.347
TOTAL	70,497,000	.412	95,388,000	1.35	15,058,000	.347

Case 3 - 10,000 TPD with 2, 11 cu. yd. shovels and 1, 10 cu. yd. loader; assumed stripping ratio 1.7

Case 4 - 15,000 TPD with 3, 11 cu. yd. shovels; assumed stripping ratio 1.4

Case 5 - Same as Case 3, but adjusted for actual production schedule stripping ratios

Case 6 - Same as Case 4, but adjusted for actual production schedule stripping ratios.

Case 1 can be rejected because the shovel-truck match is uneconomic. Cases 2, 3 and 4 were studied in order to observe the effect of increased stripping ratios on equipment capacities. These cases influenced Cases 5 and 6 only as far as flexibility in the expansion of waste production was concerned. Therefore, further discussion regarding loading equipment is restricted to the last two cases.

Drilling

Primary drilling is to be done with electric rotary blast-hole drills producing 9 7/8" diameter holes, operated by a driller and a helper. A maximum penetration is estimated to be fifty feet per drilling hour; maximum drill efficiency is assumed to be 75% of available shift time. A typical drilling pattern of 22 feet by 20 feet, with seven feet of subgrade is proposed; this will yield about 1500 tons per hole.

10,000 TPD:

With a stripping ratio of 1.25 (Case 5), the required drill footage is 254,000 feet per year. Two drills are required, each operating two shifts per day. The resulting effective utilization is 82%.

15,000 TPD:

With a stripping of 1.23 (Case 6), two machines, one operating three shifts per day, and the other two shifts per day, are required to drill 389,000 feet per year. The effective utilization is 62%.

Blasting

Bulk ammonium nitrate stored in silos is to be used as much as possible as a blasting agent. It is estimated that with a suitable blast hole dewatering unit, ammonium nitrate will make up about 80% of blasting materials consumption. The remaining 20% will consist of high density water gels stored in an explosive magazine. The power factor is estimated to be 0.6 pounds. Blast hole loading will be done by a crew of four provided with a blasting truck and a bulk prill loading truck.

Loading

10,000 TPD:

With an average digging utilization of 80%, two 11 cu. yd. shovels, each operating two shifts a day are required to load the mill feed, waste and stockpile material at a 1.25:1 ratio. Productivity is estimated at 9,000 tons per shovel shift. Backup capacity is required to maintain a realistic availability, but a third shovel would lower the required availability and the effective utilization beyond levels which could be justified. A wheel loader, is to provide backup loading capacity at a lower capital cost. A 10 cu. yd. loader is to load approximately 15% of the pit production. This amounts to 1,200,000 tons per year. The required shovel availability is an easily managed 70%, while that of the loader is 60%. The loader is also to be used for miscellaneous pit and road maintenance work. Operating costs are not expected to be significantly higher than with a third shovel, two operators and two oilers are required per shift, for a total of eight men. An increase in either mill feed or stripping requirements would result in a third shift being scheduled. In that case, a third shovel is preferred over a loader, because of the increase in available machine time and an increase in required production from the third unit. A loader is not economic when used as a primary loading machine, especially when rock fragmentation might be inconsistent.

15,000 TPD:

Six 11 cu. yd. shovel shifts per day are required to maintain production with a stripping ratio of 1.28. To maintain a required availability of 70%; three

11 cu. yd. shovels are required, two to operate at any given time. Two operators and two oilers per shift make up a total loading crew of twelve men. If the throughput is increased beyond 15,000 TPD, additional shovel shifts can be scheduled.

<u>Hauling</u>

One hundred ton capacity diesel-electric trucks have been selected for their low operating cost characteristics coupled with relatively long useful life. The maximum required fleet availability is to be 80%. The fleet size is based on cycle characteristics calculated according to the haulage people and the type of loading unit used. Roads are to have a maximum grade of 10%, and a running width of eighty feet. The typical haulage profile, based on the "centre of gravity" of the intermediate pit; consists of 2800 feet at an adverse grade of 10% and 1900 feet level for the ore haul, and 3600 at 10% with 1900 feet level for the waste haul.

	TOTAL TRUCKS	TOTAL DRIVERS
Case 1	12	27
Case 2	9	21
Case 3	9	21
Case 4	9	24

When the 100 ton units are matched to 11 yd. shovels, hauling productivity of 326 tons per truck hour in ore and 300 tons per truck hour in waste can be achieved.

10,000 TPD:

Three trucks are required per shovel shift in ore, and four trucks in waste. The total fleet size is nine trucks if the availability is to be 80%. Utilization of available time is 75%. With one extra driver per shift the total driving complement is 16 men spread over two shifts.

15,000 TPD:

The truck requirements per shift are the same as for 10,000 TPD, therefore, the fleet size is the same. Seven drivers and one spare per shift make up a haulage crew of 24 men. Any increase in either mill feed or stripping will require the purchase of additional hauling capacity, or the extension of mining operations to a seven day week.

Service Equipment

Mine service equipment is basically the same for 10,000 TPD and 15,000 TPD operations consisting of two track type bulldozers for dump and pit maintenance. A wheel dozer will be available for shovel cleanups. Roads will be maintained by two graders, one for pit use and the other for the plantsite, and by a combination sand and water truck. A fuel truck will be available to service this equipment. In addition, a small front end loader is provided in the 15,000 TPD case, since no wheel loader is otherwise provided for general pit service such as loading the sand truck. In the 10,000 TPD case, the large loader would do this work as well as loading haulage trucks. Provisions are also made for the purchase of stationary pumping equipment, cable arches, power cable and cable boats, lighting equipment, other miscellaneous equipment and tools. No secondary drilling is anticipated, but a truck mounted 3 inch percussion drill driven by a 600 c.f.m. compressor will be available for work; which the rotary drills cannot do efficiently.

CAPITAL COST

Equipment capital costs are shown in Table VI-4. The corresponding manpower requirements are summarized in Table VI-5. A capital replacement program has been drawn in Table VI-6, based on the estimated equipment working life outlined in Table VI-7.

OPERATING COST

Operating policy, equipment combinations, labour distribution and prices of consumable supplies are all significant considerations in determining operating costs. Annual operating costs are shown in Table VI-8 and unit operating costs in Table VI-9. Labour costs are detailed in Chapter IX; labour cost figures in Table VI-8 are taken from Table IX-3.

Table VI-4

MINING - EQUIPMENT CAPITAL COST SCHEDULE

			10,000 TPD		15,000 TPD	
FUNCTION	EQUIPMENT	UNIT COST	NO.	COST	NO.	COST
DRILLING	B. E. 45-R Rotary Drills	$350,000	2	$ 700,000	2	$ 700,000
	3" Airtrac & 600 Compressor	50,000	1	50,000	1	50,000
BLASTING	Bulk Prill Truck	25,000	1	25,000	1	25,000
	Bulk A. N. Silo & Magazine	30,000	1	30,000	1	30,000
	Blasthole Dewatering Unit	9,000	1	9,000	1	9,000
LOADING	P & H 1900 11 yd^3 Shovels	800,000	2	1,600,000	3	2,400,000
	Michigan 475B 10 yd^3 Loader	225,000	1	225,000	-	-
HAULAGE	U. R. M-100 Trucks c/w 71 yd^3 box	350,000	9	3,150,000	9	3,150,000
SUPPORT EQUIPMENT	Cat. 988 F.E.L.	130,000	-	-	1	130,000
	Cat. D84 Dozers	110,000	2	220,000	2	220,000
	Cat. 834 Wheel Dozer	130,000	1	130,000	1	130,000
	Cat. 16 Grader	120,000	1	120,000	1	120,000
	Cat. 14 Grader	80,000	1	80,000	1	80,000
MISCELLANEOUS	umps			15,000		15,000
	Sanding & Water Truck			50,000		50,000
	Fuel Truck			15,000		15,000
	Fuel Station			25,000		25,000
	Power Cables & Fittings			30,000		30,000
	Cable Arches & Boats			19,000		19,000
	Water Piping			18,000		18,000
	Light Standards			15,000		15,000
	Misc. Tools & Equipment			50,000		50,000
TOTAL				$6,576,000		$7,281,000

Table VI - 5

MINING - MANPOWER DISTRIBUTION

	10,000 TPD	15,000 TPD
DRILLING		
Drillers	4	5
Helpers	4	5
BLASTING		
Blaster	1	1
Helpers	3	3
LOADING		
Shovel OIP	4	6
Oilers	4	6
HAULAGE		
Drivers	16	24
TRUCK SHOP		
H.D. Mechanics	8	12
Welders	5	5
Apprentices	5	5
Service Men	2	2
Janitors	1	1
Machinists	½	½
Electricians	5	5
Master Mechanics	1	1
Shop Foremen	2	2
Mech. Planners	1	1
Clerk Typist	½	½
SUPPORT EQUIP.		
Track Dozers	4	6
Model 16 Grader	2	3
Model 14 Grader	1	1
R.T. Dozer	2	3
Ancillary Equip.	3	3
SUPERVISION		
General Foreman	1	1
Shift Bosses	2	3
Spare Shifter & Trainer	1	1
TOTAL	83	106

Table VI - 6

MINING - CAPITAL REPLACEMENT SCHEDULE

PRODUCTION YEAR	10,000 TPD	15,000 TPD
1	135,000	182,000
2	135,000	182,000
3	150,000	547,000
4	710,000	212,000
5	405,000	372,000
6	200,000	667,000
7	255,000	182,000
8	710,000	312,000
9	150,000	547,000
10	510,000	477,000
11	135,000	182,000
12	760,000	797,000
13	135,000	182,000
14	255,000	182,000
15	420,000	60,000
16	710,000	60,000
17	135,000	
18	200,000	
19	135,000	
20	150,000	
21	135,000	
22	100,000	
23	75,000	
24	75,000	
TOTAL	6,780,000	5,143,000

Table VI-7

MOBILE EQUIPMENT REPLACEMENT CRITERIA

EQUIPMENT LIFE AT 10,000 TPD

	ECONOMIC LIFE	REPLACEMENT COST	
Track Dozers	1.5 years	$ 75,000	(average)
R. T. Dozers	5 years	130,000	
988	5 years	130,000	
Piping, Arches, & Boats	INDEFINITE	10,000	
Pumps	5 years	10,000	
16 Grader	7 years	120,000	
14 Grader	10 years	80,000	
M-100 Truck	4 years	350,000	
10 yd. Loader	4 years	225,000	(when used)
Service Truck	3 years	15,000	
Sanding & Water Truck	6 years	50,000	
Bulk Prill Truck	10 years	25,000	
Miscellaneous	INDEFINITE	50,000	

EQUIPMENT LIFE AT 15,000 TPD

Track Dozers	1 years	110,000
R. T. Dozers	4 years	130,000
988	5 years	130,000
Piping, Arches & Boats	INDEFINITE	12,000
Pumps	5 years	10,000
16 Grader	6 years	120,000
14 Grader	10 years	80,000
M-100 Trucks	3 years	350,000
Service Truck	3 years	15,000
Sanding & Water Truck	5 years	50,000
Bulk Prill Truck	10 years	25,000
Miscellaneous	INDEFINITE	60,000

Table VI-8

MINE- ANNUAL OPERATING EXPENDITURES

10,000 TPD S.R. = 1.25	OPERATING LABOUR	OPERATING SUPPLIES	MAINTENANCE LABOUR	MAINTENANCE SUPPLIES	TOTAL
Drilling	89,470	170,800	40,130	20,810	321,210
Blasting	41,300	481,140	-	-	522,440
Loading	94,320	177,300	68,730	55,970	396,320
Hauling	186,670	371,280	132,390	109,200	799,540
Truck Shop	72,610	40,500	-	-	113,110
Support Equip.	109,170	156,840	63,500	44,340	373,850
General Services & Supervision	109,200	35,100	-	-	144,300
TOTAL	702,740	1,432,960	304,750	230,320	2,670,770

15,000 TPD S.R. = 1.28	OPERATING LABOUR	OPERATING SUPPLIES	MAINTENANCE LABOUR	MAINTENANCE SUPPLIES	TOTAL
Drilling	111,340	248,400	40,130	37,260	437,130
Blasting	41,300	720,360	-	-	761,660
Loading	141,440	186,300	97,340	74,520	499,600
Hauling	280,010	596,160	178,460	173,880	1,228,510
Truck Shop	72,610	62,100	-	-	134,710
Support Equip.	181,740	161,460	77,890	62,100	483,190
General Services & Supervision	143,200	37,260	-	-	180,460
TOTAL	971,640	2,012,040	393,820	347,760	3,725,260

Table VI-9

MINE - UNIT OPERATING COSTS

10,000 TPD	OPERATING LABOUR	OPERATING SUPPLIES	MAINTENANCE LABOUR	MAINTENANCE SUPPLIES	TOTAL
Drilling	.011	.021	.005	.003	.040
Blasting	.005	.059	-	-	.064
Loading	.012	.022	.008	.007	.049
Hauling	.023	.046	.016	.013	.099
Truck Shop	.009	.005	-	-	.014
Support Equip.	.013	.019	.008	.005	.046
General Services & Supervision	.013	.004	-	-	.018
TOTAL	.087	.177	.038	.028	.330

15,000 TPD	OPERATING LABOUR	OPERATING SUPPLIES	MAINTENANCE LABOUR	MAINTENANCE SUPPLIES	TOTAL
Drilling	.009	.020	.003	.003	.035
Blasting	.003	.058	-	-	.061
Loading	.011	.015	.008	.006	.040
Hauling	.023	.048	.014	.014	.099
Truck Shop	.006	.005	-	-	.011
Support Equip.	.015	.013	.006	.005	.039
General Services & Supervision	.012	.003	-	-	.015
TOTAL	.078	.162	.032	.028	.300

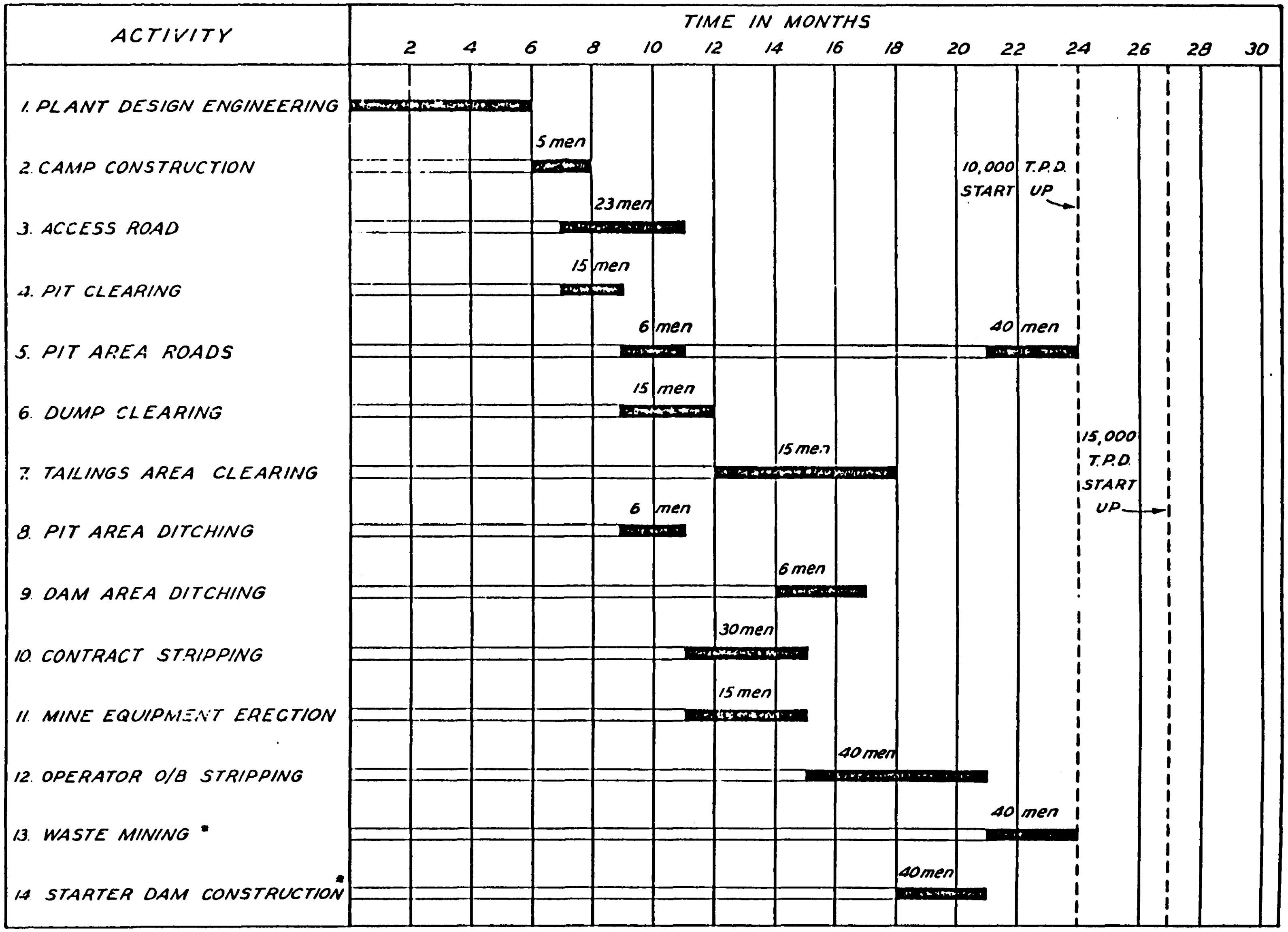
The costs of operating supplies were calculated using current prices of major consumable items, as much as possible. However, in general, operating supply costs are based on estimates of hourly equipment costs from Granisle Copper for 1973, which were adjusted for equipment changes. Information from other B. C. properties was also incorporated. This approach was used because of the difficulty in estimating the life of wearable parts and to account for minor, but important, items such as lubricants, batteries, etc. The same approach was used in calculating the repair and maintenance portions of the operating costs. Those major consumable items for which direct estimates are made; include blasting agents, truck tires and fuel. Prices used are $100 per ton for ammonium nitrate, and $15 per hundred weight for high density gels. Truck tires are priced at $4000 each. and have an estimated life of 2000 hours. Fuel is priced at $0.27 per gallon.

PREPRODUCTION

The schedule of preproduction activities, as shown in the bar chart in Figure VI-1, is to coincide with the plant construction schedule. The first ten activities are independent of the size of the concentrator with respect to time. The last four activities in the bar chart may be deferred by three months for the 15,000 TPD plant. These activities are the preproduction mining and dam construction. Figure VI-2 which shows the preproduction manpower, indicates that the average manpower demand on camp facilities is much less at 15,000 TPD than at 10,000 TPD.

A total of 1150 acres of clearing is required in the pit, dumps, and tailings areas. The whole area is to be cleared during preproduction, although clearing of 300 acres in the pit and dump areas could be deferred. However, a cost saving is possible in clearing the total area at once, because lower unit costs may be negotiated for a large acreage. In addition, fewer problems in diversion ditching and runoff control will occur by clearing the total area. The area to be covered by tailings dams, but not the pond, is to be grubbed and cleared of organic material. This area is about 200 acres.

PREPRODUCTION SCHEDULE



* Subject to suitability of material for dam construction.

Figure VI-1

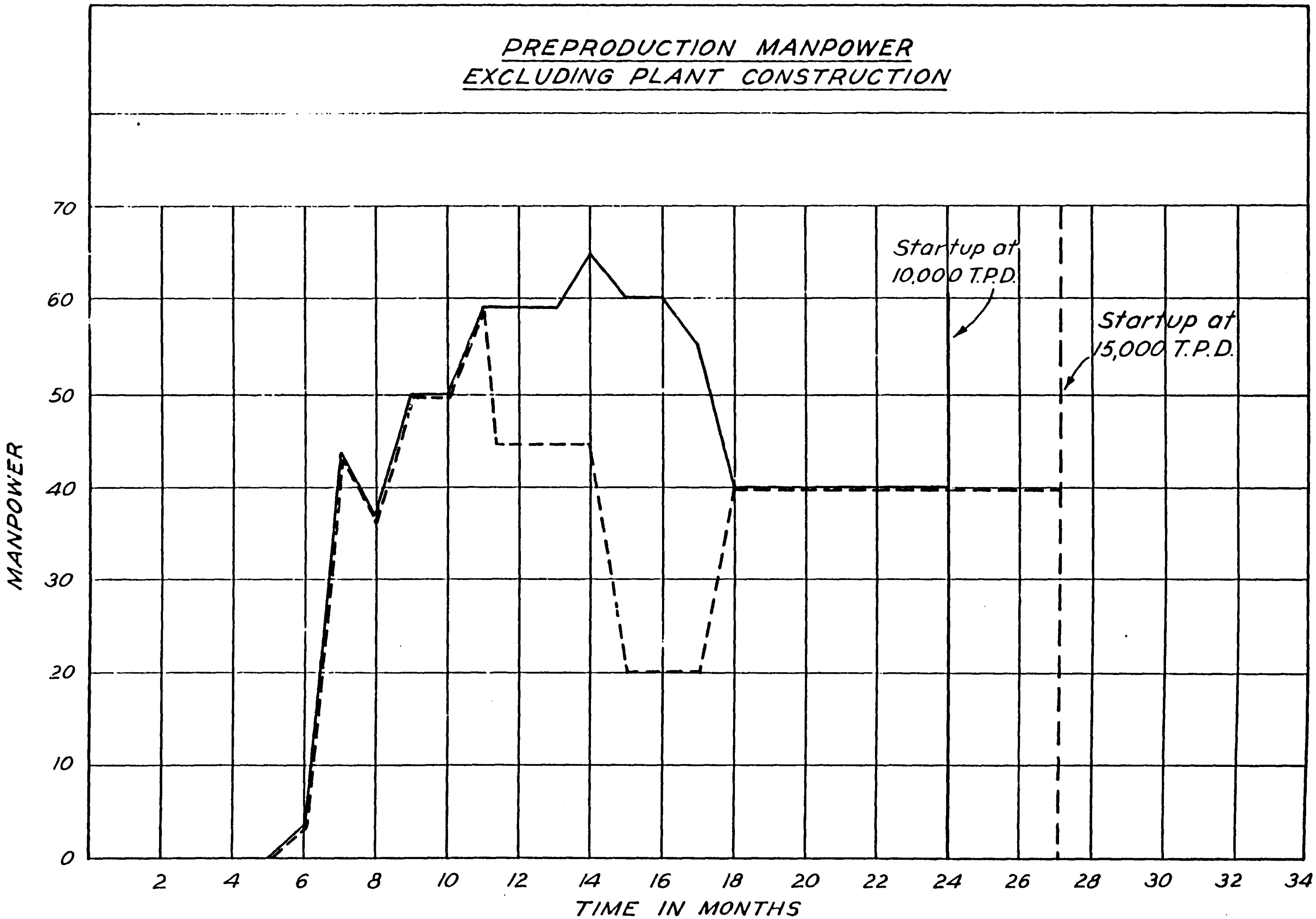

PREPRODUCTION MANPOWER
EXCLUDING PLANT CONSTRUCTION
MANPOWER
70
60
50
40
30
20
10
0
2
4
6
8
10
12
14
16
18
20
22
24
26
28
30
32
34
TIME IN MONTHS
Startup at 10,000 T.P.D.
Startup at 15,000 T.P.D.

Figure VI-2

Preproduction road building has three phases. The first is the construction of an access road from Sweeny Lake to the minesite. The cost of this phase is not shown in the table of preproduction capital expenditures, but is included in Chapter VII. The second phase consists of pit perimeter roads which would be used for servicing power lines, pumping facilities, etc. The third phase is the construction of the mining roadways such as the crusher access and the dump roads. The cost of this work does not include the cost of producing the material, which will be pit run rock.

The removal of 1,500,000 tons of overburden by contract is necessary because this material will not have sufficient bearing strength to support conventional mining equipment. The remaining preproduction stripping which consists of 3,500,000 tons of overburden and 2,000,000 tons of rock, is to be mined by the operators. The rock increment is included to provide ballast for production roadways and to provide material for starter dam construction, should the overburden prove inadequate. The extra cost associated with starter dam construction is the cost of hauling the material beyond the normal dumping point and the cost of emplacing it. The excess hauling distance is one mile This is the distance between the primary crusher and the damsite. The requirements for starter dam construction is 2,000,000 tons.

A summary of preproduction capital expenditures is shown in Table VI-10. Included is an overhead charge to account for temporary power facilities travel, communications, and general administration.

WASTE ROCK AND OVERBURDEN DISPOSAL

Types of Material

The Huckleberry deposit is covered by approximately 2.7 million cu. yd.; (5,000,000 tons) of overburden. This overburden consists of the following material classes for which exact tonnages have not been determined:

1. Black organic swamp soil which could potentially be used for reclamation.
2. Sorted glacial till (gravel) which could be used for road construction and land fill.

Table VI-10

PREPRODUCTION - CAPITAL EXPENDITURE

	CAPITALIZED
CLEARING	
Pit, Dams, Dumps & Plant, 1,150 acres x $300	$ 345,000
DAM AREA GRUBBING	
200 acres x $300	60,000
LAND PREPARATION	
Dam Diversion Ditching - 7,400' x $10	74,000
Pit Diversion Ditching - 5,000' x $10	50,000
Perimeter Roads - 5,000' x $10	50,000
Crusher Access Road - 2,000' x $50	100,000
Dump Roads - 4,000' x $50	200,000
PREPRODUCTION STRIPPING	
Contract O/B Removal - 1,500,000 x $1.00	1,500,000
Operator Removed O/B - 3,500,000 x $0.40	1,400,000
Operator Removed Rock - 2,000,000 x $0.40	800,000
STARTER DAM CONSTRUCTION (2.0 Million Tons)	
Excess Hauling Cost - 1 mile x $0.10/ton mile	200,000
Placement Cost - $0.06/ton	120,000
SUB-TOTAL	$4,899,000
COMPANY OVERHEAD (Prior to Production)	
Travel, communications, moving, salaries and administration, and temporary power, 25%	1,224,500
GRAND TOTAL PREPRODUCTION EXPENDITURES	$6,123,500

3. Unsorted, muddy glacial till.

Because of the physical characteristics and potential uses of these materials, they should be dumped in separate areas. In addition to overburden, waste rock encountered in mining will have to be stored. To the extent that it is economically convenient, waste rock will be separated into the following categories and dumped separately:

1. Sub-marginal rock returns which will not cover the costs of further processing;
2. Marginal rock which will not cover all operating and capital overhead, but which will possibly be economic after the completion of mining.

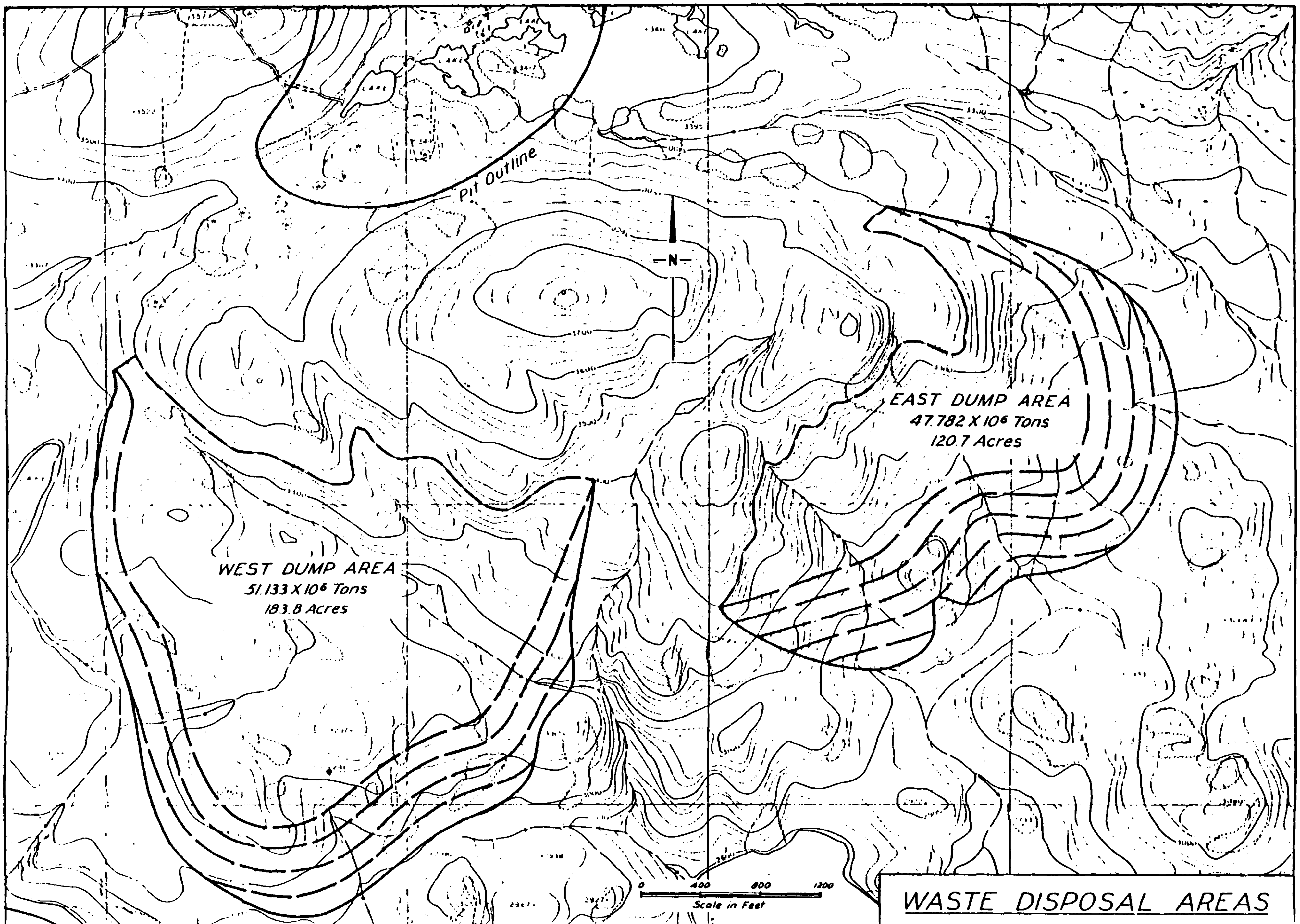
The marginal material will be stockpiled in an area from which it can easily be reclaimed for further processing.

Criteria for Dump Design

1. Material is to be sorted into the physical and economic categories described above and each category is to be stored separately.
2. Dumps are to be located as close to the pit mouth as possible in order to minimize haulage costs.
3. Extreme haulage road grades are to be avoided.
4. Dumps are not to be located over water courses unless diversion ditches are employed.
5. Dumps are to be located on stable foundation material.

Dump Locations

Suitable waste-disposal areas lie to the southwest and southeast of the prominent knoll immediately south of the mineral deposit (Figure VI-3). The southwest area is composed of several interconnected basins and is suitable for the storage of materials which will ultimately be reclaimed. With a berm elevation of 3400 feet and the angle of repose of dumped material is 34° (1:1.5), has a capacity of 48 millions tons, and a mean dump depth of 300 feet; however, a significantly greater tonnage can potentially be stored in this dump.



WASTE DISPOSAL AREAS

Site preparation costs for these two areas have been included in preproduction costs.

VII

CONCENTRATOR AND SURFACE PLANT

INTRODUCTION

A preliminary study of the design, capital and operating costs of a 10,000 TPD concentrator and related facilities was prepared by Kilborn Engineering (B.C.) Limited. Most of the information contained in this chapter is taken from their report. The cost adjustments for a 15,000 TPD plant are based on expansions for which allowance was made in the original scheme, thus minimizing the additional capital expenditure involved. Costs are based on conditions prevailing in February, 1974.

GENERAL DESCRIPTION

Ore will be delivered by 100-ton truck to a 54 by 74 inch gyratory crusher. The crusher was selected for compatibility with the trucks, and to avoid production losses due to bridging which have been experienced with smaller crushers. The crusher product will be screened at 5/8 inch. The undersize will be conveyed directly to the fine ore storage and the oversize will be conveyed to a 15,000 live-ton stockpile. The fine crushing plant will incorporate a 7 foot secondary cone crusher in open circuit, and two 7 foot tertiary cone crushers in closed circuit with screens to deliver 5/8 inch material to fine ore storage in an A-frame structure. Variable speed conveyors from the fine ore storage will feed two parallel and similar grinding circuits each consisting of a rod and a ball mill. The cyclone overflows will be distributed to two parallel banks of rougher and scavenger flotation cells. The scavenger tailings will flow to waste and the combined rougher and scavenger concentrates will be re-ground in a regrind ball mill. The regrind cyclone overflow will be distributed to cleaner and recleaner cells. The finished concentrate will be thickened, filtered and dried in conventional equipment and conveyed to a 3,000 ton concentrate storage shed.

For a 15,000 TPD plant the same circuit layout is planned with an increase in size of the grinding mills, pumps and dewatering circuit. The number of flotation cells will be increased to accommodate the additional tonnage. The service complex will be adjacent to the concentrator and will house the office, engineering, warehousing and general plant services. The truck shop will service the pit equipment and will house office and dry facilities for the mine personnel.

DESIGN CRITERIA

Design criteria have been drawn from the metallurgical projections, mineral reserves and operating experience from comparable developments in British Columbia. The general operating design parameters are shown in Table VII-1.

Grinding Circuit - Rod Mill

Work Index - Wi = 22.2

R. M. Discharge 80% -12 mesh = 80% -1,400 microns

R. M. Feed 80% -1/2 in. = 80% -16,000 microns

$$W = \frac{10Wi}{\sqrt{P}} - \frac{10Wi}{\sqrt{F}}$$

F = Feed size in microns, 16,000

P = Product size in microns, 1,400

W = Power in KWH/Ton

$$= \frac{222}{\sqrt{1,400}} - \frac{222}{\sqrt{16,000}}$$

$$= \frac{222}{37.5} - \frac{222}{127}$$

= 5.92 - 1.75 = 4.17 kilowatt hours per ton

10,000 TPD

Feed Rate $\frac{10,000}{.96 \times 24}$ = 434 tons/hour

Power Required 434 x 4.17 = 1810 kilowatts

Equivalent horsepower = 1810 x 1.34 = 2425 HP

Two 13 ft. dia. x 18 ft. mills with 1650 HP motors were selected for preliminary feasibility studies prior to the completion of grindability tests.

Table VII-1

OPERATING DESIGN PARAMETERS

	10,000 TPD	15,000 TPD
Concentrator Capacity		
Per year	3.65 Million Tons	5.475 Million Tons
Per day	10,000 Tons	15,000 Tons
Plant Availability	96%	96%
Production Schedule		
Primary crushing		
Feed rate ST per week	70,000	105,000
Shifts per week	13	15
Nominal crushing rate TPH	900	1,077
Fine crushing		
Feed rate ST per week	60,200	90,300
Shifts per week	12	18
Manual crushing rate	800	800
Coarse ore stockpile capacity	15,000 Tons	15,000 Tons
Fine ore storage capacity	10,000 Tons	15,000 Tons
Grinding		
Required grind	60% minus 200 mesh	60% minus 200 mesh
Rod mills	13' dia. x 18'	13½' dia. x 20'
Ball mills	16½' dia. x 20'	16½' dia. x 23'

15,000 TPD

Feed Rate $\frac{15,000}{.96 \times 24}$ = 651 tons/hour

Power required 651 x 4.17 = 2715 kilowatts

Equivalent horsepower 2715 x 1.34 = 3638 HP

Two 13-1/2 ft. dia. x 20 ft. mills with 2000 HP motors were selected.

Grinding Circuit - Ball Mill

Work Index 14.4

Finished Grind 60% -200 mesh = 60% -74 microns or 80% -138 microns

Feed Size 80% - 12 mesh = 80% -1,400 microns

$$W = \frac{10\ Wi}{\sqrt{P}} - \frac{10\ Wi}{\sqrt{F}}$$

$$= \frac{144}{\sqrt{138}} - \frac{144}{\sqrt{1400}}$$

$$= \frac{144}{11.8} - \frac{144}{37.5}$$

= 12.20 - 3.84 = 8.36 kilowatt hours per ton

10,000 TPD

Feed Rate = 434 tons/hour

Power required = 434 x 8.36 = 3628 kilowatts

Equivalent horsepower = 3628 x 1.34 = 4861 HP

Two 16-1/2 ft. dia. x 20 ft. mills with 3400 HP motors were selected for preliminary feasibility studies prior to the completion of grindability tests.

15,000 TPD

Feed Rate = 651 tons/hour

Power required = 651 x 8.36 = 5442 kilowatts

Equivalent horsepower = 5442 x 1.34 = 7292 HP

Two 16-1/2 ft. dia. x 23 ft. mills with 4000 HP motors were selected.

Electrical

Design criteria would be the same for both 10,000 and 15,000 TPD.

1. Incoming service from B. C. Hydro at 138 K.V.; 20 MVA transformer bank for stepdown 138/4.16 K.V.
2. Main distribution voltage 4160 volts.
3. Secondary utilization voltage 600 volts.

Buildings

1. The crusher and conveyor buildings will be single skin metal siding and roof supported by structural steel constructed over reinforced concrete foundations. These buildings will be unheated.
2. The concentrator, service complex and truck shop buildings will have insulated metal walls and roof with structural steel frame on reinforced concrete foundations. Interior partitions will be concrete block.

Equipment Selection

The selection of equipment is based on conventional methods used in concentrator plants today. No specific manuracturers have been indicated although the costs have been based on quotations from various suppliers.

Before final selection is made, many alternatives would be studied, such as:

- Autogenous vs. conventional grinding
- Ore bins vs. "A" frame fine ore storage
- Possible reduction of building heights

The size of equipment is based on the throughput, work index of the ore and experience derived from other plants of comparable capacity.

Kilborn's preliminary general arrangement drawings PR5, PR6, PR7, PR8, PR9, PR10, PR11, PR12, and PR13 for a 10,000 TPD plant are appended.

MILL FLOWSHEETS

Two preliminary mill flowsheets are appended:

- PR3 - Crushing and Grinding Flowsheet
- PR4 - Flotation, Dewatering and Drying Flowsheet

These flowsheets for a 10,000 TPD plant would be altered during final engineering depending upon the plant size selected and the results of further investigations.

CAPITAL COST

Plant capital cost estimates are compiled in Tables VII-2 and VII-3. Fine crushing and screening includes conveying and coarse ore storage. Fine ore storage may be in bins or an "A" frame structure. Concentrator building costs include concentrate storage. The water supply and reclamation and tailings disposal include pipelines, which amount to a major proportion of the costs. The figure shown in the "building" column for miscellaneous on-site services is the site preparation cost. The capital costs for a 15,000 TPD plant are the same as for 10,000 TPD in certain areas such as primary crushing, service complex, truck shop, and cold storage warehouse. The remainder of the facilities entail variably increased costs for scaling up from 10,000 TPD to 15,000 TPD.

Increments: 10,000 TPD to 15,000 TPD

Fine Crushing and Screening:
Minor changes on screens and conveyors; additional cost = $100,000.

Fine Ore Storage:
Allow for 50% increase in building capacity = $426,000

Concentrator:
Larger ball and rod mills with correspondingly larger pumping and classification equipment, 50% increase in flotation capacity, including a building addition of 3300 sq. ft., and a proportionate increase in thickening, filtering and drying capacity. Total additional cost = $922,000.

Table VII-2

CAPITAL COSTS - 10,000 TPD PLANT
(x $1,000)

ITEM	BUILDING	EQUIPMENT	TOTAL
Primary crushing	886	1,080	$ 1,966
Fine crushing & screening	1,708	1,642	3,350
Fine ore storage	956	618	1,574
Concentrator	2,179	4,005	6,184
Service complex	932	125	1,057
Truck shop	835	54	889
Fresh water supply & distribution	85	326	411
Process water reclamation			392
Tailings disposal			237
Miscellaneous on-site services	1,056	718	1,774
Electrical			4,404
Instrumentation			295
Freight			321
Cold storage warehouse			22
Construction camp			2,288
Construction overheads & management			1,734
Engineering & purchasing			2,525
Total Capital Cost			$29,424

Table VII-3

CAPITAL COSTS - 15,000 TPD PLANT
(x $1,000)

ITEM	BUILDING	EQUIPMENT	TOTAL
Primary crushing	886	1,080	$ 1,966
Fine crushing & screening	1,708	1,742	3,450
Fine ore storage	1,382	618	2,000
Concentrator	2,373	4,733	7,106
Service complex	932	125	1,057
Truck shop	835	54	889
Fresh water supply & distribution	85	446	531
Process water reclamation			442
Tailings disposal			287
Miscellaneous on-site services	1,157	817	1,974
Electrical			4,654
Instrumentation			324
Freight			401
Cold storage warehouse			22
Construction camp			2,510
Construction overheads & management			1,908
Engineering & purchasing			2,761
Total Capital Cost			$32,282

Fresh Water Supply and Distribution:
The building will remain unchanged. The pump and pipeline capacity are increased at an additional cost of $120,000.

Process Water Reclamation:
The pump and pipeline capacity are increased at an additional cost of $50,000.

Tailings Disposal:
Larger cyclones and pipelines are required - allow $50,000.

Miscellaneous On-Site Services:
Additional excavation and site preparation = $100,000; increased pipeline capacity and fire protection equipment, additional cost = $100,000.

Electrical:
Increased size of grinding mill motors, increased feeder and switch gear sizes, more flotation motors, increased size of pump motors, total additional cost = $250,000.

Instrumentation:
Additional flotation controls, wiring and piping; cost increment = $29,000.

Freight:
A 25% increase in total freight charges = $80,000.

Construction Camp:
10% of the basic capital cost; cost increment = $222,000.

Construction Overhead:
7.6% of the basic capital cost; cost increment = $174,000.

Engineering and Purchasing:
11% of the basic capital cost; cost increment = $236,000.

OPERATING COST

The total operating costs for crushing and milling at 10,000 TPD are $4,082,000 per year or 1.118 $1.18/ton milled. The total operating costs for crushing and milling at 15,000 TPD are $5,594,000 per year or $1.022/ton milled. These estimates include the cost of tailings disposal. The operating and repair supply requirements are directly related to the tonnage throughput but operating labour is the same for either 10,000 TPD or 15,000 TPD. The total cost of repair and maintenance labour increases for 15,000 TPD, but not in direct proportion to the tonnage increase. The details of these estimates are shown in Tables VII-4 and VII-5.

POWER

Power supply studies for a 10,000 TPD plant were initiated in October, 1973, at a preliminary meeting with representatives of the British Columbia Hydro and Power Authority. The following alternative sources of power have been considered:

1. On-site diesel electric units with 20,000 KVA total capacity to operate throughout the life of the operation.
2. B. C. Hydro power supplied via transmission line from Houston, B. C.
3. B. C. Hydro power supplied via transmission line from the Aluminum Company of Canada system at Kemano.
4. On-site diesel power for 1-1/2 years, then B. C. Hydro power from Houston, for the balance of the mine life.

Hydro Power Operating Costs

Cost data are compiled from correspondence with B. C. Hydro and Power Authority and M. A. Thomas and Associates, and from current operating experience at Granisle Copper Limited.

Power requirements for 10,000 TPD and 15,000 TPD plants are compiled in Tables VII-6 and VII-7 respectively. Operating costs for Hydro power supply for both plant sizes are shown in Table VII-8. As an allowance for 138/4.16 KV - 20MVA transformer bank is included in the plant electrical capital cost estimate, there should be no rental charge on transformers, and metering should be on the 138 KV side. However, B. C. Hydro policy in the past has been to allow the customer to pay for the 138/4.16 KV - 20 MVA transformer facility, then

Table VII-4

CRUSHING AND MILLING OPERATING COSTS
x $1000

10,000 TPD	OPERATING LABOUR		OPERATING SUPPLIES		REPAIR & MAINT. LABOUR		REPAIR & MAINT. SUPPLIES		TOTAL	
	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton
Crushing	164	.045	62	.017	120	.032	85	.023	431	.118
Grinding,	365	.100	1,652	.453	79	.022			2,096	.574
Flot. & Dewater.			437	.120	79	.022	178	.049	694	.190
General Service & Supervision (Including Power)	217	.060	565	.155	79	.022			861	.236
TOTALS	746	.205	2,716	.745	357	.098	263	.072	4,082	1.118

Table VII-5

CRUSHING AND MILLING OPERATING COSTS

x $1000

15,000 TPD	OPERATING LABOUR		OPERATING SUPPLIES		REPAIR & MAINT. LABOUR		REPAIR & MAINT. SUPPLIES		TOTAL	
	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton
Crushing	164	.030	93	.017	125	.023	128	.023	510	.093
Grinding,	365	.066	2,479	.453	83	.015			2,927	.535
Flot. & Dewater.			657	.120	83	.015	267	.049	1,007	.184
General Service & Supervision (Including Power)	217	.040	850	.155	83	.015			1,150	.210
TOTALS	746	.136	4,079	.745	374	.068	395	.072	5,594	1.022

Table VII-6

POWER REQUIREMENTS FOR 10,000 TPD

AREA	AVERAGE HRS. RUNNING P/D	LOAD FACTOR	H.P. (CONNECTED)	KWH PER DAY
Pit*				
Shovels*	13	0.75	1,200	6,234
Drills*	13	0.75	600	3,117
Pumping*	4	0.80	200	341
Miscellaneous*	15	0.80	150	959
Concentrator				
Crusher - primary*	11	0.70	550	2,257
- secondary	12	0.80	1,250	8,952
Grinding	24	0.95	10,400	176,370
Flotation & Dewatering	24	0.95	1,200	20,411
Mill Pumping	24	0.45	1,500	12,085
Lighting & Miscellaneous	24	0.90	300	4,834
Ancillary Buildings				
Power & Lighting	24	0.50	850	7,609
Water System				
Fresh Water	24	0.40	1,000	7,161
Reclaim	24	0.40	800	5,729
TOTALS			20,000	256,159

*5 days per week demand - averaged to 7 days

Expected demand - 14,000 KVA

Consumption per month - 7,685,000 KWH

Table VII-7

POWER REQUIREMENTS FOR 15,000 TPD

AREA	AVERAGE HRS. RUNNING P/D	LOAD FACTOR	H.P. (CONNECTED)	KWH PER DAY
Pit*				
Shovels*	20	0.75	1,200	9,591
Drills*	16	0.75	600	3,836
Pumping*	4	0.80	200	341
Miscellaneous*	15	0.80	150	959
Concentrator				
Crusher - primary*	18	0.70	550	3,693
- secondary	17	0.80	1,250	12,682
Grinding	24	0.95	12,000	205,029
Flotation & Dewatering	24	0.95	1,800	30,616
Mill Pumping	24	0.45	2,250	18,128
Lighting & Miscellaneous	24	0.90	300	4,834
Ancillary Buildings				
Power & Lighting	24	0.50	850	7,609
Water System				
Fresh Water	24	0.40	1,250	8,952
Reclaim	24	0.40	1,000	7,161
TOTALS			23,400	313,431

*5 days per week demand - averaged to 7 days

Expected demand - 16,380 KVA

Consumption per month - 9,403,000 KWH

charge him rental for it as well as increasing the charges by a constant multiplier to cover transformer power losses. This constant multiplier effectively converts metering on the secondary side to its corresponding value on the primary side; the multiplier factor is 1.025 at 14,000 KVA demand or under and 1.032 at over 14,000 KVA demand. Costs per KWH for both ways of metering are shown on Table VII-8; the higher operating cost is included in onward considerations. Hydro power costs are based on the 1821' rate schedule.

Comparison of Alternatives

Capital and operating costs for the four power supply alternatives were estimated for the 10,000 TPD plant size. Capital costs would be the same for 15,000 TPD and the operating costs are used only for comparing the alternatives.

Table VII-8

COST OF HYDRO POWER

		10,000 TPD	15,000 TPD
Demand	=	14,000 KVA	16,380 KVA
Consumption	=	7,685,000 KWH	9,403,000 KWH
Demand charge:			
14,000 x $2.00	=	28,000	
16,380 x $2.00	=		32,760
Balance			
(7,685,000 - 14,000) .003	=	23,013	
(9,403,000 - 16,380) .003	=		28,160
Sub-total	=	51,013	60,920
5% Provincial Sales Tax	=	2,551	3,046
Monthly Cost - primary metering	=	53,564	63,966
Cost/KWH - primary metering	=	.697¢	.680¢
Conversion - secondary to primary metering:			
51,013 x 1.025	=	52,288	
60,920 x 1.032	=		62,869
Transformer rental	=	3,021	3,021
Sub-total	=	55,309	65,890
5% Provincial Sales Tax	=	2,765	3,295
Monthly cost - secondary metering	=	58,074	69,185
Cost/KWH - secondary metering	=	.756¢	.736¢

Alternative 1:

On-site diesel for full mine life. Five 4,000 KVA units, four to operate under normal load and one spare; capital cost includes building and heavy fuel oil facilities.

Capital Cost: $200/KVA	$ 4,000,000
Operating Cost:	
Operating supplies @ 2¢/KWH 92,220.--- KWH/year	1,844,000
Labour (operating, repair & maintenance)	120,000
Depreciation & interest @ 20% of capital cost	800,000
Less: recovery of waste heat	- 95,000
TOTAL	$ 2,669,000
Present value of cost over 15 years discounted 10% =	$20,300,000

Alternative 2:

B. C. Hydro transmission line from Houston - 138 KV

Capital Cost:	
Transmission line from Houston - 74 miles (27 December, 1973 estimate)	$ 4,873,000
Townsite stepdown & distribution	450,000
less refund: 252 customers @ $250	- 63,000
TOTAL	$ 5,260,000
Operating Cost:	
Depreciation & interest @ 20% of capital cost	$ 1,052,000
Annual power billing under B. C. Hydro schedule 1821: 12 x $58,074	697,000
TOTAL	$ 1,749,000
Present value of operating cost over 15 years of discounted at 10%	= $13,303,000

Alternative 3:

Transmission line from Kemano - 288 KV. Line and power costs have not been fully investigated and are derived from early discussions with B. C. Hydro.

Capital Cost:

Main transmission line: 37 miles @ 50% greater /mile cost than above	$ 3,655,000
Additional cost of stepdown from 288 KV	90,000
Townsite line: 20 miles @ $40,000 /mile	800,000
Townsite stepdown & distribution	450,000
less refund: 252 customers @ $250	- 63,000
TOTAL	$ 4,932,000

Operating Cost:

Depreciation & interest @ 20% of capital cost	$ 986,000
Annual power billing under B. C. Hydro schedule 1821: 12 x $58,074	697,000
TOTAL	$ 1,683,000

Present value of operating cost over 15 years discounted at 10% = $12,801,000

Alternative 4:

Diesel for 1 1/2 years, then transmission line from Houston. Half the capital cost of the diesel plant is recovered and this is applied against the cost of the transmission line.

Capital Cost:

Diesel plant	$ 4,000,000
Net cost of Hydro facility discounted @ 10%	2,817,000
TOTAL	$ 6,817,000

Operating Cost:

1 1/2 years @ $2,669,000 discounted @ 10%	$ 3,634,000
13 1/2 years @ $1,749,000 discounted @ 10%	11,103,000
TOTAL	$14,737,000

Present value of operating cost over 15 years discounted at 10% = $14,737,000

Power Cost Summary

The comparative costs of the four alternatives for power supply are summarized in Table VII-9. On a total cost basis, Alternatives 1 and 4 can be ruled out and Alternative 3 is marginally lower than 2. However, the terrain and climatic conditions between Kemano and Huckleberry are severe to extreme, so that line construction costs are likely to be higher than estimated and interruptions due to line problems are to be expected. Favourable terrain along the Houston route coupled with proximity to the main access road and relatively mild climatic conditions will ensure minimum of construction and operating problems. Consequently, construction should take less than three years and capital costs should be lower than $66,000/mile as currently estimated. Thus, Alternative 2 appears to be the most favourable and those corresponding cost figures are used in all power considerations.

A payment of $10,000 to B. C. Hydro will be required to initiate detailed engineering studies to establish the final route and a reliable capital cost.

ANCILLARIES

Ancillary equipment and services include tools and equipment for the plant mechanical, electrical and services departments, service vehicles, office equipment, heating fuel distribution system, communications equipment, fire protection and first aid equipment, and four miles of new plant access road from the east end of Sweeney Lake. Capital costs of installed ancillaries including road building costs are compiled in Table VII-10. Tools and equipment for the mechanical department include those required for the mechanical section of the service building, crusher, mill tool crib and truck shop. Electrical department tools and equipment include shop equipment, instrumentation test equipment, general test equipment, and portable equipment. The service department will include a complete carpenter shop, miscellaneous mobile equipment for plant site and tailings area use, lifting devices and portable tools. The service vehicles include buses, and could be leased rather than owned, at an additional operating cost of $75,000 per year. Office equipment includes furniture, secretarial, accounting, and engineering

Table VII-9

POWER COST COMPARISON
(x $1000)

ALTERNATIVE:	1	2	3	4
Capital	4,000	5,260	4,932	8,545
Operating	2,699	1,749	1,683	2,669(1) 1,749(2)
PV of 15 years operating	20,300	13,303	12,801	14,737
Total - Capital & 15 years operating	24,300	16,563	17,733	23,282

NOTE:

(1) Operating cost for the first 1 1/2 years.

(2) Operating cost after the first 1 1/2 years.

Table VII-10

CAPITAL COSTS OF ANCILLARIES
(x $1000)

Mechanical department - tools & equipment		202
Electrical department - tools & equipment		18
Service department - tools & equipment		242
Service vehicles		172
Office equipment		58
Heating fuel distribution system		34
Communications equipment		180
Fire protection & first aid equipment		20
Sub-total		926
Plant Access Road: 4 miles		
clearing and burning	40	
rough grading	60	
rock base	110	
gravelling	80	
culverts & ditching	30	
Sub-total		320
TOTAL		1,246

equipment. Heating fuel distributions system includes tanks, pumps, pipe and fittings. Communications equipment requirements have not been studied in detail; the capital cost figure is based on costs incurred at Granisle Copper Limited.

CONCENTRATE HANDLING

Concentrate handling considerations were based on the 10,000 TPD plant size, which would produce approximately 50,000 tons of concentrates per year. The facilities outlined will be adequate for the additional output of a 15,000 TPD plant, so that the capital cost will be unchanged and operating costs per ton will also be the same for both plant sizes. Storage facilities for 3,000 tons of concentrate and a front-end loader at the mine site have been included in the plant capital cost. The concentrates will be hauled to Houston on a contract basis, where loading and storage facilities for 1,800 tons at a rail siding are to be provided. Consideration was given to hauling to Topley, an additional distance of 18 miles, and using the Granisle rail siding; but the additional operating cost of $1.40/ton of concentrate, plus the cost of increased storage space at Topley, effectively outweigh the cost saving in eliminating the Houston facility. From Houston, the concentrates are to be hauled in covered cars to Prince George by Canadian National Railways and onward to Vancouver by British Columbia Railway. Storage facilities at Vancouver for 10,000 tons of concentrates would be provided by Vancouver Wharves at an estimated cost of $200,000 and paid for by an additional 98¢/ton charge which is included in operating costs. This charge would apply until the capital cost of the facility is written off.

Capital Cost

Houston Facility:

Land - 3 acres @ $8,000	-	$ 24,000	
Building - 66,000 sq. ft. @ $20	-	132,000	
Front-end loader - 2-1/2 yars	-	45,000	
Car puller, etc.	-	12,000	
Scale - installed	-	27,000	
TOTAL			$240,000

TOTAL	c/f	$240,000
Rail car covers: 51 @ $3,530		180,000
TOTAL		$420,000

Operating Cost

Truck haulage: 70 miles @ 12¢/ton	$ 8.40
Loading	.40
Rail freight	9.75
Handling cost at Vancouver	3.76
Company expense: operating, repair & maintenance to buildings	1.80
TOTAL cost per ton moved	$24.11

Assuming that the concentrates contain 7% moisture, $24.11/ton moved would be equivalent to $26.00/short dry ton. These costs are based on current costs at Granisle and other comparable operations with an allowance for rail car covers and other improved handling facilities.

If facilities were available for shipping concentrates at either Prince Rupert or Kitimat, the operating cost could be reduced by approximately $5.00/short dry ton.

VIII

WATER, TAILINGS AND RECLAMATION

WATER SUPPLY AND DISTRIBUTION

The concentrator will operate principally on water recycled from the tailings disposal area. It is estimated that 80% of the water contained in tailings pulp will be reclaimed. To allow for interruptions in the supply of reclaimed water due to relocation of the pump barge, and initial filling of the tailings pond; fresh-water capacity will be sufficient to permit the full operation of the concentrator. Three matched vertical turbine pumps located at a station on Tahtsa Reach (nominal elevation 2,790 ft.) will supply a 480,000-U.S.-gal. tank located near the plantsite at 3,425 feet elevation. Because the water level at the pumphouse can vary as much as 30 feet, special design will be incorporated. The large tank at the plantsite will have a 300,000-U.S.-gal. fire reserve, a 150,000-U.S.-gal. process-makeup and pump-gland reserve and a 30,000-U.S.-gal. domestic water capacity. Water from this tank will supply fire and domestic water systems as well as providing make-up water supply for a 150,000-U.S.-gal. process-water head tank located at 3,325 feet elevation, from which all process water would be drawn. A 150-U.S.-gpm. pump located near the large fresh-water supply tank will feed an 80,000-U.S.-gal. head tank at 3,525 feet elevation. This tank, with a 60,000-U.S.-gal. fire reserve will supply the truck shop. The reclaim system will consist of two barge-mounted vertical turbine pumps located at the tailings pond which supply the 150,000-U.S.-gal. process-water tank. The layout and schematic design are shown in Drawing PR 13 in the Appendix.

Equipment Requirements

A. Required for both 10,000 and 15,000 ton per day plants:

1. Storage Tanks

   - 1 @ 480,000-U.S.-gal.
   - 1 @ 150,000-U.S.-gal.
   - 1 @ 80,000

2. Pumps

   - 1 @ 150-U.S. gpm. & 100 ft. head

3. Distribution Lines

   - truck shop domestic (4-in.-dia.)
   - truck shop fire (10-in.-dia.)
   - truck shop tank supply (4-in.-dia.)
   - plantsite domestic (8-in.-dia.)
   - crusher cooling (4-in.-dia.)
   - plantsite fire (10-in.-dia.)

B. Required for 10,000 ton per day plant

1. Pumps

   - 3 @ 1,550-U.S.-gpm. & 725-ft. head (fresh water)
   - 2 @ 2,000-U.S.-gpm. $ 500-ft. head (reclaim water)

2. Distribution Lines

   - reclaim pumps to process tank (16-in.-dia.)
   - fresh-water pumps to fresh-water tank (16-in.-dia.)
   - process tank to concentrator (16-in.-dia.)

C. Required for 15,000 ton per day plant

1. Pumps

   - 3 @ 2,250-U.S.-gpm. & 725-ft. head (fresh water)
   - 2 @ 3,000-U.S.-gpm. & 500-ft. head (reclaim water)

2. Distribution Lines

   - reclaim pumps to process tank (18-in.-dia.)
   - fresh-water pumps to fresh-water tank (18-in.-dia.)
   - process tank to concentrator (18-in.-dia.)

TAILINGS DISPOSAL

An area immediately to the west of the proposed millsite, behind a low ridge parallel to the shore of Tahtsa Reach, drains westerly for about two miles through a series of connected swampy ponds. The following assessment of tailings disposal requirements must be regarded solely as an indication of the feasibility of storing tailings in this area. A specific proposal cannot be developed without soils investigations and studies by geotechnical consultants. An initial tailings disposal layout has been prepared and is shown in Figures VIII-1 and VIII-2 in the Appendix. Three tailings ponds numbered with decreasing elevation and increasing distance from the mill have been laid out. The ultimate crest of the No. 1 dam will be 65 feet below the level of the mill floor. The storage capacity of this dam will be sufficient to fulfill most of the requirements of the operation. Production beyond the capacity of No. 1 dam will be placed in areas 2 and 3. The following design criteria were used, based on experience at other similar operations.

1. The angle of repose of rock fill is 35.5 degrees (1:1.4). The tonnage factor for this material is 18 cu. ft. per ton.
2. The ultimate slope of cycloned-sand dam faces is 16 degrees (1:3.5). The tonnage factor for cycloned sand is 22 cu. ft. per ton.
3. Uncycloned tailings have a tonnage factor of 24 cu. ft. per ton, corresponding to a density of 83.3 lb. per cu. ft.
4. The fine fraction of tailings (slimes) have a tonnage factor of 25 cu. ft. per ton (80 lb. per cu. ft.). The angle of repose of spiggotted tailings is 10% close to the point of discharge and 5% further away. In the zone of settling, the angle of repose is 0.
5. At any point in the pond, the minimum distance from the sand-slime interface to the water is 4 times the dam height at that point.

Tailings Dam Design

Before mill operation commences, it will be necessary to do some preparatory work for initial tailings impoundment in the No. 1 pond area. The south wall of this basin has been broadened by a deep stream channel draining southerly into Tahtsa Reach. A rock fill structure will be necessary to block this channel and provide a core for the tailings dam. The No. 1 starter dam with associated dams, seepage control structures, and drains is estimated to require 1.8 million tons of rock fill. From the starter dam, construction of the tailings dam can proceed following the Centreline Method, with two-stage separation of sands from slimes. A final slope of the cycloned-sand face of 16 degrees

(1:3.5) is assumed. In order to attain this slope, it is required that 36% of the tailings solids be separated as sands. Experience at other British Columbia operations show that a maximum of 40% sands can be recovered in the short-run. Therefore, the 36% sand requirement of the preliminary design represents a difficult target. Further refinements in dam location, beach requirements, and dam-crest freeboard requirements should bring the sands requirement to a more practicable level. The calculated capacities are based on the assumption that the ultimate dam-crest elevation would be uniform; sands ratios for No. 2 and No. 3 dams are better than that of No. 1 dam. At the point of greatest dam depth, the water level will be 50 ft. below and 1,000 ft. distant from the dam crest. Average water depth will be 25 feet.

Clearing and stripping to glacial till will be required in the area immediately underlying the sand portion of the ultimate dam, a total of 200 acres. The tailings-pond, an additional 400 acres, will require logging and burning of trees. Because of high annual precipitation and heavy spring run-off, it will be necessary to construct a system of diversion ditches to prevent surface run-off from reaching the pond. Since the dam is to be constructed of cycloned tailings sands and because of anticipated seepage through the dam; it will be necessary to construct catchment basins at the base of the dam, from which water will be returned to the tailings pond.

Capital Costs

These costs apply to both 10,000 TPD and 15,000 TPD plants.

1. Earthwork costs are included in preproduction costs.
2. Tailings lines, reclaim lines, pumps and hydroclones are included in the plant capital cost estimates.
3. Costs not included elsewhere are:

- Insulated enclosure for first stage hydroclones	-$10,000
- Pumps to feed second-stage hydroclones (10-X10 SRL unit, installed)	-$10,000
- Catchment and seepage-return system	-$40,000
TOTAL COSTS	$60,000

RECLAMATION

The area to be disturbed by the mine, plantsite, waste dumps, and tailings ponds is 1,150 acres. Before the minesite can be abandoned, it will be necessary to ensure that banks, dams and dump faces have been graded to a stable profile and that an adequate self-sustaining cover of vegetation has been established.

During the early years of mining, the reclamation program will be confined to stablizing road cuts and the plantsite fill area. At the same time, a research program will be conducted to establish optimal mixes of plant species for reclamation purposes. Although some experimentation will be conducted with commercial legumes and grasses, particular emphasis will be devoted to the development of indigenous grasses, shrubs, and trees which are amenable to reclamation work.

As dumps, pit areas and tailings ponds become completed, they will be stablized and seeded with plant mixes which have been developed in the research program. Monitoring of the relative success of various plant species under differing environmental conditions (i.e. soil type, fertilization, pre-existing species) will be done on a routine basis in order to determine the best reclamation practices. These practices will be employed on a large scale basis after the completion of mining.

The objectives of the reclamation program will be as follows:

1. minimizing erosion and other damage to the environment;
2. ensuring a pleasing and natural appearance of the reclaimed land;
3. minimizing potential ecological unbalance by using indigenous species where practicable.

IX

PERSONNEL

Introduction

The calculation of labour costs for the Huckleberry project is based on the current Granisle rates which are competitive for the (British Columbia) mining industry.

An overtime factor is applied which includes scheduled overtime of two hours per week and 10% non-scheduled overtime for those employees on continuous operation. Overtime factors of 15% are applied to the employees on maintenance work and 10% for mine operating personnel.

The projected escalation of labour costs for the next few years is expected to be in the range of 10% to 15% per year.

Operating Schedules

Several shift schedules have been studied for both the 10,000 TPD and 15,000 TPD operations. It has been concluded that the most practicable schedule is the following:

1) Mining and Primary Crushing - 3 shifts of 5 days per week or a 5 and 2 schedule with weekends off.
2) Secondary Crushing and Milling - 3 shifts of 7 days per week or a 6 and 2 schedule.
3) Mechanical maintenance personnel. Mainly a 5 and 2 schedule with some scheduled for 3 shifts.
4) Salaried and other personnel, excluding shift supervisors - 5 days per week with weekends off.

Tables IX-1 and IX-2 are appended showing the annual labor cost for each job position, including overtime and premium payments. The calculation of annual

Table IX-1

ANNUAL LABOUR COST BY JOBS

OPERATING

JOB	BASE/HR.	PER YEAR SHIFT PREMIUMS	5 & 2 O.T. 10% 6 & 2 O.T. 12.3%	LOADED COST PER HR.	REG.HRS. PER YEAR	COST PER YEAR
MINE						
Shovel Op.	$5.80	$167	$.87	$6.67	2,000	$13,507
Shovel Oiler	4.30	167	.65	4.95	2,000	10,067
Truck Drivers	5.00	167	.75	5.75	2,000	11,667
Support Equip.	5.20	167	.78	5.98	2,000	12,127
Drillers	5.40	167	.81	6.21	2,000	12,587
Helpers	4.25	-	.64	4.89	2,000	9,780
Blaster	5.20	-	.78	5.98	2,000	11,960
Blaster's Helper	4.25	-	.64	4.89	2,000	9,780
CRUSHER						
Primary Crusher	4.70	167	.71	5.41	2,000	10,987
Secondary Crusher	4.85	337	.90	5.75	2,184	12,895
Crusher Helpers	4.25	337	.79	5.04	2,184	11,344
MILL						
Grinding Op.	5.20	337	.96	6.16	2,184	13,790
Flotation Op.	5.20	337	.96	6.16	2,184	13,790
Filter Op.	4.40	337	.81	5.21	2,184	11,716
Reagent Op.	4.40	-	.66	5.06	2,000	10,120
Mill Helpers	4.25	337	.78	5.03	2,184	11,323
Lab Helpers	4.25	-	.64	4.89	2,000	9,780
Tailings Op.	4.25	337	.78	5.03	2,184	11,323
Bucker	4.25	-	.64	4.89	2,000	9,780

Table IX-2

ANNUAL LABOUR COST BY JOBS

REPAIR & MAINTENANCE

JOB	BASE/HR.	PER YEAR SHIFT PREMIUMS	5 & 2 -15% O.T.	LOADED COST PER HR.	REG.HRS. PER YEAR	COST PER YEAR
PLANT						
H.D. Mechanics	$5.80	$167	$1.31	$7.11	2,000	$14,387
Welders	5.80	-	1.31	7.11	2,000	14,220
Apprentices	4.70	-	1.06	5.76	2,000	11,520
Service	4.70	167	1.06	5.76	2,000	11,687
Machinist	5.80	-	1.31	7.11	2,000	14,220
Janitor	4.20	-	.95	5.15	2,000	10,300
Millwright	5.80	-	1.31	7.11	2,000	14,220
Labourer	4.20	-	.95	5.15	2,000	10,300
Carpenter	5.80	-	1.31	7.11	2,000	14,220
Equip. Op.	5.20	-	1.17	6.37	2,000	12,740
Electrician	5.80	-	1.31	7.11	2,000	14,220
WAREHOUSE						
Counterman	4.50	-	1.01	5.51	2,000	11,020

labour costs for each department is shown in Tables IX-3 and IX-4 for hourly paid staff and in Tables IX-5 for monthly paid staff.

Fringe Benefits

The calculation of fringe benefits has been based on those in effect at Granisle Copper Limited, and are competitive for the mining industry in B. C. No segregation is made between hourly paid and salaried personnel. Salaried personnel are not normally paid for statutory holidays but there are other compensating fringe benefits. The total fringe benefit package has been estimated to be 14.6% of annual wages. Details are shown in Table IX-6.

The fringe benefits for the 10,000 TPD operation total:

($2,944,931 x 0.14) or $429,959

The fringe benefits for the 15,000 TPD operation would total:

($3,351,052 x 0.146) or $489,254

Summary

A summary of the manpower requirements and annual wage costs is provided in Table IX-7. The total number of personnel for the 10,000 TPD operation has been estimated to be 222 at a labour cost of $2,944,931 per annum. Fringe benefits would be an additional $429,959.

The total number of personnel for the 15,000 TPD operation has been estimated to be 254 at a labour cost of $3,351,052 per annum. Fringe benefits would be $489,254.

Table IX-3

MANPOWER SCHEDULE - HOURLY

OPERATING

JOB TITLE	COST/YEAR/JOB	10,000 TPD NUMBER OF MEN	10,000 TPD COST/YEAR	15,000 TPD NUMBER OF MEN	15,000 TPD COST/YEAR
MINE					
Shovel Operator	$13,507	4	$ 54,028	6	$ 81,042
Shovel Oiler	10,067	4	40,268	6	60,402
Truck Driver	11,667	16	186,672	24	280,008
Support Equipment	12,127	12	145,524	15	181,905
Drillers	12,587	4	50,348	5	62,935
Helpers	9,780	4	39,120	5	48,900
Blaster	11,960	1	11,960	1	11,960
Blaster Helper	9,780	3	29,340	3	29,340
Sub-Total		48	$557,260	65	$756,492
CRUSHER					
Primary Op.	$10,987	3	$ 32,961	3	$ 32,961
Secondary Op.	12,895	4	51,580	4	51,580
Crusher Helpers	11,344	7	79,408	7	79,408
Sub-Total		14	$163,949	14	$163,949
MILL					
Grinding Op.	$13,790	4	$ 55,160	4	$ 55,160
Flotation Op.	13,790	4	55,160	4	55,160
Filter Op.	11,716	4	46,864	4	46,864
Reagent Op.	10,120	1	10,120	1	10,120
Mill Helpers	11,323	8	90,584	8	90,584
Lab Helpers	9,780	3	29,340	3	29,340
Tailings Op.	11,323	6	67,938	6	67,938
Bucker	9,780	1	9,780	1	9,780
Sub-Total		31	$364,946	31	$364,946
TOTAL		93	$1,086,155		

Table IX-4

MANPOWER SCHEDULE - HOURLY

REPAIR & MAINTENANCE

JOB TITLE	COST/YEAR/JOB	10,000 TPD NUMBER OF MEN	10,000 TPD COST/YEAR	15,000 TPD NUMBER OF MEN	15,000 TPD COST/YEAR
PLANT					
H.D. Mechanics	$14,387	8	$ 115,095	12	$ 172,644
Welders	14,220	5	71,100	6	85,320
Apprentices	11,520	5	57,600	5	57,600
Service	11,687	3	35,061	3	35,061
Machinist	14,220	1	14,220	1	14,220
Janitor	10,300	2	20,600	2	20,600
Millwright	14,220	16	227,520	20	284,400
Labourer	10,300	9	92,700	11	113,300
Carpenter	14,220	3	42,660	3	42,660
Equipment Op.	12,740	6	76,440	6	76,440
Electrician	14,220	7	99,540	8	113,760
Sub-Total		65	$ 852,536	77	$1,016,005
PURCHASING & WAREHOUSE					
Countermen	$11,020	2	$ 22,040	3	$ 33,060
TOTAL		67	$ 874,576	80	$1,049,065

MANPOWER SCHEDULE - SALARY

JOB	COST/YEAR/JOB	10,000 TPD NUMBER OF MEN	10,000 TPD COST/YEAR	15,000 TPD NUMBER OF MEN	15,000 TPD COST/YEAR
ADMINISTRATION					
Manager		1		1	
Mine Superintendent	Allow $25,000	1	Allow $25,000	1	Allow $25,000
Mill Superintendent	per job	1	per job	1	per job
Plant Superintendent		1		1	
Personnel		1		1	
Chief Accountant		1		1	
Chief Engineer		1		1	
Sub-Total		7	$175,000	7	$175,000
Accountant	$14,400	1	$ 14,400	2	$ 28,800
Paymaster	12,000	1	12,000	1	12,000
Clerk	9,500	2	19,000	2	19,000
Stenographers	8,400	2	16,800	2	16,800
Switchboard	8,400	1	8,400	1	8,400
Sub-Total		7	$ 70,600	8	$ 85,000
MILL					
General Foreman	$19,200	1	$ 19,200	1	$ 19,200
Tailings Foreman	15,600	1	15,600	1	15,600
Shift Boss	18,000	4	72,000	4	72,000
Metallurgist	19,800	1	19,800	1	19,800
Process Engineer	20,400	1	20,400	1	20,400
Clerk	9,600	1	9,600	1	9,600
Chief Chemist	18,000	1	18,000	1	18,000
Assayer	15,600	1	15,600	1	15,600
Lab Technician	13,200	2	26,400	2	26,400
Sub-Total		13	$216,600	13	$216,600
MINE					
General Foreman	$19,200	1	$ 19,200	1	$ 19,200
Shift Bosses	18,000	3	54,000	3	54,000
Spare S.B. & Trainer	18,000	1	18,000	1	18,000
Sub-Total		5	$ 91,200	5	$ 91,200
PLANT					
Planners	$14,400	2	$ 28,800	2	$ 28,800
Clerk Typist	9,600	1	9,600	1	9,600
Gen. Mech. Foreman	19,200	1	19,200	1	19,200
Shop Foreman	18,000	2	36,000	2	36,000
Mill Maint. Foreman	18,000	1	18,000	1	18,000
Chief Electrician	19,200	1	19,200	1	19,200
Crusher Maint. Foreman	18,000	-	--	1	18,000
Instrument Mech.	14,400	1	14,400	1	14,400
Services Foreman	15,600	1	15,600	1	15,600
Sub-Total		10	$160,800	11	$178,800
PERSONNEL					
Safety Director	$18,000	1	$ 18,000	1	$ 18,000
Security Men	12,000	4	48,000	4	48,000
Sub-Total		5	$ 66,000	5	$ 66,000
PURCHASING					
Chief Warehouseman	$19,200	1	$ 19,200	1	$ 19,200
Clerk Typist	9,600	2	19,200	2	19,200
Inventory Control	12,000	1	12,000	1	12,000
Sub-Total		4	$ 50,400	4	$ 50,400
ENGINEERING					
Mine Engineer	$19,200	2	$ 38,400	2	$ 38,400
Computer Tech.	18,000	1	18,000	1	18,000
Geologist	18,000	1	18,000	1	18,000
Draftsman	14,400	1	14,400	1	14,400
Surveyors & Tech.	12,000	3	36,000	3	36,000
Clerk	9,600	1	9,600	1	9,600
Surveyors Helpers	9,600	2	19,200	2	19,200
Sub-Total		11	$153,600	11	$153,600

Table IX-6

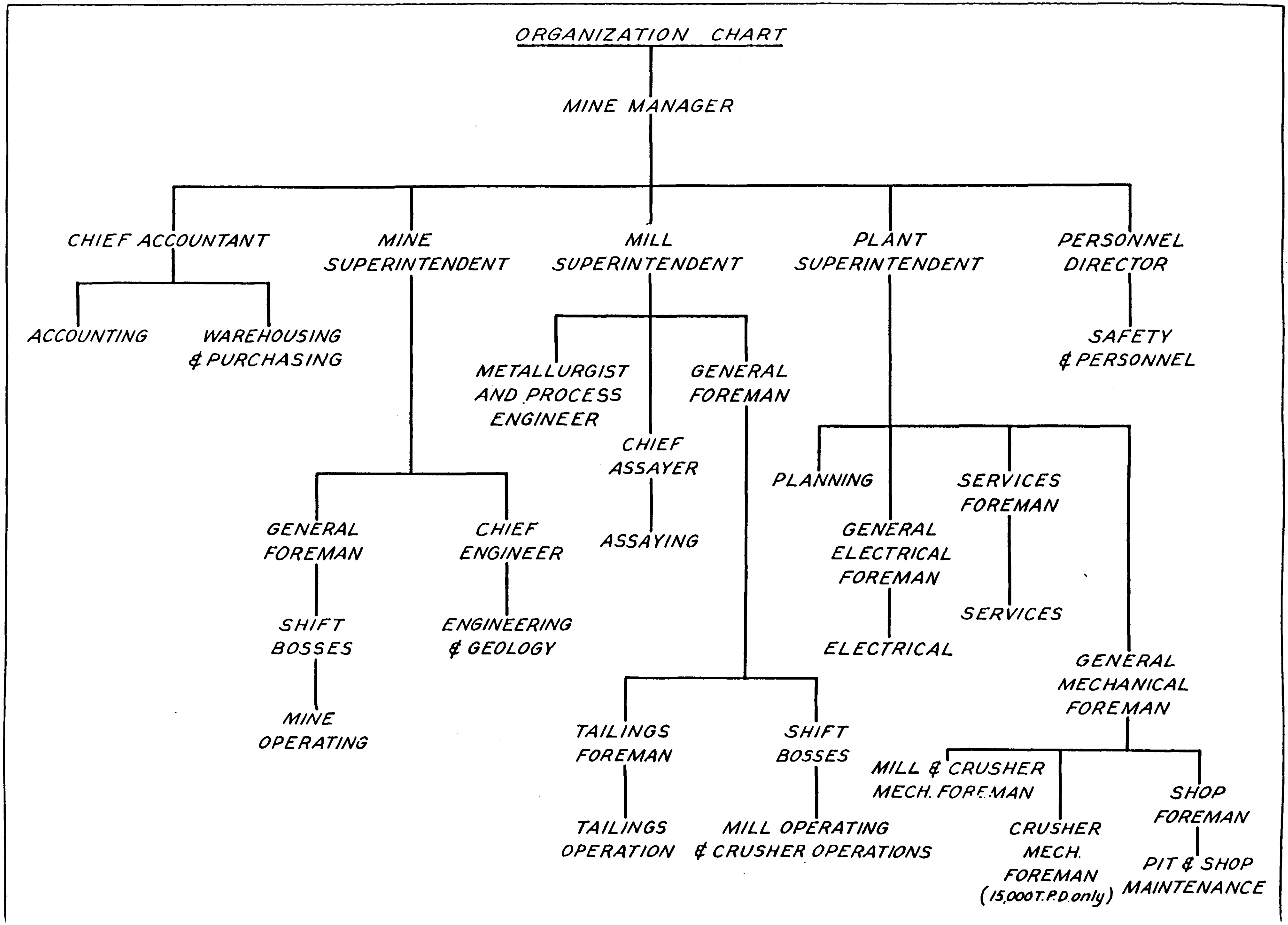
FRINGE BENEFITS

Average annual wage of Salary & Hourly personnel	$13,300
Fringe Benefits in percent of annual wage:	
Statutory Holidays	3.0%
Holiday Pay	6.0%
Canada Pension Plan	.8%
Unemployment Insurance	1.3%
M.S.A.	.6%
Group Insurance	.6%
W.C.B.	1.8%
Silicosis (50% of employees)	.5%
Total	14.6%

Table IX-7

SUMMARY MANPOWER SCHEDULE

DEPARTMENT	10,000 TPD NUMBER OF MEN	10,000 TPD COST/YEAR	15,000 TPD NUMBER OF MEN	15,000 TPD COST/YEAR
HOURLY				
MINE	48	$ 557,260	65	$ 756,492
CRUSHER	14	163,949	14	163,949
MILL	31	364,946	31	364,946
PLANT & WAREHOUSE	67	874,576	80	1,049,065
SUB-TOTAL	160	$1,960,731	190	$2,334,452
SALARY				
ADMINISTRATION	7	$ 175,000	7	$ 175,000
ACCOUNTING	7	70,000	8	85,000
MILL & CRUSHER	13	216,600	13	216,600
MINE	5	91,200	5	91,200
PLANT	10	160,800	11	178,800
PERSONNEL	5	66,000	5	66,000
PURCHASING	4	50,400	4	50,400
ENGINEERING	11	153,600	11	153,600
SUB-TOTAL	61	$ 984,200	63	$1,016,600
TOTALS	222	$2,944,931	254	$3,351,052


ORGANIZATION CHART
MINE MANAGER
CHIEF ACCOUNTANT
MINE SUPERINTENDENT
MILL SUPERINTENDENT
PLANT SUPERINTENDENT
PERSONNEL DIRECTOR
ACCOUNTING
WAREHOUSING & PURCHASING
METALLURGIST AND PROCESS ENGINEER
GENERAL FOREMAN
SAFETY & PERSONNEL
CHIEF ASSAYER
PLANNING
SERVICES FOREMAN
GENERAL FOREMAN
CHIEF ENGINEER
ASSAYING
GENERAL ELECTRICAL FOREMAN
SHIFT BOSSES
ENGINEERING & GEOLOGY
SERVICES
ELECTRICAL
GENERAL MECHANICAL FOREMAN
MINE OPERATING
TAILINGS FOREMAN
SHIFT BOSSES
MILL & CRUSHER MECH. FOREMAN
SHOP FOREMAN
TAILINGS OPERATION
MILL OPERATING & CRUSHER OPERATIONS
CRUSHER MECH. FOREMAN (15,000 T.P.D. only)
PIT & SHOP MAINTENANCE

X

TOWNSITE

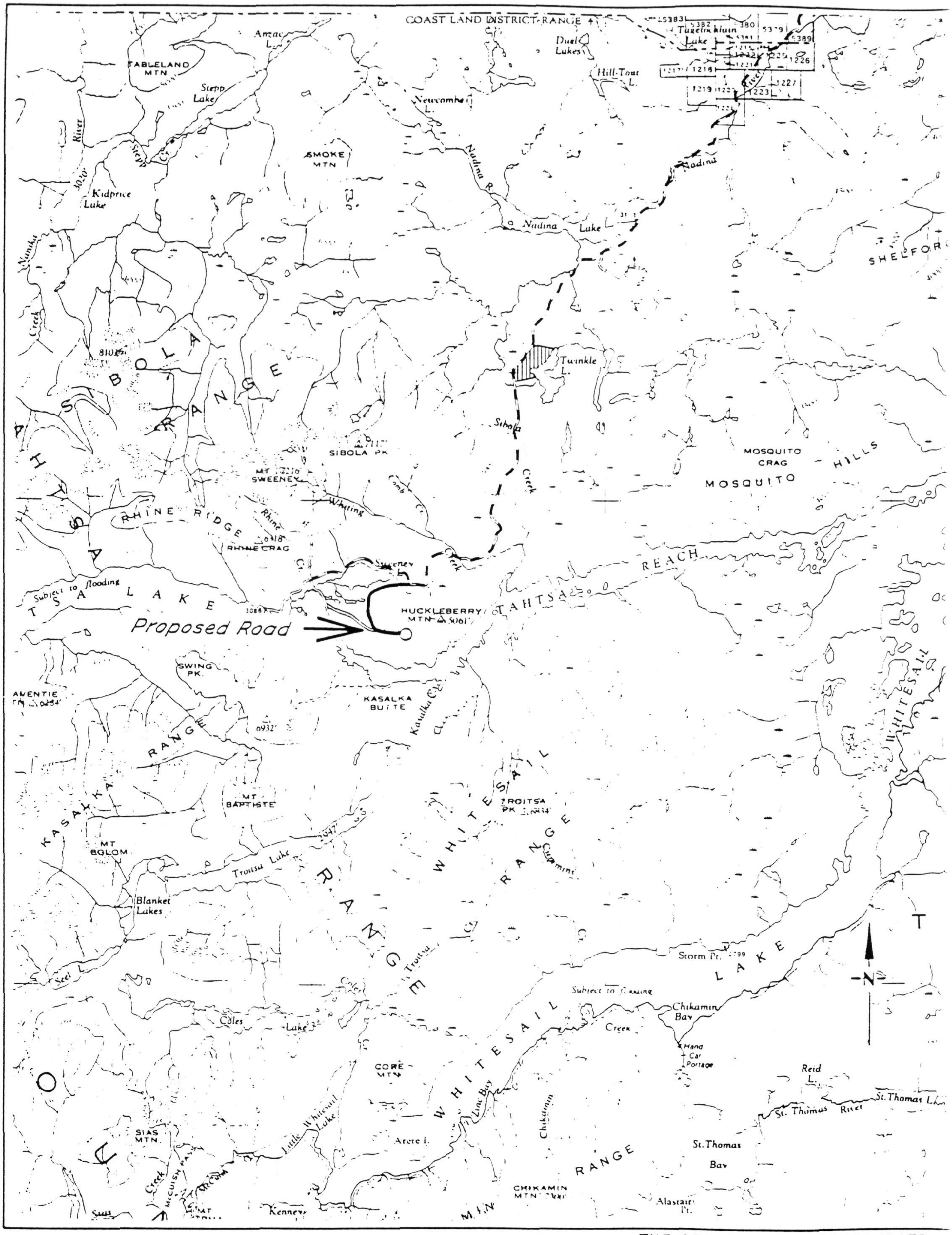
INTRODUCTION

The remote location of the Huckleberry property precludes the establishment of living facilities for operating personnel in any existing community. Experience at several operating mines has shown that a commuting distance of more than 20 miles (30 minutes) results in undue personnel problems. The west shore of Twinkle Lake (Figure X-1) has been chosen as the prospective townsite, since it fulfills the commuting requirement, is physiographically suitable and is located on the access road from Highway 16. Lying beyond the main shadow of the Coast Mountains this area is subject to less than half the precipitation of the minesite area.

The preliminary conceptual design of the townsite is based on the accommodation requirements of the project coupled with terms of reference determined from experience at the Granisle townsite and a series of meetings with government Central Mortgage and Housing Corporation officials and representatives of other industries. Population and capital cost figures are tabulated for both 10,000 TPD and 15,000 TPD operations. Townsite operating costs are effectively the same for both plant sizes. Further investigation of the townsite location and the development of a detailed town plan should be undertaken by the appropriate consultants as part of the final feasibility study.

Population

Ideally the work force should have as high a proportion of married people as possible to provide stability within the project and the community. The high mobility of non-married workers results in rapid turnover and inhibits the development of a community identity. Provision of sufficient accommodation for married personnel will tend to promote the desired population mix; for planning purposes it is assumed 80% of the work force will be married. Townsite services personnel will number 40 and since many of these will be school teachers, store clerks, and RCMP officers, 24 of them will be assumed to be unmarried.



THE GRANBY MINING COMPANY LTD.

LOCATION MAP
HUCKLEBERRY PROPERTY &
TWINKLE LAKE TOWNSITE

Figure X-1

In total, the townsite for a 15,000 TPD mining operation will be required to provide accommodation for 196 families and 69 single status personnel. The population is projected to be approximately 950. The basis for these projections is detailed in Table X-1.

Facilities

As well as providing accommodation for the population the townsite should include a number of community facilities:

(a) School to grade 10, with multiple use of recreational facilities and space.

(b) Services: water storage and mains, sewer lines and treatment plant, hard topped streets, fire break and fire protection (fire hall and truck), village offices and service garage complete with ambulance, grader and snow plough.

(c) Amenities: Hotel with lounge and dining facilities, shopping centre with bank, post office, food store, etc., medical clinic with doctor and nurse, cable T. V. and radio, light industrial area, community church.

(d) Recreational Facilities: Curling rink, skating and hockey rink, ski hill and tow, tennis courts, marina, picnic and beach area.

Organization

In order to take full advantage of all available government assistance in establishing townsite facilities a village should be incorporated under the Municipal Act as soon as construction begins. The administration of the village would rest with an appointed council for 2-4 years or until a properly constituted, elected village council was able to take over administration of the municipality's affairs.

Table X-1

ACCOMMODATION AND POPULATION PROJECTIONS

ACCOMMODATION				10,000 TPD	15,000 TPD
Total project personnel				222	254
Say 20% single status				- 44	- 50
Married				178	204
10% of project personnel live in same house				- 20	- 23
Accommodation required for married project personnel				158	181
Services Personnel:	RCMP	4			
	School to Grade 10	15			
	Medical	3			
	Store	4			
	Hotel	8			
	Recreation	2			
	Mgr. single accom.	1			
	Village Admin.	3			
		40			
	60% Single	24		16	16
Total married accommodation				174	197
Single Accommodation:					
	Project personnel	44	50		
	Services personnel	24	24	68	74
Total accommodation required				242	271
POPULATION					
Married:					
	Project personnel	158	181		
	Services personnel	16	16		
		174	197 x 4.5	783	887
Single:					
	Project personnel	44	50		
	Services personnel	24	24	68	74
				851	961

CAPITAL COST

Municipal Services

The municipality will require approximately two hundred acres of land of which seventy-five acres would have to be cleared for housing and public facilities. The cost of these activities together with lot preparation, roads, water supply, sewage disposal, provision of power and natural gas, municipal offices, and related equipment has been illustrated in Table X-2 to be $2,373,000. The costs of land, roads, sewers and water are equivalent to about $7,000 per developed lot.

Accommodation

The housing should consist of a mix of mobile homes, houses, apartments, and apartment-type single accommodation; similar to the existing facilities at Granisle. Permanent bunkhouses have not been considered as they lead to a social environment that is unacceptable to most employees. Mobile homes would be privately owned, but located in a park developed by the company or possibly a private developer. It has been assumed that all houses except those for the Manager and Superintendents (6) would be sold to the employees by way of N.H.A. mortgages. Granby would provide the serviced and landscaped lot to the employee as part of his down payment.

The accommodation mix has been estimated on the basis of Granby's experience at Granisle. Single status accommodation would consist of seventy percent one-man suites and thirty percent two-man suites. Accommodation for married personnel would consist of a thirty lot mobile home court, thirty apartments or row houses and in excess of one hundred single family dwellings. The number of units and the related cost estimates are detailed in Table X-3 and summarized at the end of this section. The capital requirement at 15,000 TPD is $4.95 million. Granby's investment would be $1.7 million; the remainder being disbursed by Granby and repaid when the employees mortgage the houses at the start of operations.

Table X-2

CAPITAL COST OF LAND AND MUNICIPAL SERVICES

LAND ACQUISITION AND PURCHASE

Purchase Crown land - 200 acres at $100	$ 20,000	
Plan and survey	80,000	
Clear 75 acres, at $300 per acre	225,000	
Lot preparation - equivalent of 200 lots at $3,000	600,000	
Sub-total		$ 925,000

SERVICES

Roads - 4 miles at $50,000	$200,000	
Sewer lines at $16 foot	240,000	
Sewerage Treatment Plant - less Federal grant	120,000	
Water lines at $10 foot	140,000	
Water tank and pumping	90,000	
Power (refer Section VII - Power)	388,000	
L. P. Gas System for heating	150,000	
Telephones - by others	-	
Fire truck, Amublance and Hall	70,000	
Village office, service garage & storage	50,000	
Sub-total		$1,448,000
TOTAL		$2,373,000

Table X-3

CAPITAL COST OF ACCOMMODATION

Type of Accommodation	Unit Cost	10,000 TPD Plant No. Units	10,000 TPD Plant Cost	15,000 TPD Plant No. Units	15,000 TPD Plant Cost
Single status - cost/employee	$ 8,000	75	$ 600,000	80	$ 640,000
Mobile home court - refer to Municipal		30	-	30	-
Apartments - 900 square feet	16,200	30	486,000	30	486,000
Houses - 1000 square feet	25,000	109	2,725,000	130	3,250,000
- 1500 square feet	40,000	6	240,000	6	240,000
Landscaping	2,000	145	290,000	166	332,000
TOTAL CAPITAL REQUIREMENT			$4,341,000		$4,948,000
Less: Mortgages by Employees			2,725,000		3,250,000
CAPITAL COST OF ACCOMMODATION			$1,616,000		$1,698,000

Public Facilities

It has been assumed that the hotel and shopping centre would be built by a private developer with a portion of the mortgage guaranteed by Granby. It has been estimated that a 20,000 square foot shopping complex would be required to house the medical clinic, post office, stores and service station. The minimum size hotel would contain 20 rooms, a cafeteria, dining room and bar facilities. The cost of these buildings is assumed to be $1,000,000 of which an $800,000 mortgage would be guaranteed by Granby.

The school would be planned and built by the provincial government in conjunction with the local school board. Definitive planning at an early stage should enable the school building to be utilized for social, religious and athletic activities. Other recreational facilities such as a skating rink, curling rink, beach and marina will be eligible for provincial grants up to one-third of the capital cost. Detailed plans have not been developed, however, it is estimated that Granby's contribution to the cost of these facilities would be in the order of $300,000.

CAPITAL COST SUMMARY

	Capital Expenditures (15,000 TPD)	Guaranteed Mortgages
Municipal Services		
Land Acquisition and Purchase	$ 925,000	-
Services	1,448,000	-
Accommodation	1,698,000	$3,250,000
Public Facilities		
Hotel and Shopping Centre	-	800,000
Recreation Facilities	300,000	
Sub-total	$4,371,000	$4,050,000
Less: Power costs included in Section VII	($ 388,000)	-
Capital Investment Required	$3,983,000	$4,050,000

At a mining rate of 10,000 TPD, the capital investment would be $3,901,000 and the guaranteed mortgages would be $3,525,000.

OPERATING COST

It is expected that the operation of the village will be financed by the Provincial Government per capita grant and by property taxes, a portion of which will be paid by Granby. A preliminary estimate of municipal operating costs and the assessment base indicate that the property tax rate would be in the order of 25 mils. School taxes and the Regional District tax have been assumed to be 40 mils, the same as Granisle.

The estimated operating costs for dwelling, maintenance, living subsidies and Granby's municipal taxes, detailed in Table X-4, amount to $241,000 per year. This estimate assumes that the staff living in the single accomodation continues to be subsidized for room and board to the extent of $2,500 per man-year.

MAIN ACCESS ROAD

The main road from Highway 16 to the Huckleberry property will need appreciable upgrading to meet project requirements and to permit year-round access to the Tahtsa Reach area.

The 74 miles of road from Highway 16 to the proposed plantsite are treated in three sections. The first 27 miles, to the Tahtsa Lake turnoff on the Francois Lake road, are moderately good quality gravel road and should not require upgrading. The following 43 miles, which are currently Forest Access Road, will require 7 miles of re-routing and 36 miles of upgrading. Road construction should be relatively easy, since much of the terrain is favourable, with ample supply of gravel and very little rock. The last 4 miles from the Sweeney Lake turnoff of the plantsite are new road, the cost of which is covered under Ancillary Services.

Capital Cost

7 miles @ $100,000 per mile	=	$ 700,000
Upgrading 36 miles @ $10,000 per mile	=	360,000
Engineering and Supervision	=	70,000
Total		$1,130,000

Table X-4

TOWNSITE OPERATING COSTS

		$/Year
Municipal taxes - property		10,000
- school		16,000
Maintenance and management of company owned dwellings		
- single accommodation	$20,000	
- apartments	$20,000	
- homes and trailers	$ 8,000	
		48,000
Subsidies - 80 single status personnel at $2,500/man-year		200,000
- home owners property tax relief		70,000
Sub-total		344,000
Less: Credit on rents		
- single accommodation	$50,000	
- apartments and houses	$43,000	
- trailer court	$10,000	
		103,000
TOTAL TOWNSITE OPERATING COST		$ 241,000

Since the improvement of this road and keeping it open during the winter months will have substantial benefits to the Pacific Salmon Fisheries Commission, the forest industry and recreation users; it has been assumed in this study that the maintenance costs will be borne by others. Discussions will have to be held with various government departments to determine what portion of the capital costs will be borne by the government.

XI

MARKETING

MARKETS

Specific market investigations have not been undertaken for this study; however, considering the anticipated concentrate grades and our general knowledge of current market conditions, the concentrates should be readily saleable. Japan, which has been the major market for British Columbia's copper concentrates for the past decade, will require increasing amounts of concentrates to feed its existing and proposed smelters by the late 1970's. Other markets, albeit of lesser importance, currently exist in eastern Canada, the United States and western Europe. Japan continues to provide the most favourable market; however, this condition could change due to the increasing cost of Japan's energy and the heavy reliance of European smelters on concentrate supplies from some of the politically uncertain developing countries.

There is a strong possibility of a copper smelter in British Columbia by 1977-78. The current level of copper production would be sufficient to support two or three conventional pyrometallurgical smelters. Barring a significant and non-competitive increase in mining taxation or royalties; British Columbia's copper production should equal or exceed the 1973 figure of 350,000 tons contained metal in future years. British Columbia's proximity to relatively cheap energy sources and its political stability may be sufficient to offset its high capital and labour costs, thereby making a B. C. smelter competitive in the world copper markets. There have been indications that both the federal and provincial governments wish to encourage the processing of copper concentrates. If these result in positive actions; such as reduced taxes or royalties for domestically processed concentrates, both smelting and new mine production would be encouraged. Alternatively, if mines are coerced, by means of restrictive legislation, into selling concentrates to a non-competitive smelter, new mine production will be inhibited and British Columbia will not be able to ensure itself of a major long term position in world copper markets.

For the purposes of this study, the economic projections have been based upon current smelter terms and ocean freight costs assuming that the concentrates are shipped to Japan. Therefore, it is implicitly assumed that, if the concentrates were shipped to a domestic smelter, it would be on competitive terms. The terms of the smelter contract are discussed in Chapter XII.

METAL PRICE FORECASTS

COPPER

The estimation of a long-term average copper price is the most critical factor in the economic analysis next to the estimation of reserve grade; yet in a period of rapid inflation and escalating smelting charges it is the most uncertain. The current market prices reflect these factors but are too volatile to be useful in determining a long-term projection. The London Metal Exchange price is inordinately high due to a shortage of supply and to the effect of commodities speculators while the U.S. Producer Price is held at an artificially low level by price controls in the United States.

Since the primary reason for a long-term average price forecast is to predict the economic viability of a project during its productive life, it is essential that the price forecast be on the same basis as the cost forecasts. In this instance the base time is February, 1974.

A copper price forecast was made, based on the Foreign Refinery price as reported in the Engineering and Mining Journal; adjusted by the Bureau of Labour Statistics Consumer Price Index (1967=100) for the years 1935 to 1973. These data were plotted and a least-squares trend line calculated. The price data and trend line are shown in Figure XI-1. This calculation shows that the 1974 trend line price, adjusted for inflation to a 1967 base, is $0.49/lb. copper. The increased cost of smelting and refining due to environmental controls, which commenced in 1971, have not yet affected the trend line. Typical charges for ocean freight, smelting and refining in 1970 were 8.4¢/lb. compared to 17.5¢/lb. today. This difference must be accounted for in the determination of a price forecast as the costs of the higher smelter charges were included in the economic projections.

COPPER PRICE IN CONSTANT DOLLARS

(E. & M.J. FOREIGN REFINERY)

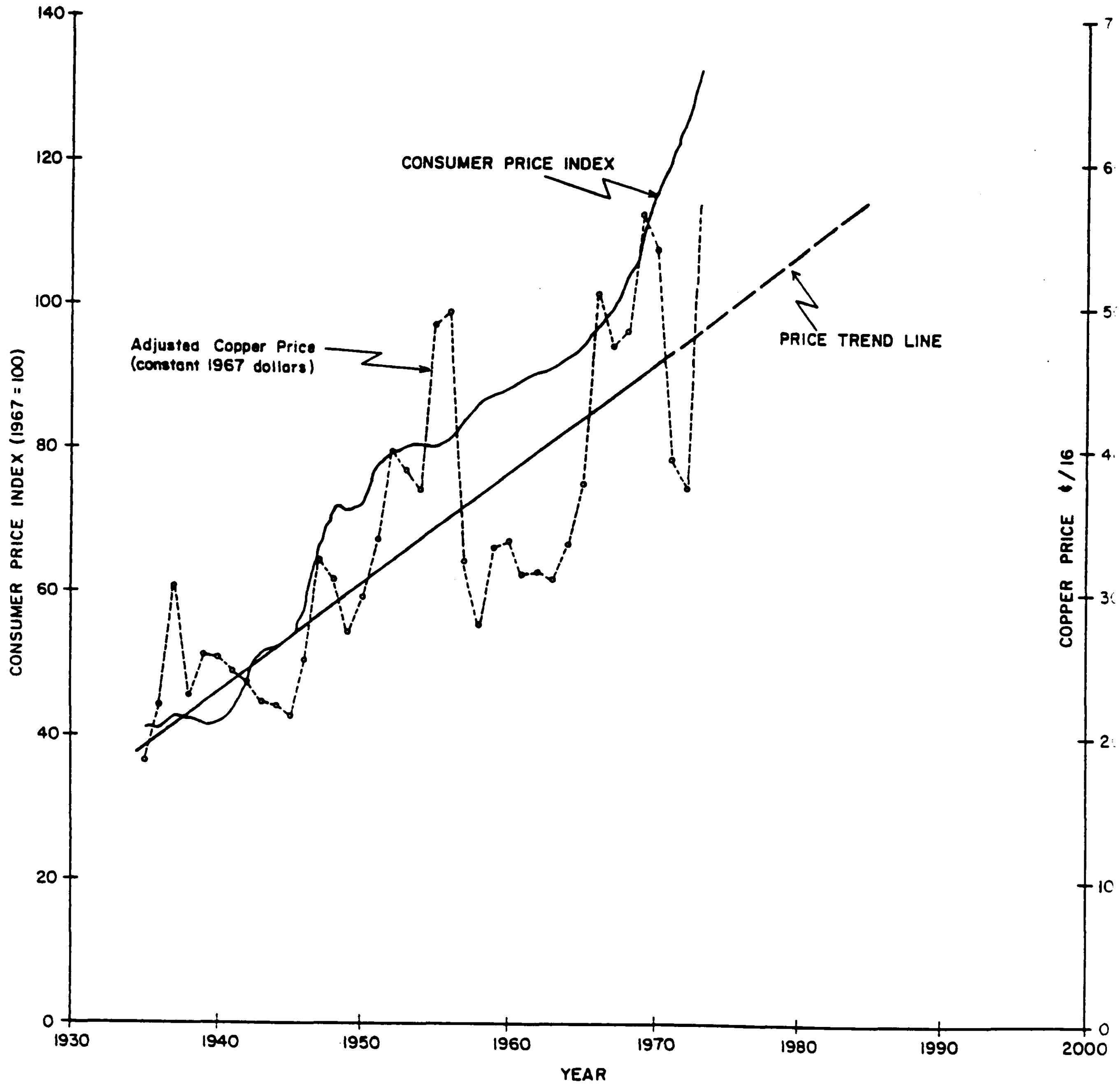


Figure XI-1

In periods of rapid inflation such as the past year, the Consumer Price Index tends to lag behind the Wholesale Industrial Price and Commodity Price Indices. Since the Consumer Index contains a heavy weighting from non-durable goods and the Commodity Index is weighted by copper itself; the Wholesale Index was used as the criterion index for predicting the price of copper. That is to say, the Wholesale Industrial Price Index is assumed to be the best adjustment factor for metal prices considering the cost base for the mining industry. The Bureau of Labor Statistics Wholesale Industrial Price Index was 142.5 for February, 1974, and was approximately 110 for 1970. Therefore, smelting, refining and ocean freight charges have increased by (17.5/1.425) - (8.5/1.10) = 12.28 - 7.73 = 4.55¢/lb. in terms of 1967 dollars. The forecast long-term copper price would be 49¢ + 4.55¢/lb. in 1967 dollars or 76.3¢/lb. in February, 1974 dollars.

Because of the sensitivity of the profitability of the project to the price of copper, the decision to continue development of the property could hinge upon the accuracy of the price forecast. Detailed studies by others have been examined and their price forecasts escalated for both increased smelting charges and inflation. Comparative figures are as follows:

		$/lb.
U.S. B.M.	1968 computer model	0.74
David Lowell	1970 graphical projection	0.81
Producers opinions	1971 survey	0.74
Granby forecast	1974 graphs	0.76

Commodities Research Unit, in their publication "Copper Studies" dated March 5, 1974, analysed the effect of inflation upon the primary copper producers, secondary producers and consumers to determine a combined index for inflation. Their study also considered the recent changes in currency values, a factor which the other studies omitted. Although the nineteen year time span is somewhat short for a trend line based upon widely varying data; the 1974 trend line projection would be about $0.73/lb.

In our opinion, the preceding projections are somewhat conservative because they either do not consider the full impact of currency fluctuations or are based upon inadequate accounting of the impact of higher smelting charges.

Consequently, it is recommended that the project be evaluated on the basis of a long-term average copper price of $0.80 per pound based on February, 1974 dollars. It is also recommended that prices of $0.75 and $0.85 be investigated as being the range within which the long-term average is likely to occur.

Effect of Price Cycles

The historical pattern of London Metal Exchange copper prices has been cyclical with shallow troughs and sharp peaks reoccuring on a six year cycle. Periods of peak copper prices have lasted less than two years. Attempts to control the cycle have generally not been effective and there is no reason to believe that the current rumors of a copper producer's cartel will affect the cyclical pattern in the longer term.

The mining industry and the financial institutions have traditionally relied upon the period of peak prices to repay substantial amounts of debt and to virtually ensure that the debt is repaid within the first five to seven years of production. If the "super" royalty provision of the minerals royalty legislation that is currently before the British Columbia legislature is enacted, the benefit of the peak in the price cycle will be substantially reduced. Consequently, the long-term average return per pound, which will be reduced by approximately three cents per pound by the five per cent royalty on net mine value, will be further reduced by the effect of the "super" royalty. The impact of the five per cent royalty has been incorporated in this study, however, the "super" royalty has not.

PRECIOUS METALS

Because gold and silver occur in relatively minor amounts in the deposit, long-term price trends for these metals have not been investigated. A gold price of $150 per Troy ounce has been used in the revenue projections since it is probable that the price of gold will stay at or above this level. A price of $5.00 per Troy ounce has been used for predicting revenues from the silver contained in the copper concentrates.

XII

ECONOMIC PROJECTIONS

COST SUMMARY

Capital Costs

Capital costs including a 10% contingency excluding working capital and inventory as shown in Table XII-1 are:

$59,500,000 for 10,000 TPD operation

$63,500,000 for 15,000 TPD operation

Working capital requirements are estimated at four months operating costs. These will amount to:

$3,221,000 for 10,000 TPD operation

$4,170,000 for 15,000 TPD operation

Inventory estimates are based on experience obtained from existing operations and to the equipment selection proposed in this study. The respective costs of inventory are:

$1,200,000 for 10,000 TPD operation

$1,350,000 for 15,000 TPD operation

Operating Costs

Operating costs are summarized in Tables XII-2 and XII-3 and amount to:

$9,663,000 per year)
$2,663 per ton milled) for 10,000 TPD operation

$12,506,000 per year)
$2.301 per ton milled) for 15,000 TPD operation

Construction Schedule

The proposed construction schedule, shown in Figure XII-1 in the Appendix, is based on the assumption that manpower and materials shortages do not adversely affect construction.

For 10,000 TPD, construction would be completed in 24 months including 6 months engineering.

For 15,000 TPD, construction would be completed in 27 months including 6 months engineering.

In both cases, the peak manpower would be 220 men during the 20th and 21st months.

Table XII-1

CAPITAL INVESTMENT SUMMARY

(x $1000)

	10,000 TPD Plant	15,000 TPD Plant
Mining - Table VI-4	$ 6,576	$ 7,281
Concentrator and Surface Plant - Tables VII-2 and VII-3	29,424	32,282
Preproduction Expense - Table VI-10	6,124	6,124
Ancillary Services - Table VII-10	1,246	1,246
Concentrate Handling - Chapter VII	420	420
Primary Power - Chapter VII	5,261	5,261
Townsite - Chapter X	3,901	3,983
Main Road - Chapter X	1,130	1,130
SUB-TOTAL	$54,082	$57,727
Contingency - 10%	5,418	5,773
TOTAL FIXED ASSETS	$59,500	$63,500
Working Capital	3,221	4,170
Inventory	1,200	1,350
TOTAL CAPITAL INVESTMENT	$63,921	$69,020

Table XII-2

SUMMARY OF ANNUAL OPERATING COSTS - 10,000 TPD

x 1000 DOLLARS

	Operating Labour		Operating Supplies		Repair & Maint. Labour		Repair & Maint. Supplies		TOTAL	
	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton
Mining										
Drilling	89	.011	171	.021	40	.005	21	.003	321	.040
Blasting	41	.005	481	.059	-	-	-	-	522	.064
Loading	94	.012	177	.022	69	.008	56	.007	396	.049
Hauling	187	.023	371	.046	132	.016	109	.013	800	.099
Support Equipment	109	.013	157	.019	64	.008	44	.005	374	.046
Gen. Serv. & Supervision	109	.013	35	.004	-	-	-	-	144	.018
Repair Shop	73	.009	41	.005	-	-	-	-	113	.014
Sub Total	702	.086	1,433	.176	305	.038	230	.028	2,670	.330
Sub Total Per Ton Milled		.198		.398		.085		.063		.759
Crushing & Concentrating										
Crushing	164	.045	62	.017	120	.032	85	.023	431	.118
Grinding	365	.100	1,652	.453	79	.022			2,096	.574
Flotation & Dewatering			437	.120	79	.022	178	.049	694	.190
Gen. Serv. & Supervision	217	.060	565	.155	79	.022			861	.236
Sub Total	746	.205	2,716	.745	357	.098	263	.072	4,082	1.118
Plantsite	166	.045	200	.055	103	.028	48	.013	517	.141
Townsite	42	.012	177	.035			16		235	.054
Engineering	154	.042	17	.005					171	.047
Mine Administration	796*	.218	906	.248					1,702	.466
Company Administration	96	.026	190	.052					286	.078
TOTALS	2,702	.748	5,639	1.538	765	.211	557	.148	9,663	2.67

Table XII-3

SUMMARY OF ANNUAL OPERATING COSTS - 15000 TPD

x 1000 DOLLARS

	Operating Labour		Operating Supplies		Repair & Maint. Labour		Repair & Maint. Supplies		TOTAL	
	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton	Cost	Per Ton
Mining										
Drilling	111	.009	248	.020	40	.003	37	.003	436	.035
Blasting	41	.003	720	.058	-	-	-	-	761	.061
Loading	141	.011	186	.015	97	.008	75	.006	499	.040
Hauling	280	.023	596	.048	178	.014	174	.014	1,228	.099
Support Equipment	182	.015	161	.013	78	.006	62	.005	483	.039
General Serv. & Supervision	143	.012	37	.003	-	-	-	-	180	.015
Repair Shop	73	.006	62	.005	-	-	-	-	135	.011
Sub Total	971	.079	2,010	.162	393	.031	348	.028	3,722	.300
Sub Total Per Ton Milled		.182		.373		.071		.064		.690
Crushing & Milling										
Crushing	164	.030	93	.017	125	.023	128	.023	510	.093
Grinding	365	.066	2,479	.453	83	.015			2,927	.535
Flotation & Dewatering			657	.120	83	.015	267	.049	1,007	.184
General Serv. & Super	217	.040	850	.155	83	.015			1,150	.210
Sub Total	746	.136	4,079	.745	374	.068	395	.072	5,594	1.022
Plantsite	179	.033	241	.044	113	.021	58	.011	591	.114
Townsite	42	.008	177	.023			16		235	.043
Engineering	154	.028	17	.003					171	.031
Mine Administration	866*	.158	1,023	.187					1,889	.345
Company Administration	104	.019	200	.037					304	.056
TOTALS	3,062	.564	7,747	1.039	880	.160	817	.149	12,506	2.301

*[illegible]

REVENUE PROJECTIONS

SMELTER SCHEDULE

The following smelter schedule is based upon typical contracts as they apply to March, 1974, copper concentrates sold to Japanese smelters. In keeping with the balance of the report, the schedule refers solely to March, 1974, and is not indicative of the terms that are currently being negotiated or that would apply at the start of production.

Copper Payment: Pay for 100% less 1.25 units at the London Metal Exchange cash asked wirebar price.

Gold Payment: If gold content is greater than 1 gram/metric ton, pay for 90% of the contained gold at the London Gold Price less $5.00, less 3%. Otherwise, no payment.

Silver Payment: If silver content is greater than 1 Troy ounce, pay for 90% of the contained silver, less 1 ounce, at the London Silver price.

Treatment Charges: Assume $35.00/dry metric ton.

Ocean Freight: Assume $18.00/long wet ton.

Refining Charge: 5.5¢/lb. copper paid for plus 1/10¢ for each 1.0¢ increase in the copper price above 55¢/lb.

REVENUE CALCULATIONS

Basic Calculation

Assume a concentrate grade of 27% Cu, 0.059 oz. Au and 1.94 ozs. Ag per short dry ton and that the concentrates will be shipped at 7% moisture.

Copper content of concentrates = 540 lb.
Copper content paid for = 540 - 25 = 515 lb.
Gold content of concentrates = 1.83 grams
Silver content of concentrates = 1.94 ounces.

The total precious metal payment per ton of concentrate has been calculated on the basis of gold prices of $150.00/ounce and silver prices of $5.00/ounce.

Gold Payment = 0.059 ounces x 0.90 x (150 - 5 - 4.50)
= 0.059 x $126.45
= $7.46/ton concentrate

Silver Payment = (1.94 - 1.00) x 0.90 x $5.00
= 0.94 x $4.50
= $4.23/ton concentrate

Total precious metal payment = $11.69/ton concentrate

Treatment charges = $35.00/dmt or $35.00 x 0.90718 = $31.75/sdt
Ocean Freight = $18.00/lwt or $18.00 x 0.8929 = $16.07/swt or $17.28/sdt at 7% moisture
Total treatment plus Ocean Freight charges = $49.03/ton

The costs of hauling the concentrates from the mine site to Houston, B. C., rail shipment to Vancouver, storage and loading on board ship have been estimated in Chapter VII to be $26.00 per short dry ton.

The calculation of the net smelter return per ton concentrate and the net return to operations per pound copper produced, for a range of copper prices, is shown in Table XII-4. The net return to operations at copper prices of $0.80 and $1.00 are $0.57 and $0.74 respectively. Losses incurred in handling and shipping, which have been estimated at 1%, have been deducted in the cash flow projections.

Table XII-4

REVENUE PER TON CONCENTRATE

Copper Price/Lb.	$ 0.60	$ 0.80	$ 1.00	$ 1.20
Copper payment	309.00	412.00	515.00	618.00
Precious metal payment	11.69	11.69	11.69	11.69
Total Payments	$320.69	$423.69	$526.69	$629.69
Less: Refining Chg.	30.90	41.20	51.50	61.80
Treatment & Shipping	49.03	49.03	49.03	49.03
Total Deductions	$ 79.93	$ 90.23	$100.53	$110.83
Net Smelter Return	$240.76	$333.46	$426.16	$518.86
Less: Haulage & Loading	26.00	26.00	26.00	26.00
Net Return to Operations per ton concentrate	$214.76	$307.46	$400.16	$492.86
Net Return/lb. Cu produced	39.8¢	56.9¢	74.1¢	91.3¢

ECONOMIC PROJECTIONS

TAXATION

Capital Cost Allowance

All initial capital expenditures including infrastructure and townsite costs have been assumed to be Class 28 assets. Capital cost allowance for this asset class may be claimed to the extent of operating income. Replacement mobile and plant equipment is included in Asset Class 10. Capital cost allowance for Class 10 assets may be claimed at an annual declining balance rate of 30% of the net class asset value or to the extent of operating income, whichever amount is smaller. The net class asset value for a particular year is defined as the asset balance of the previous year plus the difference between asset additions and disposals, less capital cost allowance claimed in the previous year.

British Columbia Mining Tax

The British Columbia Mining Tax was calculated at a rate of 15% of base income, but not to exceed 65% or be less than 15% of net profit from mining operations less Federal capital cost allowance. Base income is income from mining operations net of Federal capital cost allowance, less $10,000, less an abatement for processing assets. The processing assets allowance is 8% of the net capital cost (last year's balance plus acquisitions less disposals) of processing assets, on a declining balance basis. Until 1977, the B. C. Mining Tax is deductible from income for Federal tax purposes. After 1976, this deduction will not be allowed. There is a strong possibility that the rate of taxation and/or the tax base will be changed at that time; however, in this study, 1974 rates and bases are used.

In summary:

B. C. Mining Tax = 0.15 x (Net Mine Income - Federal Capital Cost Allowance - $10,000 - Processing Assets Allowance)

Processing Assets Allowance = 0.08 x (Processing Assets + Acquisitions - Disposals - Last Year's Processing Assets Allowance)

B. C. Mining Tax is not to exceed 0.65 x (Net Mine Income - Federal Capital Cost Allowance) or to be less than 0.15 x (Net Mine Income - Federal Capital Cost Allowance).

Federal Income Tax

Federal Income Tax (which includes Provincial Corporate Income Tax until 1977) has been determined as follows:

1975 and 1976: 46% x (Net Mine Income - Federal Capital Cost Allowance - B. C. Mining Tax)

Beyond 1976: 31% x (Net Mine Income - Federal Capital Cost Allowance)

Other Taxes

Allowance for property and other taxes is made in indirect operating costs. Sales taxes, where applicable, have been included in supply costs. Depletion allowance is permitted up to one-third of eligible expenditures; however, it is assumed that no eligible expenditures (primarily exploration) would apply.

ROYALTIES

Because the Huckleberry deposit is situated on located mineral claims, minerals extracted from it will be subject to the provisions of the British Columbia Mineral Royalties Act (Bill 31). This legislation has not been enacted to date; therefore, certain draft provisions are unclear. Nevertheless, royalty calculations have been based on the following interpretation. The basic rate of royalty is 5% of "the net value of a designated mineral produced," where "net value" is net smelter return less freight and handling on concentrates. In addition to the basic royalty, there is a super royalty of "50% of the amount by which the gross value.....exceeds the basic value.....by 20% or more." Expressions in parentheses are taken from the legislation. The definition of "basic value" is not certain but it is assumed to be the arithmetic mean metal price for the 5 years preceding the taxation year, with the price adjusted to account for inflation in costs.

For the purpose of determining Huckleberry cash flows, it is assumed that metal prices will remain relatively stable at a level less than 20% higher than any given 5-year average; therefore, the super royalties do not apply and the royalty will be 5% of the "net value."

PROGRAM DESCRIPTION

Yearly cash flows were calculated by means of a computer program. The following annual input parameters were employed:

1. Calendar year.
2. Capital expenditures.
3. Tons and grade of ore mined.
4. Tons of waste and overburden mined.
5. Tons and grade of stockpile milled.
6. Concentrate grades of gold, silver, and copper.
7. Metal prices for gold, silver, and copper.
8. Copper recovery.
9. Mining, milling, and indirect costs per ton of mill feed.
10. Smelting, freight, and handling charges per ton of ore.
11. Expenditures on processing assets.
12. Allowable expenses for depletion.
13. New loans and prevailing interest rates.

These variables are subscripted according to year number and stored in matrix form. The program then calculates concentrate production, costs, revenues, income and mining taxes, royalties, and cash flows on an annual basis. Negative cash flows in preproduction years are carried forward as loans at the rate of interest which is input. All taxes and costs are carried on a current rather than a payable basis. Transit losses of concentrate are considered by reducing concentrator recoveries by one percentage point from expected values.

After the cash flow calculation has been completed, an iterative routine is employed to determine the internal rate of return of the cash flow pattern. Because the program cannot easily be modified, when debt is employed, it is necessary to calculate the internal rate of return manually for these cases.

CASH FLOW PROJECTIONS

In order to test factor sensitivities, a series of 15 cash flow analyses was run. The basic input data are summarized in Table XII-5 and the results in Table XII-6.

Table XII-5

CASH FLOW - BASIC INPUT DATA

VARIABLE	10,000 TPD	15,000 TPD
Mining and Milling schedule	Table VI-2	Table VI-3
Capital Investment schedule	Table XII-1	Table XII-1
Replacement Capital - Mobile Equipment	Table VI-6	Tabel VI-6
- Plant	$125,000/yr.	$150,000/yr.
Copper price	Variable	Variable
Gold price	$150/oz.	$150/oz.
Silver price	$5.00/oz.	$5.00/oz.
Income tax rate (After 1976)	0.31	0.31
Depletion rate	N/A	N/A
Cost of mining overburden/ton	$1.00	$1.00
Initial ore mining cost/ton	$0.33	$0.30
Initial waste mining cost/ton	$0.32	$0.31
Stockpile handling cost/ton	$0.15	$0.15
Crushing and concentrating cost/ton	$1.12	$1.02
Indirect costs/ton	$0.79	$0.59
Smelting costs/ton concentrate	$49	$49
Freight and Handling/ton concentrate	$26	$26
Processing assets	$32.3 million	$29.5 million
Interest rate	--- 9% in debt cases ---	
B. C. Royalty rate	5%	5%
Concentrate grade - gold	0.059 oz./sdt.	0.059 oz./sdt.
Concentrate grade - silver	1.94 oz./sdt.	1.95 oz./sdt.
Concentrate grade - copper	27.0%	27.0%
Recovery	--- Fig. V-2 minus 1% ---	

Table XII-6

SUMMARY OF ECONOMIC PROJECTIONS

CASE NO.	PLANT CAPACITY SDT/DAY	COPPER PRICE $/LB.	OPERATING COST $/SDT	EQUITY INVESTMENT $ x 10^6	DEBT INVESTMENT $ x 10^6	GROSS CASH FLOW $ x 10^6	NET CASH CASH $ x 10^6	INCOME TAX $ x 10^6	B.C. MINING TAX $ x 10^6	B.C. ROYALTY $ x 10^6	TOTAL TAXES & ROYALTIES $ x 10^6	INTERNAL RATE OF RETURN *	PRESENT VALUE (10%) $ x 10^6
						10,000 TPD BASE CASE							
	10,000	1.00	2.70	67.218	0	153.407	73.336	46.758	19.201	27.808	93.767	11.1	+ 4.641
						15,000 TPD BASE CASE							
1	15,000	0.85	2.30	69.020	0	128.142	49.529	33.149	13.614	23.782	70.545	11.0	+ 3.671
						15,000 TPD - METAL PRICE SENSITIVITY							
2	15,000	1.00	2.30	69.020	0	172.002	93.389	57.317	23.549	27.885	108.751	18.1	+30.253
3	15,000	0.90	2.30	69.020	0	142.760	64.147	41.205	16.928	25.149	83.282	13.5	+12.621
4	15,000	0.80	2.30	69.020	0	113.523	34.910	25.093	10.302	22.414	57.809	8.5	- 5.350
						15,000 TPD - CAPITAL COST SENSITIVITY							
5	15,000	0.85	2.30	64.020	0	125.955	52.342	34.699	14.252	23.782	72.733	12.3	+ 7.527
6	15,000	0.85	2.30	74.020	0	130.410	46.797	31.599	12.896	23.782	68.227	10.0	- 0.126
						15,000 TPD - OPERATING COST SENSITIVITY							
7	15,000	0.85	2.20	69.020	0	132.952	54.399	35.800	14.704	23.782	74.286	11.8	+ 6.409
8	15,000	0.85	2.40	69.020	0	123.378	44.765	30.498	12.479	23.782	66.759	10.3	+ 0.967
						15,000 TPD - DEBT CASE							
9	15,000	0.85	2.30	15.000	59.774	119.265	37.098	28.257	11.604	23.782	63.463	12.5	+ 5.030

* The method employed in calculating the Internal Rate of Return rounds all values up to the next even 0.1%.

10,000 Tons/Day

A cash flow projection was run at a copper price of $1.00/lb. and the capital and operating costs in Tables XII-1 and XII-2. The calculation indicated that the internal rate of return on the capital investment of $63.9 million would be 11.1%. Other variables were not investigated.

15,000 Tons/Day

This series of cash flow projections was run to investigate the effect of copper price on the base case of:

Capital Investment in Fixed Assets:	$63.5 million
Working Capital and Inventory:	$ 5.5 million
Operating Costs:	$ 2.30/ton milled

Cases 1-4: The profitability, expressed as the internal rate of return (IRR) was as follows:

Copper Price $/lb.	IRR%
0.80	8.5
0.85	11.0
0.90	13.5
1.00	18.1

These results are shown graphically in Figure XII-2.

Cases 5-6: Decreasing the capital cost by $5.0 million increased the IRR from 11.0% to 12.3%; whereas, increasing the capital cost by $5.0 million decreased the IRR to 10.0%.

Cases 7-8: Reducing the operating costs to $2.20/ton increased the IRR from 11.0% to 11.7%; whereas, increasing the operating costs to $2.40/ton decreased the IRR to 10.2%.

Case 9: The effect of leverage was investigated for the base case. The project was assumed to be financed by $15 million equity and $54 million debt at 9% interest, repayable from the first cash flow. The debt could be repaid within the first five years of production. The IRR on the equity investment was 12.5%.

SENSITIVITY ANALYSIS

The sensitivity of the profitability of the project to changes in revenue and costs has been determined from the cash flow projections. The term "internal rate of return" (IRR) is the discount rate of which the present value of the cash inflows equals the present value of the capital investment.

The base case for the investigation was Case 10: 15,000 TPD, copper price \$0.85/lb., capital costs of \$64.85 million plus \$4.17 million working capital and operating costs of \$2.30 per ton milled.

Copper Price

Case 3, \$0.90/lb Cu, IRR = 13.5%
Case 1, \$0.85/lb Cu, IRR = 11.0%
Case 4, \$0.80/lb Cu, IRR - 8.5%

A price change of \$0.10/lb (12%) changes the IRR by 5.0 units. Therefore, a ten per cent change from the base price of \$0.85/lb changes the IRR by 4.2 units or 40%.

Capital Cost

The effect of changing the basic capital cost, including the ten per cent contingency but excluding inventory and working capital is considered in the following cases:

Case 1, Capital = \$63.5 million, IRR = 11.0%
Case 7, Capital = \$58.5 million, IRR = 12.3%
Case 8, Capital = \$68.5 million, IRR = 10.0%

Changing the capital cost by \$5.0 million (7%) changes IRR rate of return by 1.2 units. Therefore, a ten per cent change (\$6.4 million) changes the IRR by 1.7 units or 16%.

Granby staff have subjectively estimated the accuracy of the capital cost estimates exclusive of the uncertainty introduced by inflation. The estimated ranges for each major cost centre are shown in Table XII-7. On this basis, the 90% confidence interval for the project capital cost is \$67.4 million to

Table XII-7

RANGE OF CAPITAL COST ESTIMATES
($ millions)

COST CENTRE	LOW	MID-POINT	HIGH
Mining	$ 7.0	$ 7.3	$ 8.0
Mill	31.5	32.3	35.0
Pre-production	6.0	6.1	7.0
Surface Equipment	1.0	1.2	1.4
Concentrate Handling	0.4	0.4	0.4
Power	4.0	5.3	6.0
Townsite	3.7	4.0	4.3
Main Road	1.1	1.1	1.1
Sub-total	$54.7	$57.7	$63.2
Contingency	5.8	5.8	5.8
Working Capital & Inventory	5.5	5.5	5.5
TOTAL	$66.0	$69.0	$74.5
PREDICTED TOTAL(1)	$67.4	$69.0	$72.1

NOTE: (1) Assuming each component is normally distributed on either side of the mid-point and calculating the standard deviation of the total from the components; these figures represent the range at the 90% confidence level.

$72.1 million with a mid-point of $69.0 million. This range represents a corresponding range of plus 0.4 to minus 0.8 units in the internal rate of return.

Operating Cost

Case 4, Operating cost = $2.40/ton, IRR = 10.0%
Case 10, Operating cost = $2.30/ton, IRR = 11.0%
Case 11, Operating cost = $2.20/ton, IRR = 11.7%

Changing the operating costs by $0.10/ton milled (4.4%) changes the IRR by 0.8 units. Therefore, a ten per cent change (23 cents) would change the rate of return by 1.8 units or 16%.

An estimate of the expected range of operating costs was not attempted because of the critical and complex relationship between grindability and grinding circuit operating costs. The variability of factors other than grinding costs is not likely to exceed 7% of the total operating costs.

The effect of the parameters mentioned above, as well as changes in the reserve grade on the project rate of return, are shown in Figure XII-3. The extreme sensitivity of profitability to changes in reserve grade and metal price should be noted as these are also the areas of most uncertainty in the project.

DISCUSSION OF RESULTS

On the basis of the current estimates of reserve grade, cost parameters and grindability parameters, a 15,000 ton per day operation could be expected to return approximately nine per cent on investment at the projected long-term average copper price of $0.80 per pound. This level of profitability would be inadequate to either obtain production financing or provide a reasonable return to the investor.

The copper content of the deposit, particularly the higher grade zone is the most critical factor affecting profitability. Additional drilling will probably be required to determine both the limits of the higher grade zone and the reliability of the reserve estimates. This problem is currently

COPPER PRICE vs. IRR

(15,000 tpd. CONCENTRATOR)

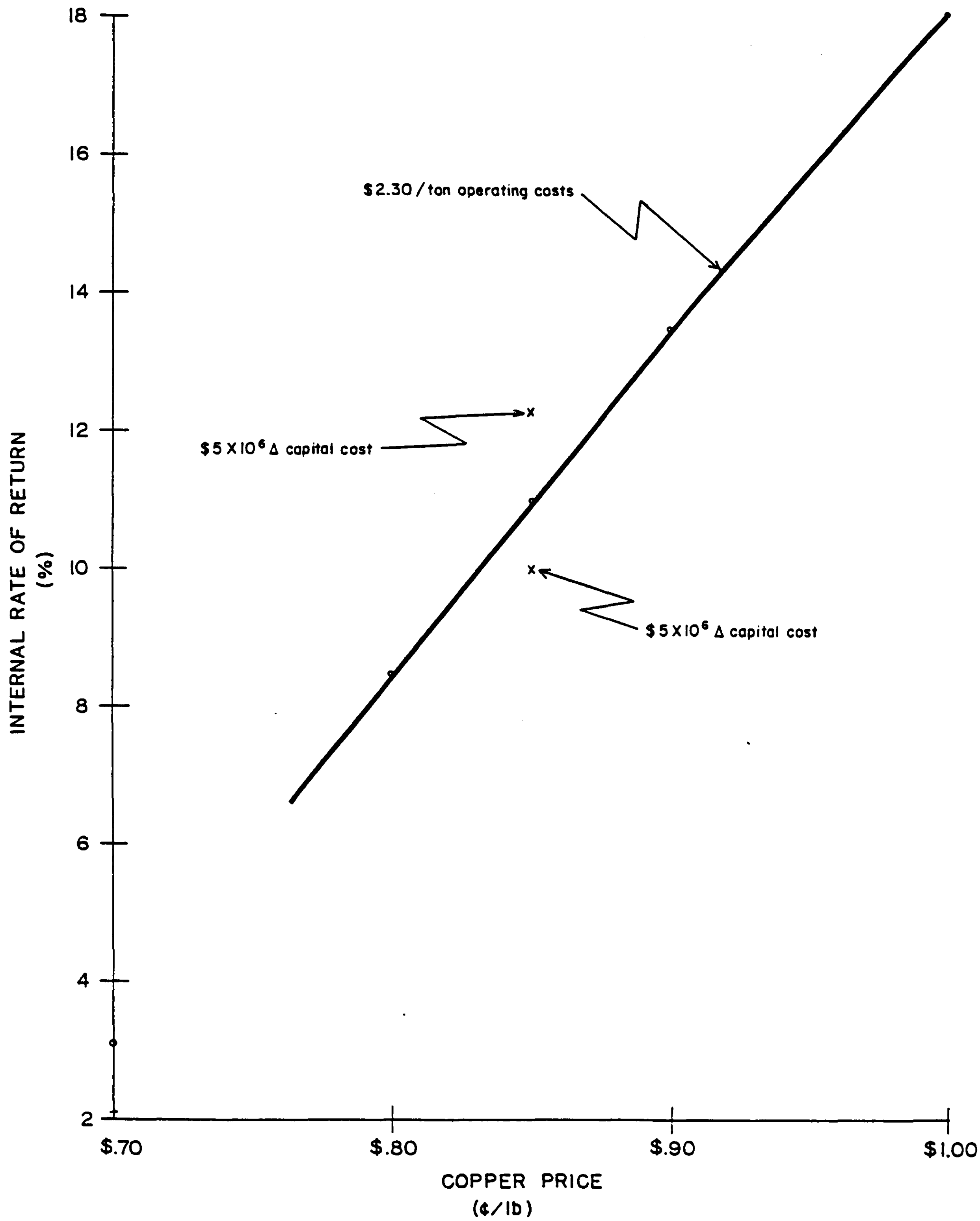


Figure XII-2

FACTOR SENSITIVITIES WITH RESPECT TO INTERNAL RATE OF RETURN

BASE CASE: 15,000 tpd., 85¢ Cu, $69 million Capital cost,

$2.30/ton Operating cost, 11.0% IRR

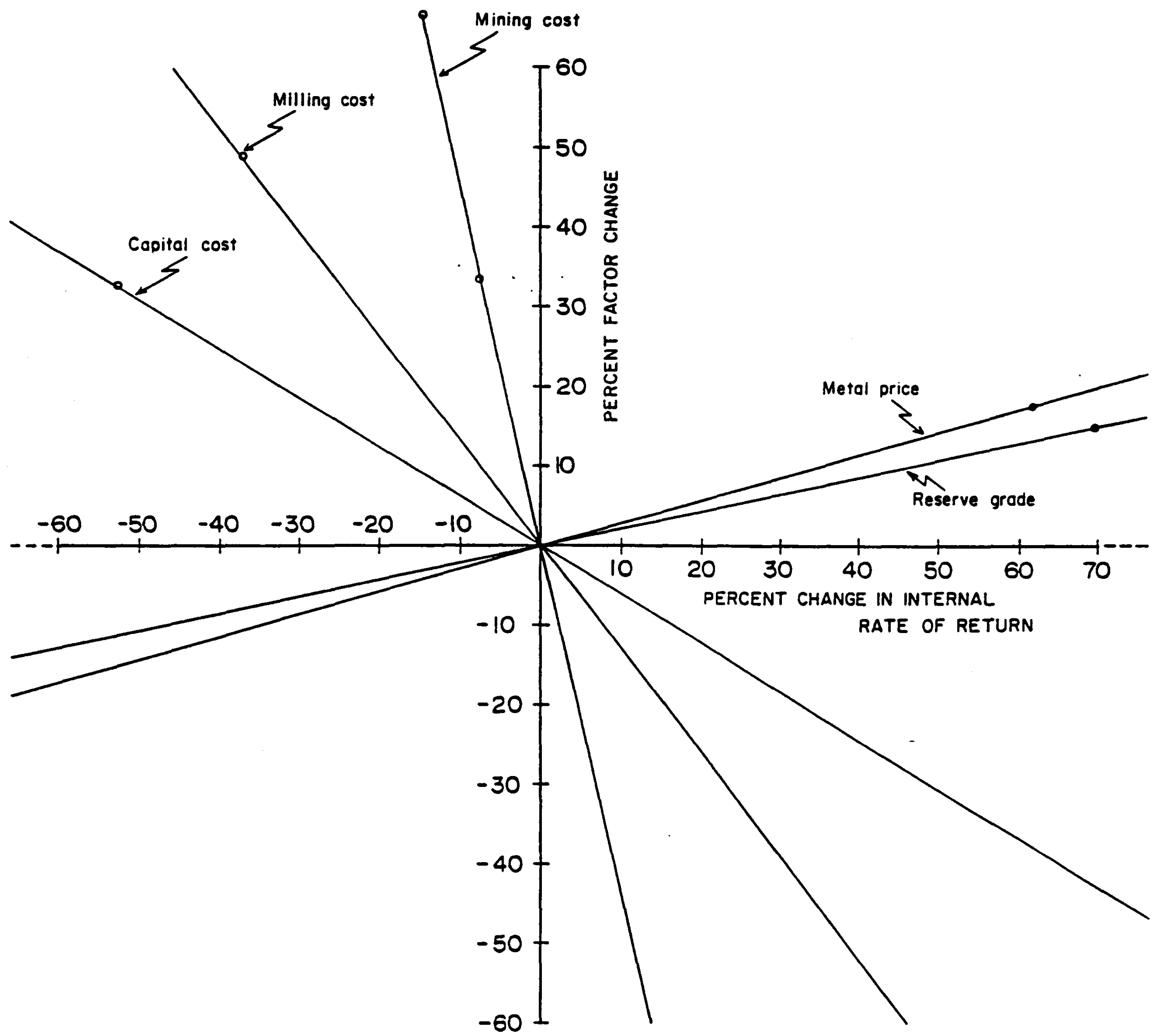


Figure XII-3

under investigation.

The other critical parameter is the price of copper. A long term average price of $0.85/lb would increase the internal rate of return to 11.0%. This in itself could be sufficient to make the project viable. The proposed royalty legislation has as direct an impact on the profitability as metal price since the royalty is a direct deduction from revenue. The five per cent royalty on net mine value is approximately equivalent to a three cent per pound reduction in metal price. Thus, eliminating the royalty would increase the rate of return by 1.5 units.

The potential benefits of molybdenite recovery have not been investigated in this study. However, if the deposit contains recoverable molybdenite as indicated in the metallurgical investigations, operating profits might be increased by $0.10 per ton by operating a molybdenite recovery circuit. This would not be adequate to make a sub-economic venture profitable, but it should be investigated in conjunction with other studies. The proposed program would entail assaying all the drill core for molybdenite, interpreting the probable molybdenite distribution and, if there appeared to be adequate amounts, commencing a metallurgical testing program.

It has been assumed that the deposit consists of uniformly hard material with a Ball Mill Work Index of 14.4 KWH/ton; however, the deposit actually contains material of varying grindabilities ranging from 10 to 19. The possibility that the western fringe, including the zone that would be mined in the intial years, is easier to grind than the average requires further investigation. If this material is more amenable than the average, either the plant capacity would be increased or capital and operating costs would be lower. Conversely, the harder material could only be treated at lower tonnage rates than the 15,000 tons/day rate forecast for the entire project life. It might be advantageous to construct a 15,000 tons/day plant initially with provision to expand it to 20,000 tons/day when the lower grade, harder ore is encountered.

Contingent upon the results of additional grindability tests, investigation of the possibility of utilizing autogenous grinding mills in place of the fine crushing and rod mill circuits may be required. The capital cost saving is estimated to be in the order of two to four million dollars. A portion of the deposit may be amenable to autogenous primary grinding, thereby eliminating grinding steel costs of $0.20/ton milled. Grinding of other portions of the deposit could require the addition of steel balls to the autogenous mills at comparable costs to the conventional circuits. A preliminary assessment of this alternative is proposed concurrently with further grindability studies. If autogenous grinding appears to be worthy of further investigation after the grindability studies are complete, a bulk sample could be obtained late in 1974 and an autogenous testing program could be undertaken during the winter of 1974-75.

The design criteria provided for Kilborn Engineering (B.C.) Ltd., stipulated that they were to design for maximum operating efficiency and low operating costs rather than low capital costs. While we believe that this philosophy must prevail in a 10,000 or 15,000 ton/day plant, there are some areas where savings can be made without sacrificing efficiency. Kilborn and H. A. Simons (International) Ltd., have reviewed the plant capital cost estimates from this perspective and agree that savings in the order of $1.5 to $2.5 million could be realized. They also agree that the unit prices for structural steel are lower than they should be for a February 1974 base, thereby eliminating a majority of the savings. The net benefit is likely to be less than $1.0 million. There is also a possibility of reductions in the cost of the concentrator building by placing the flotation machines on the floor slab and re-evaluating pumping requirements. An estimate of the potential savings should be made in the next series of engineering studies.

The above-mentioned possibilities can be summarized as follows, with the potential benefits converted to a copper price equivalent:

Item	Study Cost	Study Time	Benefit (Cu price equiv)
Copper price	(not applicable)		(unknown)
Copper grade or tonnage in high grade zone	$200,000	3 months	6¢
Royalty change by 1%	(not applicable)		0.6¢
Molybdenite assaying	$ 7,000	2 months	Nil
Molybdenite recovery	$ 25,000	3 months	1.0¢
Grinding testing	$ 20,000	3 months	Unknown

Item	Study Cost	Study Time	Benefit (Cu price equiv)
Autogenous grinding	$200,000	12 months	Capital - 1¢ Operating - 2¢
Building redesign	$ 10,000	2 months	0.5¢

If a copper price of $0.80/lb is considered to be a realistic estimate of long-term copper prices, then benefits equivalent to in excess of $0.05/lb. copper produced must be realized in other areas if the project is to be economically viable. In view of the study costs, study times and potential benefits it is recommended that all of the above areas, except autogenous grinding testing should be investigated during the summer of 1974. In addition it is recommended that surveys of the environmental conditions, tailings dam, plant site, townsite and powerline should be made in order to provide the required engineering data for final design. This work could be completed by October, 1974 at a cost of $350,000 to $400,000 dependent upon the extent of the drilling program. Concurrently with these investigations, discussions should be held with various departments of the provincial government to ensure that our investigations meet their criteria and to discuss the extent of their contributions to the townsite and infrastructure. The economic studies should be continuously updated during this period so that a complete review of the project can be made at any time.

TECHNICAL REPORT 71-10
AMENABILITY TESTING OF SAMPLES FROM KENNCO'S HUCKLEBERRY PROJECT

April 22, 1971

BY

A. W. LAST
J. L. FRASER 034-0110

KENNECOTT RESEARCH CENTER -- METAL MINING DIVISION
KENNECOTT COPPER CORPORATION

Salt Lake City, Utah
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D. A. Barr (2)
C. S. Ney (5)
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RESEARCH DIRECTOR

ADDRESS REPLY TO:
P.O. BOX 11299
SALT LAKE CITY, UTAH 84111

April 22, 1971

Mr. D. A. Barr
Acting General Manager
Kennco Explorations (Canada) Limited
P. O. Box 19
Toronto-Dominion Centre
Toronto 1, Ontario, Canada

Dear Mr. Barr:

Attached is a copy of Technical Report 71-10, "Amenability Testing of Samples from Kennco's Huckleberry Project."

The laboratory tests show that an excellent recovery of copper can be obtained from material as represented by the samples submitted to us. In a locked-cycle test, made on a composite of the four samples submitted, 94.36 percent of the copper was recovered in a concentrate assaying 23.63 percent copper. Copper minerals are liberated at a relatively coarse grind (20 to 30 percent plus 100 mesh) but regrinding was required to obtain high-grade (>20 percent Cu) concentrates.

Molybdenite recovery was found to be lower than copper recovery. In the locked-cycle test, only 75.13 percent of the molybdenite was recovered in the copper concentrate. Further tests showed that the molybdenite could be separated from the copper concentrate and recovered as a high-grade product, but insufficient work was done to establish an optimum molybdenite recovery process for this mineralization.

Mr. D. A. Barr

In general, the test results indicate that a straightforward, conventional copper flotation process would be applicable to processing of the Huckleberry ore. However, the studies were not sufficiently extensive to establish detailed flowsheet parameters.

Very truly yours,

H. R. Spedden

HRS/cp

TABLE OF CONTENTS

LIST OF TABLES

This report contains 13 pages.

INTRODUCTION AND SUMMARY

Laboratory amenability tests have been made on diamond drill core samples from the Huckleberry deposit being prospected by Kennco Explorations (Canada) Limited. The study was requested by Mr. C. J. Sullivan in a letter to A. W. Last, dated August 10, 1970.

Economic minerals in the Huckleberry samples were chalcopyrite and molybdenite. The chalcopyrite was sufficiently liberated to achieve high flotation recovery after grinding as coarse as 30 percent plus 100 mesh. Molybdenite was liberated much less effectively than chalcopyrite. With a relatively coarse primary grind employed, it was necessary to regrind rougher concentrates prior to cleaning in order to obtain high grade (>20 percent Cu) final concentrates.

A locked-cycle flotation test, made on a composite of the samples submitted by Kennco, recovered 94.36 percent of the copper and 75.13 percent of the molybdenite in the feed. Cleaner concentrates assayed 23.63 percent Cu and 1.63 percent MoS_2. In a batch test, concentrates of similar composition were processed to separate and recover the molybdenite; 66.96 percent of the molybdenite in the feed material was recovered in the form of a concentrate assaying 92.60 percent MoS_2. The copper recovery tests and the copper-molybdenite separation test indicated no factors to be present which would adversely affect milling of ore having the mineralogical characteristics of the samples submitted for study.

II. SAMPLE DESCRIPTION

Four samples were submitted for test work. The samples were prepared from rejects from the sampling of diamond drill core from holes 10, 13 and 14.

Upon receipt, the four samples were stage-crushed to minus 10 mesh, mixed, sampled for assay and split into test charges. Table I presents assay data on the four samples and the composite.

An examination showed that chalcopyrite and molybdenite were the only economic minerals present in the samples in significant amounts. There was minimal oxidation of sulfide minerals; both chalcopyrite and pyrite presented bright, clean surfaces. Most of the chalcopyrite occurred as coarse (> 150 micron diameter) grams but a small amount was sufficiently fine that they would be incompletely liberated by a 100 mesh grind. The size-characteristics of the molybdenite were not estimated.

TABLE 1

ANALYSES OF HUCKLEBERRY DIAMOND DRILL CORE SAMPLES SUBMITTED FOR AMENABILITY TESTING

Sample Lot Number	Drill Hole Number	Analysis, Percent				
		Cu(T)	Cu(NS)	MoS_2	Fe	S
1	10	0.32	0.014	0.038	3.0	1.5
2	13	.50	.014	.022	6.4	1.9
3	14	.48	.012	.053	4.3	1.6
4	14	.36	.012	.030	5.4	1.3
Composite 1/		.43	.020	.037	4.8	1.4

1/ Equal-weight composite from sample Lots 1, 2, 3 and 4.

III. RECOVERY OF COPPER AND MOLYBDENITE FROM ORE SAMPLES

Batch flotation tests were made to determine grinding requirements for effective liberation of ore minerals and to establish a suitable flotation reagent regime.

The tests showed that Z-6 (potassium amyl xanthate) was a suitable collector when used at a level of 0.07 pound per ton in conjunction with a pulp alkalinity, from lime, of pH 9.5 to 10.0. A mixture of 50 percent pine oil and 50 percent Dowfroth 250 produced a good quality froth. The sulfide minerals floated rapidly and the froth column was substantially barren after 5 minutes flotation time. A typical flotation test procedure, with metallurgical data, is presented in Table II.

A series of tests, conducted as described in Table II, showed only slight variation in the copper content of tailing as the primary grind was coarsened from 10 percent plus 100 mesh to 30 percent plus 100 mesh. The pertinent data are presented in Table III.

Using the test procedure described in Table II and a primary grind of approximately 20 percent plus 100 mesh, tests were made on each of the four lot samples submitted by Kennco. The metallurgical results, only, of these tests are presented in Table IV.

The metallurgical data developed from the four samples and the composite sample (Table II) are very consistent. More than 90 percent of the copper and 70 to 85 percent of the molybdenite were recovered from the ore in the form of a low-grade rougher concentrate, amounting to 5 to 7 weight percent of the ore. After regrinding, the bulk of the diluting minerals were rejected into a flotation cleaner tailing, together with a significant percentage of the copper. Recleaner flotation responses were typical of copper flotation metallurgy. It was concluded from these data (Tables II and IV) that the four samples submitted by Kennco were sufficiently similar that further studies could be made on a composite sample.

A locked-cycle test was made using the composite sample as feed. The test procedure and the metallurgical results obtained from the test are presented in Table V. In this test, 94.36 percent of the copper and 75.14 percent of the molybdenite in the feed were recovered in a concentrate assaying 23.63 percent Cu and 1.63 percent MoS_2. Of interest, the test showed that most of the copper and molybdenite that had reported into cleaner tailing products in batch tests was recovered in a cyclic test approximating floation plant practice.

An examination of a flotation tailing showed no oxide molybdenum minerals to be present. Molybdenite losses were divided between (1) very fine inclusions of molybdenite in gangue and (2) very coarse, approx-

TABLE II

TYPICAL BATCH FLOTATION TEST PROCEDURE AND METALLURGICAL DATA, HUCKLEBERRY COMPOSITE SAMPLE

Grinding:	1000 grams 10 mesh ore 665 ml water 0.8 gram hydrated lime Grind 8 minutes with 9 Kg ball charge
Rougher Flotation:	0.07 pound Z-6 per ton 0.16 pound frother per ton Float 8 minutes in 1000 gram Denver cell at 2000 rpm Tailing pH = 9.5
Concentrate Regrind:	Wet screen on 270 mesh sieve. Grind oversize, with 1.0 gram hydrated lime, 10 minutes with 3 Kg ball charge. Combine reground solids with screen undersize.
Cleaner Flotation:	Condition reground rougher concentrate with 0.01 pound Z-6 and 0.05 pound frother per ton of ore feed. Float 4 minutes in 500 gram Denver cell at 1500 rpm. Tailing pH = 11.6
Recleaner Flotation:	Condition cleaner flotation froth with 1.0 gram hydrated lime and 0.05 pound frother per ton. Float 3 minutes in 250 gram Denver cell at 1200 rpm. Tailing pH = 11.4

Metallurgical Results

Product	Weight Percent	Assay, Percent Cu	Assay, Percent MoS_2	Distribution, Percent Cu	Distribution, Percent MoS_2
Cleaner Concentrate 1/	1.74	28.9	1.869	95.02	71.12
Recleaner Tailing	.48	1.47	.159	1.35	1.75
Cleaner Tailing	6.33	.19	.022	2.26	3.06
Rougher Tailing 2/	91.45	.008	.012	1.37	24.07
Calculated Feed	100.00	.53	.046	100.00	100.00

1/ Additional assays: 26.0% Fe, 31.9% S, 7.8% Insol, 1.81 oz Ag per ton.

2/ Sizing: 21.6 percent plus 100 mesh

TABLE III

RELATIONSHIP BETWEEN PRIMARY GRINDING FINENESS AND ASSAYS OF FLOTATION TAILING - HUCKLEBERRY COMPOSITE SAMPLE

Flotation Tailing, Percent +100 Mesh	Flotation Tailing Assays, Percent	
	Cu	MoS_2
11.0	0.019	0.010
21.6	.015	.010
31.6	.020	.010

TABLE IV

LABORATORY FLOTATION TEST DATA - HUCKLEBERRY LOT SAMPLES

Sample	Lot 1 DDH 10	Lot 2 DDH 13	Lot 3 DDH 14	Lot 4 DDH 14
Recleaner Concentrate				
Percent Cu	23.9	26.0	24.2	18.8
Percent Distribution Cu	58.56	65.05	66.35	69.41
Percent MoS_2	3.14	1.10	3.40	1.57
Percent Distribution MoS_2	68.11	58.00	74.28	62.83
Weight, Percent of Feed	.81	1.32	1.33	1.45
Recleaner Tailing				
Percent Cu	4.42	5.11	5.32	3.60
Percent Distribution Cu	4.80	6.57	7.14	5.32
Percent MoS_2	.267	.100	.284	.154
Percent Distribution Cu	2.68	2.80	2.95	2.47
Weight, Percent of Feed	.36	.68	.65	.58
Cleaner Tailing				
Percent Cu	2.84	3.09	2.30	1.72
Percent Distribution Cu	30.90	25.18	20.72	19.32
Percent MoS_2	.093	.053	.103	.073
Percent Distribution MoS_2	8.84	9.20	7.37	8.81
Weight, Percent of Feed	3.60	4.30	4.37	4.41
Rougher Tailing				
Percent Cu	.020	.018	.030	.025
Percent Distribution Cu	5.74	3.20	5.79	5.95
Percent MoS_2	.008	.008	.010	.010
Percent Distribution MoS_2	20.37	30.00	15.40	25.89
Percent +100 Mesh Material	20.9	18.0	20.3	21.5
Weight, Percent of Feed	95.23	93.70	93.65	93.56
Calculated Feed				
Percent Cu	.33	.53	.52	.39
Percent MoS_2	.040	.025	.051	.037

TABLE V

LOCKED-CYCLE FLOTATION TEST, HUCKLEBERRY COMPOSITE SAMPLE

Grind:	2 @ 1000 gram charges ground separately for 15 minutes at 60 percent solids with 9 Kg ball charge. 0.4 gram hydrated lime added to each charge.
Rougher Flotation:	Ground pulps combined in 2000 gram Denver cell and conditioned with 0.07 pound Z-6 and 0.10 pound frother (50%) pine oil - 50% Dow froth 250) per ton. Floated for 8 minutes at 2000 rpm. Tailing pH = 9.8
Regrind:	Rougher froth wet-screened on 270 mesh sieve. Plus 270 mesh material plus 0.3 gram hydrated lime ground 10 minutes with 3 Kg ball charge. Ground solids combined with 270 mesh screen undersize material.
Cleaner Flotation:	Reground rougher concentrate conditioned in 250 gram Denver cell with 0.025 pound frother per ton. Floated 4 minutes at 1200 rpm. Tailing pH = 11.5
Recleaner Flotation:	Cleaner froth conditioned with 0.025 pound frother per ton in 250 gram Denver cell. Floated 3 minutes at 1200 rpm. Tailing pH = 11.4

Metallurgical Results (Combined cycles 4 and 5 of 5 cycle test)

Product	Weight Percent	Assay, Percent Cu	Assay, Percent MoS_2	Distribution, Percent Cu	Distribution, Percent MoS_2
Concentrate	1.74	23.63	1.63	94.36	75.14
Tailing 1/	98.26	.025	.0096	5.64	24.86
Calculated Feed	100.00	.44	.038	100.00	100.00

1/ Sizing - 24.7 percent plus 100 mesh.

imately equidimensional, molybdenite grains. Apparently the latter were of sufficiently large mass that they could not be floated effectively; with closed-circuit grinding, losses of the second type would be reduced as a consequence of the differential grinding of minerals of high specific gravity.

IV. SEPARATION OF MOLYBDENITE FROM COPPER CONCENTRATE

A single test was made to investigate the recovery of molybdenite from the copper-molybdenite concentrates recovered from the ore. This single test is not sufficient to define an optimum process applicable to the Huckleberry ore but did suffice to demonstrate that the molybdenite occurs in such a form that it can be recovered as a high-grade, saleable, product.

For this test, most of the "ore" samples on hand were combined and processed to recover a concentrate assaying 26.7 percent copper and 1.42 percent MoS_2 as feed to the copper-molybdenite separation test.

The concentrate was conditioned with sufficient lime to yield a pH of 12.5. The slurry was then thickened by settling and decantation to 50 percent solids. The thickened pulp was heated to boiling and boiling continued for 1 hour; $CuSO_4$ (0.37 pound per ton solids) was added when boiling began and sodium hypochlorite (0.28 pound per ton) was added after 30 minutes of boiling. The pulp was then cooled to 25° to 30° C.

The pulp was conditioned in a laboratory flotation cell with LR-744 (0.74 pound per ton), burner oil (0.08 pound per ton) and methyl amyl alcohol (.035 pound per ton) and then floated for 4 minutes. The froth was conditioned with LR-744 (0.19 pound per ton) and burner oil (0.04 pound per ton) and re-floated for 3 minutes. The two tailing products were combined as the Copper Concentrate; the froth product was designated Molybdenite Rougher Concentrate. A metallurgical balance for this portion of the test is presented as A - Initial Separation in Table VI.

The molybdenite rougher concentrate was filtered, dried and heat-treated in an open pan in a muffle furnace for 30 minutes at a temperature varying between 260°C and 305°C (500°F. and 580°F.). It was then cooled, pulped in a laboratory flotation cell and conditioned with sodium silicate (0.05 pound per ton), burner oil (0.34 pound per ton) and methyl amyl alcohol (0.07 pound per ton) and floated for 4 minutes. The tailing product was designated Molybdenite Tailing; in practice, this product would be combined with the copper concentrate.

The rougher molybdenite froth product was cleaned twice by re-floation. Additions were 0.09 pound LR 744 and 0.035 pound methyl amyl alcohol per ton to the cleaner stage and 0.04 pound LR 744 and 0.07 pounds methyl amyl alcohol to the recleaner stage. The two cleaner tailing products were combined; in practice, these products would be re-cycled to recover a portion of the contained molybdenite. A metallurgical balance for this portion of the test will be found as B - Molybdenite Upgrading in Table VI. The metallurgical results for the combination of the two portions of the test are presented as C - Combined Metallurgical Results in Table VI.

TABLE VI

SEPARATION OF MOLYBDENITE FROM CONCENTRATES RECOVERED FROM HUCKLEBERRY "ORE" SAMPLES - METALLURGICAL DATA

A - Initial Separation

Product	Weight Percent	Assay, Percent Cu	Assay, Percent MoS_2	Distribution, Percent Cu	Distribution, Percent MoS_2
Copper Concentrate	84.45	26.7	0.142	87.48	8.12
Molybdenite Rougher Concentrate	15.55	20.75	8.71	12.52	91.88
Calculated Feed	100.00	25.77	1.475	100.00	100.00

B - Molybdenite Upgrading

Product	Weight Percent	Assay, Percent Cu	Assay, Percent MoS_2	Distribution, Percent Cu	Distribution, Percent MoS_2
Molybdenite Tailing	87.86	22.6	.65	95.83	6.65
Molybdenite Cleaner Tailing	5.38	15.1	32.53	3.92	20.39
Molybdenite Cleaner Concentrate	6.76	.77	92.60	.25	72.96
Calculated Feed	100.00	20.72	8.58	100.00	100.00

C - Combined Metallurgical Results

Product	Weight Percent	Assay, Percent Cu	Assay, Percent MoS_2	Distribution, Percent Cu	Distribution, Percent MoS_2
Copper Concentrate	84.45	26.7	.142	87.50	8.23
Molybdenite Tailing	13.66	22.6	0.65	11.98	6.10
Molybdenite Cleaner Tailing	.84	15.1	32.53	.49	18.73
Molybdenite Concentrate 1/	1.05	.77	92.60	.03	66.94
Calculated General Concentrate	100.00	25.77	1.454	100.00	100.00

1/ Additional Assays: 1.2% Fe, 3.9% Insol, 0.008% Pb, 0.02% Re

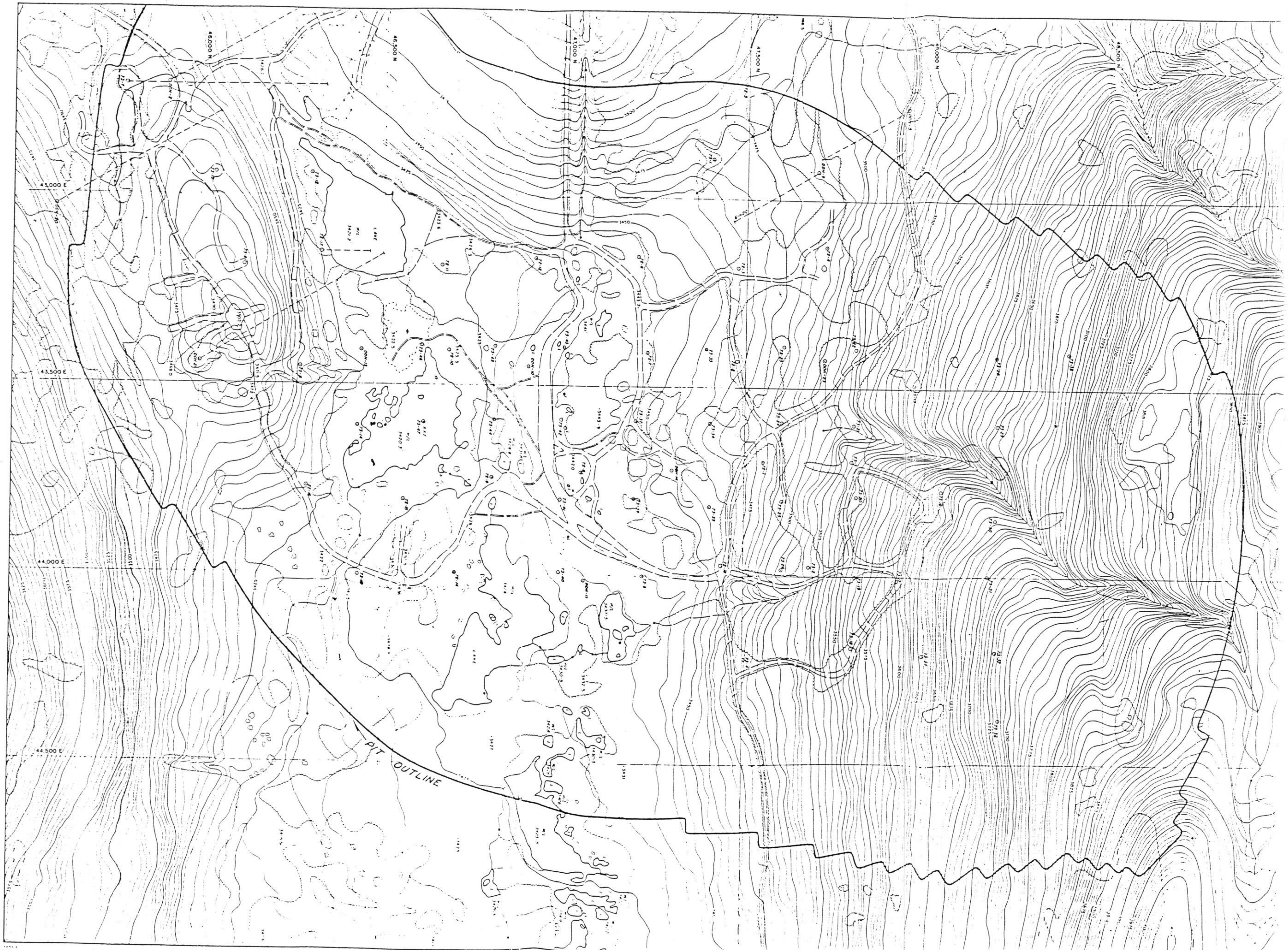
In the test 66.94 percent of the molybdenite in the general flotation concentrate was recovered as a product assaying 92.60 percent MoS_2 and 0.77 percent Cu; an additional 18.73 percent of molybdenite reported into cleaner tailing products which, in practice, would re-cycle for additional molybdenite recovery. Recovered molybdenite concentrates were low in lead content (0.008 percent Pb) and low in rhenium content (0.02 percent Re).

V. CONCLUSIONS

The amenability test work has demonstrated that the copper minerals in the Huckleberry samples are readily amenable to concentration by flotation, with a recovery level of 95 percent indicated to be achievable in practice.

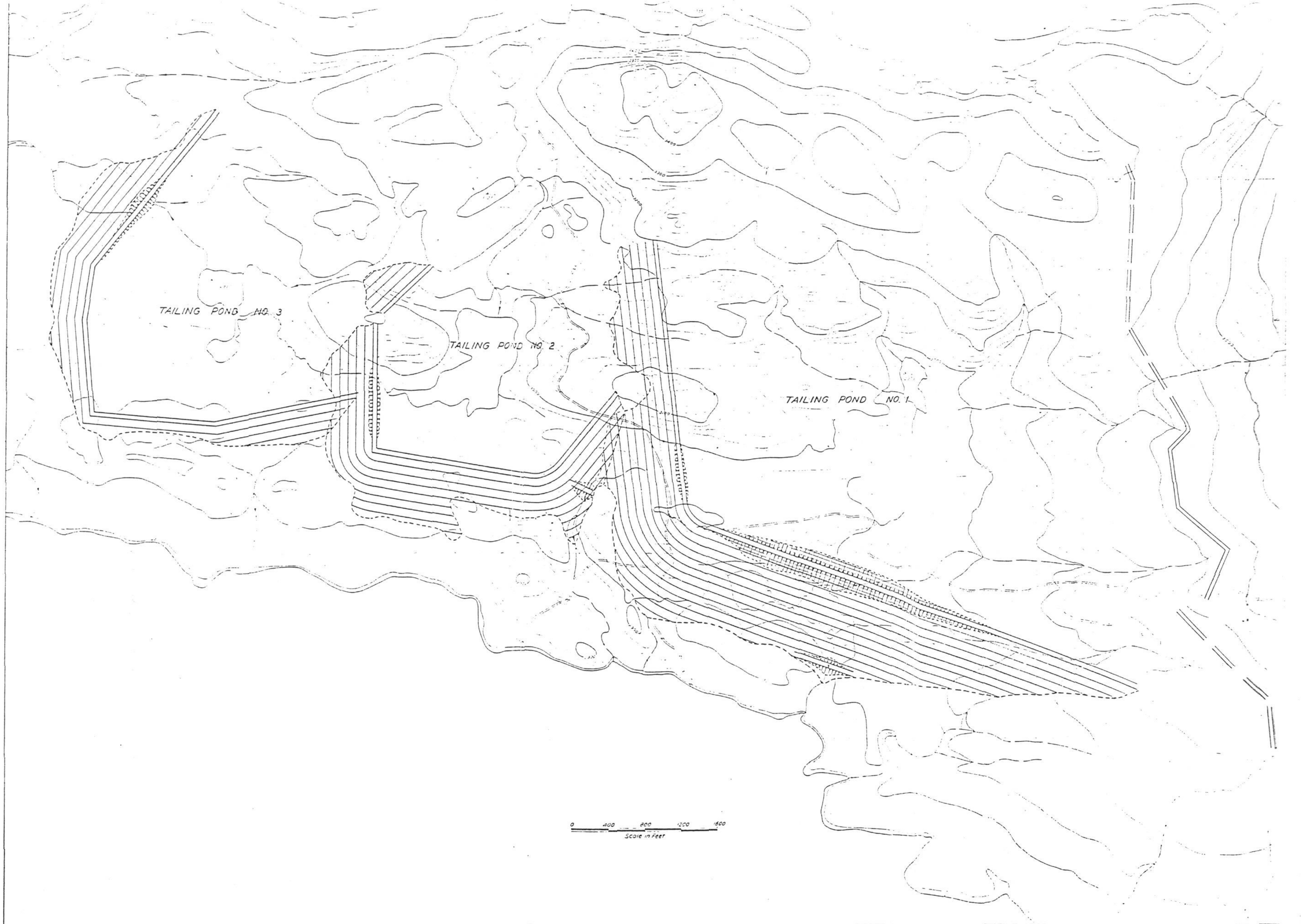
Recovery of molybdenite is poorer than recovery of copper. The test data indicate that, in practice, approximately 75 percent of the molybdenite can be recovered in a general flotation concentrate. Recovery of molybdenite from the general flotation concentrate into a molybdenite concentrate cannot be projected; however, the test data does indicate that a high-grade molybdenite product should be obtainable.

Insufficient test work has been done on Huckleberry ore samples to develop a recommended concentrator flowsheet. The available data do indicate, however, that a straight-forward, conventional, process will be sufficient for recovery of metal values from this relatively simple ore.



COMPOSITE FROM [illegible] AERIAL SURVEY

DATE REVISED / BY	DRAWN BY / CHECKED / APPROVED / DATE JANUARY 24, 1974	THE GRANBY MINING CO. LTD. / PROJECT HUCKLEBERRY / SCALE	TITLE PIT AREA TOPOGRAPHY / NO.



DATE REVISED	BY			
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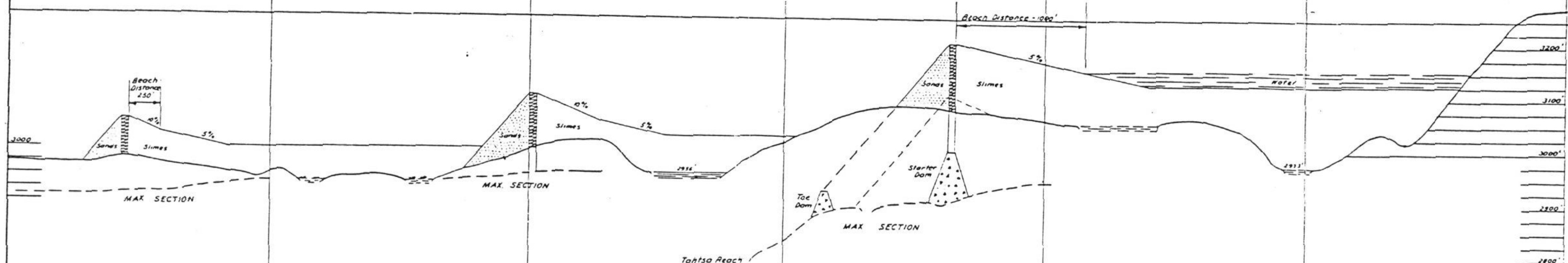
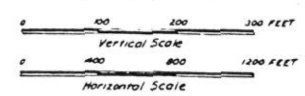
		CHECKED	PROJECT: HUCKLEBERRY	
		APPROVED		NO
		DATE April 9 1974	SCALE: 1 INCH = FEET	

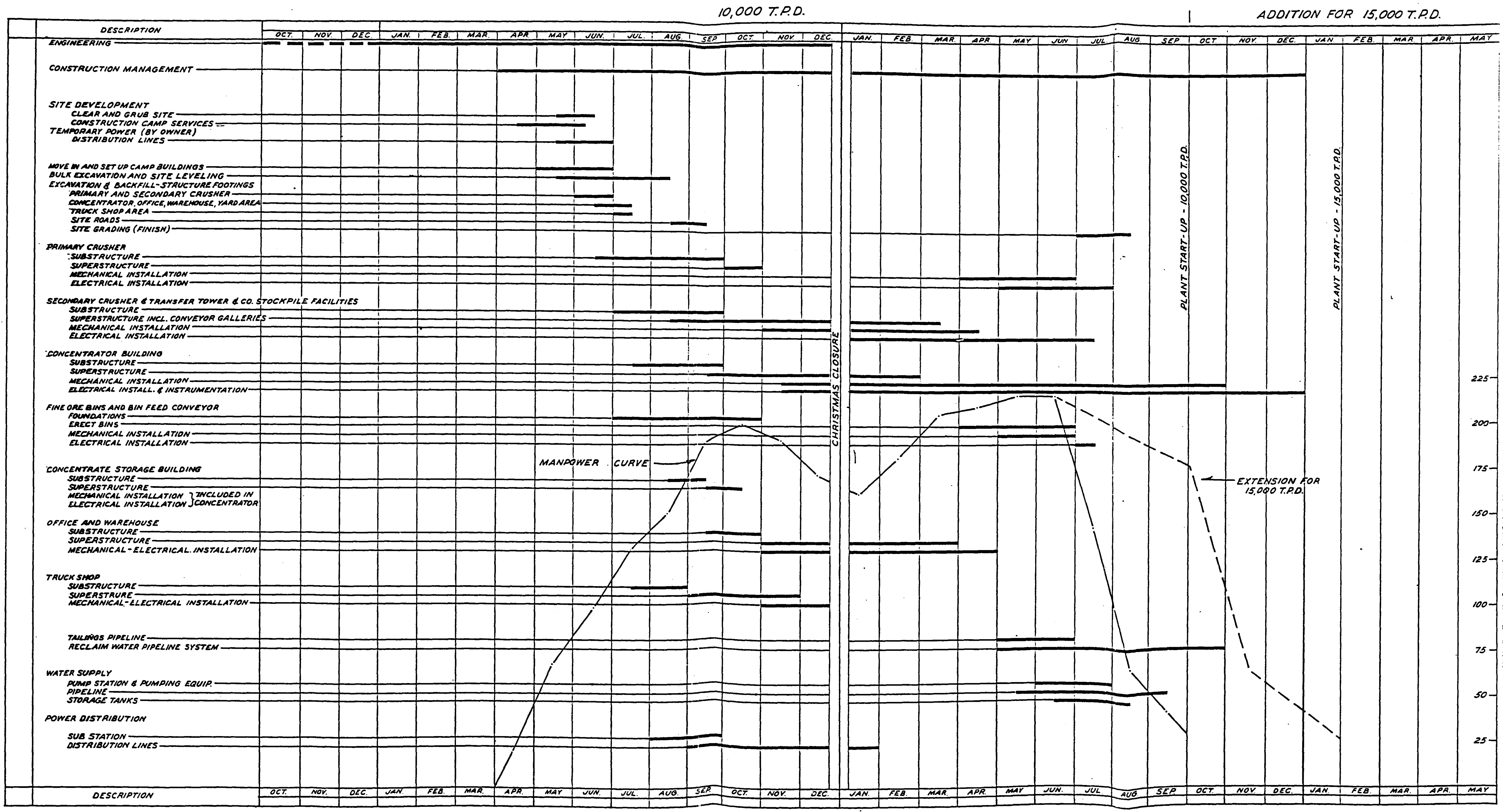


SCALE OF SECTION



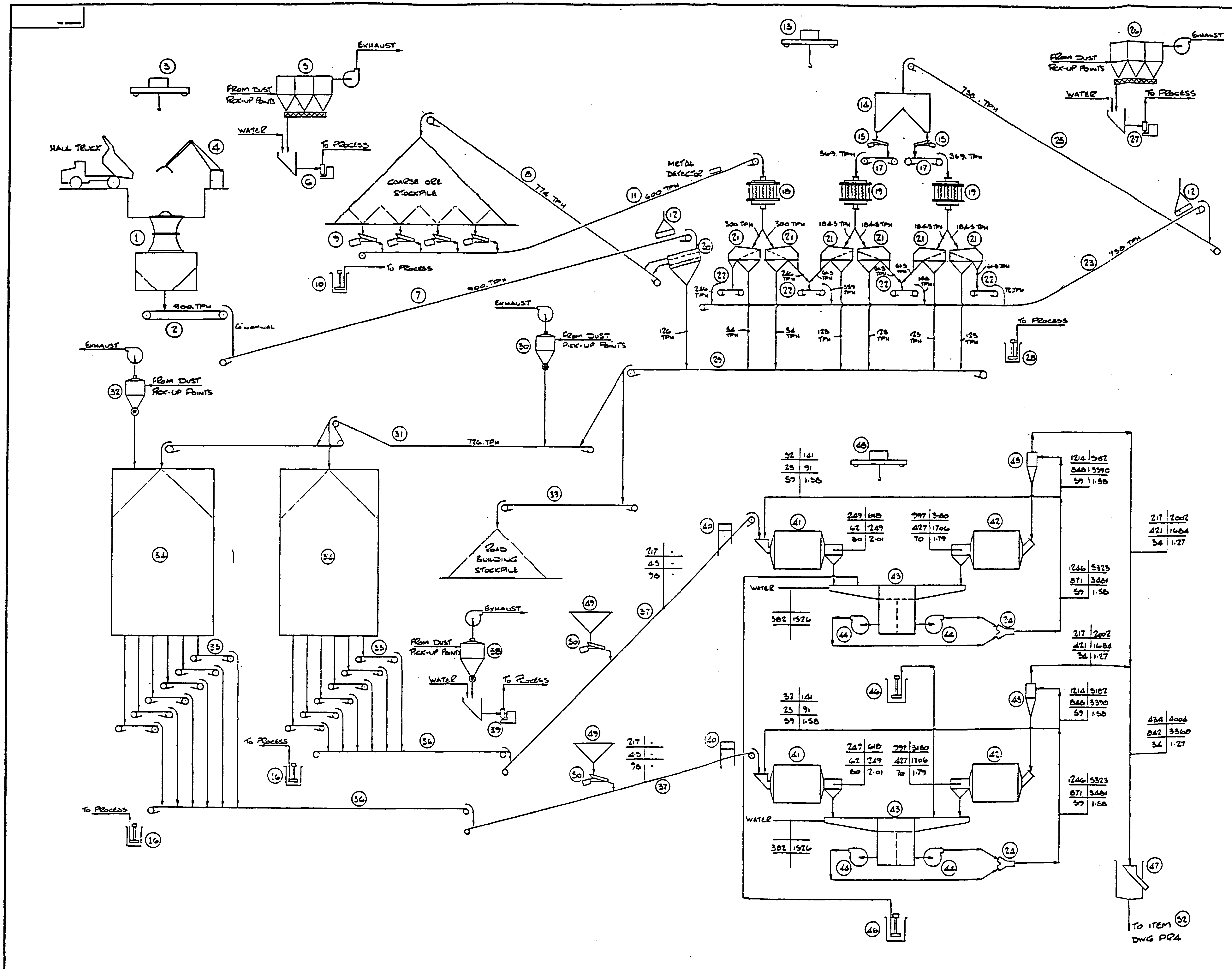
DRAWN BY: H G T	THE GRANBY MINING CO. LTD.	TITLE TAILINGS DISPOSAL (X-SECTION)
DATE Ap. 1 9 1974	PROJECT: HUCKLEBERRY	NO.
	SCALE: 1 INCH = FEET	

PROPOSED PROJECT SCHEDULE



Note: Manpower curve excludes personnel for townsite, tailings development, transmission power to site, pit development and exploration crews.

Figure XII- 1



EQUIPMENT LIST

EQUIP'T N°	N° REQ'D	DESCRIPTION
1	1	54"x74" PRIMARY GYRATORY CRUSHER
2	1	72" WIDE x 22' LG APRON FEEDER
3	1	50T/5T CAP: OVERHEAD TRAVELLING CRANE
4	1	ROCK GRAPPLE
5	1	DUST COLLECTION BAG HOUSE
6	1	SLURRY TANK & PUMP
7	1	42" WIDE x 210' LG BELT CONVEYOR
8	1	42" WIDE x 370' LG BELT CONVEYOR
9	4	42" WIDE VIBRATING FEEDER
10	1	2"x2" VERTICAL SUMP PUMP
11	1	42" WIDE x 400' LG BELT CONVEYOR
12	2	TRAMP IRON MAGNET
13	1	30T/5T CAP: OVERHEAD TRAVELLING CRANE
14	1	100T CAPACITY SURGE BIN
15	2	54" WIDE VIBRATING FEEDER
16	2	2"x2" VERTICAL SUMP PUMPS
17	2	36" WIDE x 18' LG RETRACTABLE BELT CONVEYORS
18	1	7'-0" SECONDARY CONE CRUSHER
19	2	7'-0" TERTIARY CONE CRUSHERS
20	1	8'x20' D.D. PRIMARY SCREEN
21	6	6'x12' ROD DECK SCREENS
22	4	30" WIDE x 24' LG BELT CONVEYORS
23	1	48" WIDE x 280' LG BELT CONVEYOR
24	2	14" TECH TAYLOR VALVES
25	1	48" WIDE x 210' LG BELT CONVEYOR
26	1	DUST COLLECTION BAG HOUSE
27	1	SLURRY TANK & PUMP
28	1	2"x2" VERTICAL SUMP PUMP
29	1	36" WIDE x 300' LG BELT CONVEYOR
30	1	DUST COLLECTION BAG HOUSE
31	1	36" WIDE x 825' LG BELT CONVEYOR
32	1	DUST COLLECTION BAG HOUSE
33	1	24" WIDE x 280' LG BELT CONVEYOR
34	2	70'ø x 70' HIGH F.O. BINS
35	12	36" WIDE x 30' LG BELT FEEDER CONVEYORS
36	2	30" WIDE x 55' LG BELT CONVEYORS
37	2	30" WIDE x 112' LG BELT CONVEYORS
38	1	DUST COLLECTION BAG HOUSE
39	1	SLURRY TANK & PUMP
40	2	BELT SCALES
41	2	13'ø x 18' ROD MILLS
42	2	13½'ø x 28' BALL MILLS
43	2	ROD/BALL MILL DISCHARGE PUMP BOXES
44	4	14"x14" HORIZONTAL PUMPS
45	2	9x20"ø CYCLOPAC
46	2	3"x3" VERTICAL SUMP PUMPS
47	1	AUTOMATIC SAMPLER
48	1	40T/2T OVERHEAD TRAVELLING CRANE
49	2	EMERGENCY DUMP HOPPERS
50	2	30" WIDE VIBRATING FEEDERS

LEGEND

SOLIDS STPH	SLURRY USGPM
WATER STPH	WATER USGPM
% SOLIDS	S.G. SLURRY

1	REVISION TO MATERIAL BALANCE		
0	ORIGINAL		
No.	DESCRIPTION	DATE	

REVISIONS

DRAWN BY: R.A. DEC/79
CHECKED BY: CRO

CLIENT: GRANBY MINING Cº LTD
LOCATION: TANTSA REACH B.C.

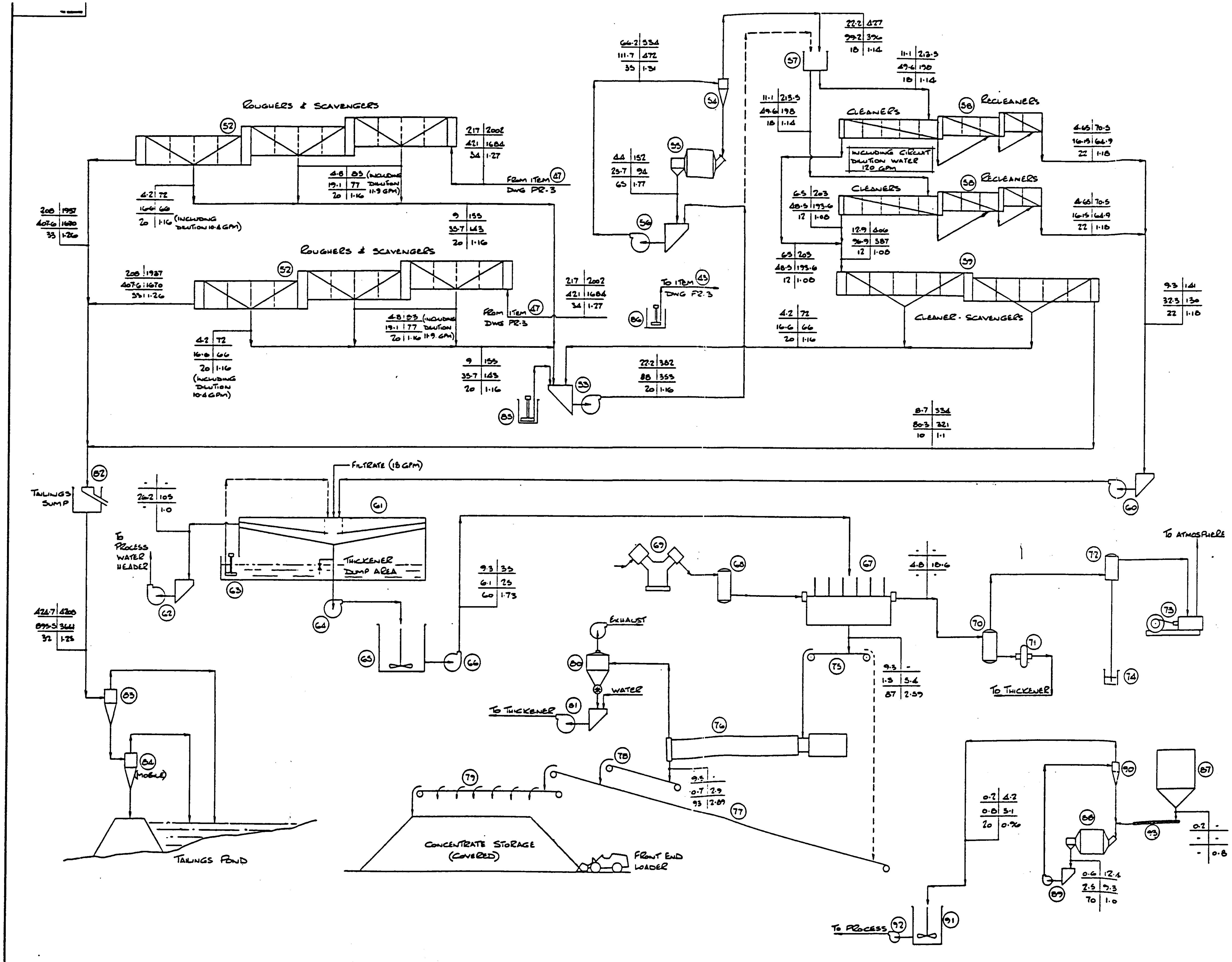
KILBORN

TITLE: HUCKLEBERRY PROJECT CRUSHING & GRINDING FLOWSHEET

PROJECT No. 7051

DRAWING NUMBER: PR-3

REV: 1



EQUIPMENT LIST

EQUIP'T N°	N° REQ'D	DESCRIPTION
51		
52	2	12 x 400 cu ft CELLS
53	1	5'x5' REGRIND MILL CYCLONE FEED PUMP
54	2	6' CYCLONES
55	1	8'ø x 8' REGRIND MILL
56	1	5'x5' REGRIND MILL DISCHARGE PUMP
57	1	2 - WAY PULP DISTRIBUTOR
58	2	10X N° 24 CELLS
59	1	10X N° 30 CELLS
60	1	3'x3' THICKENER FEED PUMP
61	1	40'ø THICKENER (UNBALANCED TRAY)
62	1	2'x2' THICKENER O/FLOW PUMP
63	1	3'x3' THICKENER DUMP RECLAIM PUMP
64	1	3'x3' THICKENER U/FLOW PUMP
65	1	14'ø STOCK TANK
66	1	3'x3' FILTER FEED PUMP
67	1	6'ø x 6 DISC FILTER
68	1	AIR RECEIVER
69	1	AIR COMPRESSOR
70	1	VACUUM RECEIVER
71	1	FILTRATE PUMP
72	1	MOISTURE TRAP
73	1	VACUUM PUMP
74	1	SEAL TANK
75	1	36" WIDE x 15' LG BELT CONVEYOR
76	1	4'ø x 28' LG ROTARY DRYER
77	1	24" WIDE x 200' LG BELT CONVEYOR
78	1	24" WIDE x 20' LG BELT CONVEYOR
79	1	24" WIDE x 65' LG BELT CONVEYOR
80	1	DUST COLLECTION SCRUBBER
81	1	SLURRY TANK & PUMP
82	1	AUTOMATIC SAMPLER
83	3	20" CYCLONE
84	4	6" CYCLONE
85	2	FLOTATION BAY SUMP PUMP 3'x3'
86	1	REGRIND MILL SUMP PUMP
87	1	LIME BIN - 150 TON CAPACITY
88	1	LIME BALL MILL - 4'ø x 5'
89	1	BALL MILL DISCHARGE BOX & PUMP
90	1	6' CYCLONE
91	1	LIME SLURRY STORAGE TANK
92	1	LIME SLURRY PUMP
93	1	SCREW CONVEYOR

LEGEND

SOLIDS STPH	SLURRY U.S.GPM
WATER S.T.P.H	WATER U.S.G.P.M
% SOLIDS	S.G. SLURRY

No.	DESCRIPTION	DATE	BY
1	REVISIONS TO MATERIAL BALANCE	FEB/74	
0	ORIGINAL		

REVISIONS

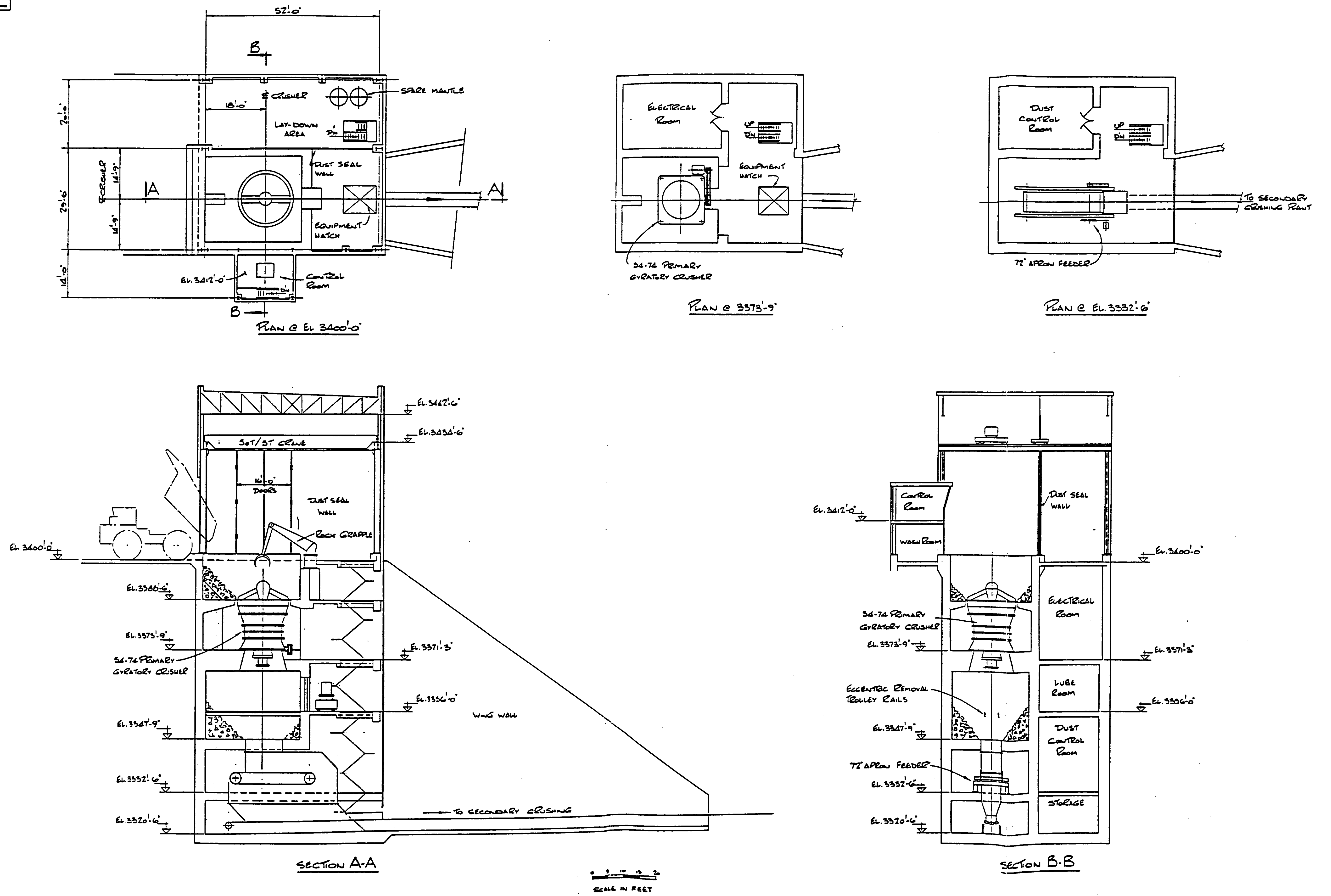
DRAWN BY: R.A. DEC/73
CHECKED BY: ORD JAN/74

CLIENT: GRANBY MINING C° L^TD
LOCATION: TAHTSA REACH. B.C.

KILBORN

TITLE
HUCKLEBERRY PROJECT
FLOTATION, DEWATERING, DRYING FLOWSHEET

PROJECT No. 7031
DRAWING NUMBER FR-4
REV. 1



1	CRUSHER SIZE CHANGED FROM ... CRANE CAPACITY CHANGED FROM 40T/10T TO 50T/5T	FEB/74	
0	ORIGINAL		
NO.	DESCRIPTION	DATE	BY
	REVISIONS		

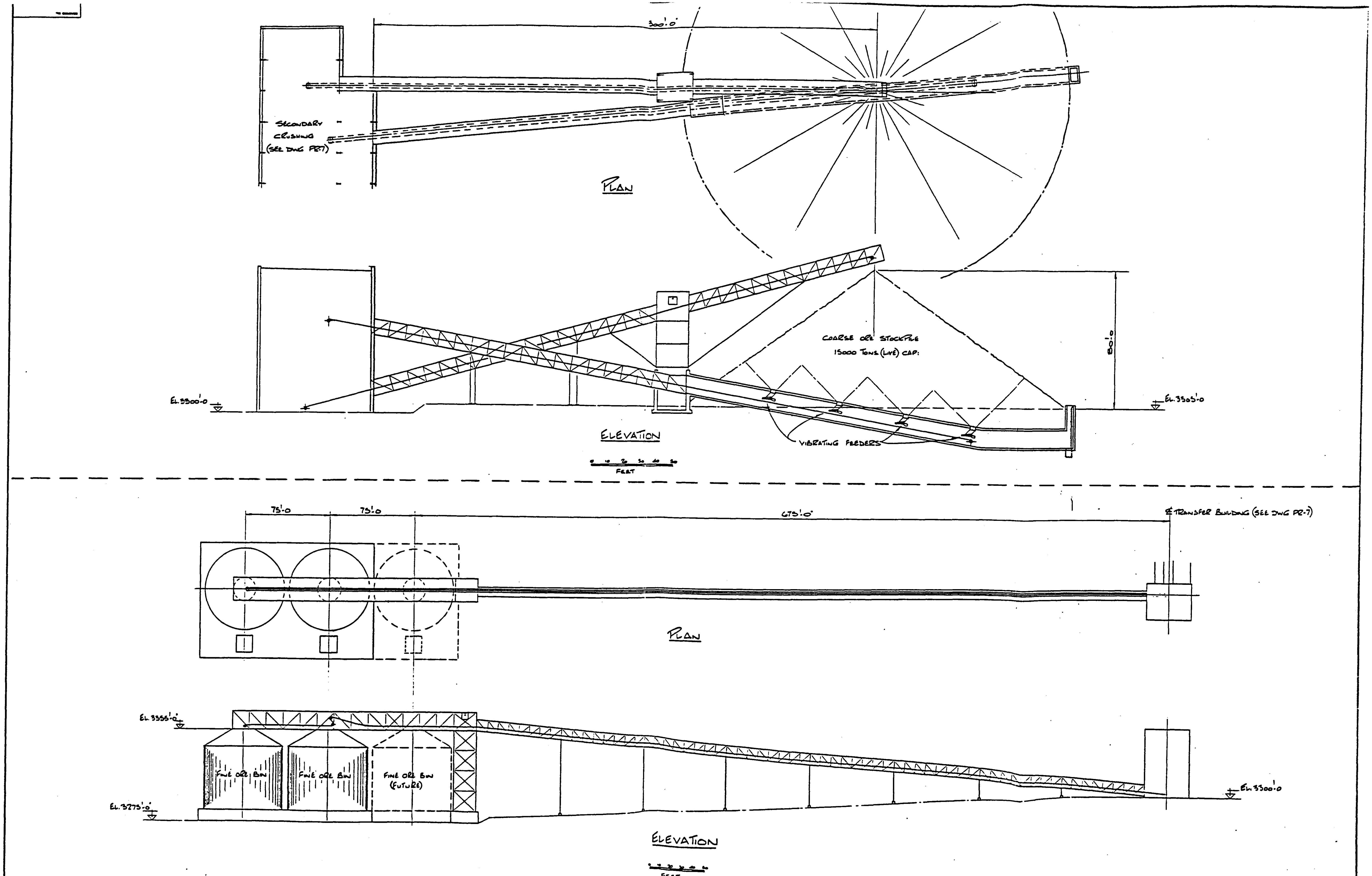
SCALE: 3/32" = 1'-0"
DRAWN BY: R.A. DEC/73
CHECKED BY: JRO JAN/74

CLIENT: GRANBY MINING C° LTD
LOCATION: TAHTSA REACH B.C.

KILBORN

TITLE
HUCKLEBERRY PROJECT
PRIMARY CRUSHING PLANT
GENERAL ARRANGEMENT
PLANS ELEVATION & SECTION

PROJECT NO. 7051
DRAWING NUMBER PR-5
REV. 1



REFERENCE DRAWINGS | REVISIONS | REVISIONS | REVISIONS

SCALE: AS SHOWN
DRAWN BY: RA JAN/74
CHECKED BY: JLO JAN/74

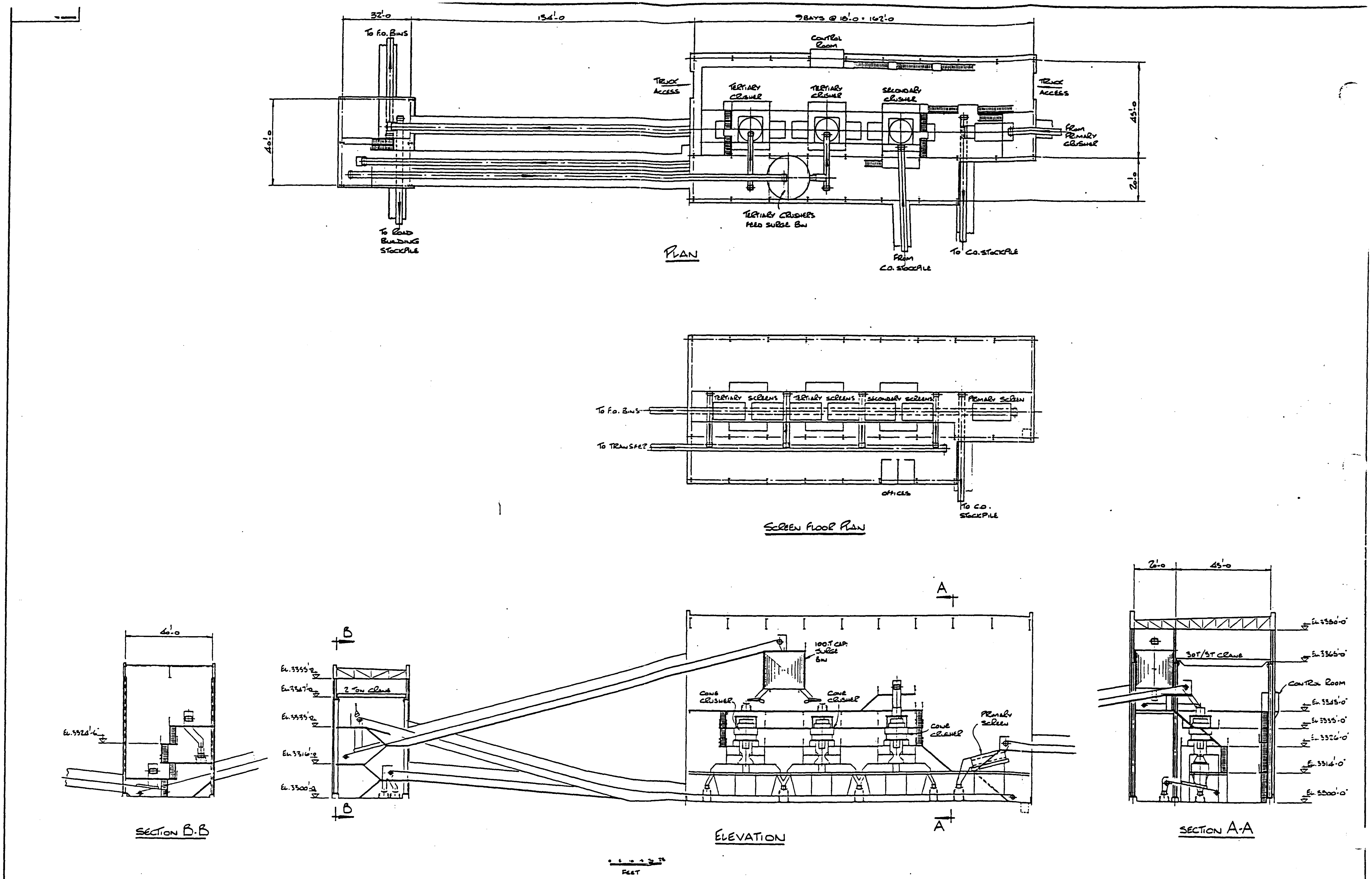
CLIENT: GRANBY MINING Co. LTD
LOCATION: TAHTSA REACH B.C.

KILBORN

TITLE: HUCKLEBERRY PROJECT COARSE ORE STORAGE & FINE ORE STORAGE CONVEYOR SYSTEMS

PROJECT NO. 7051

DRAWING NUMBER PR-6



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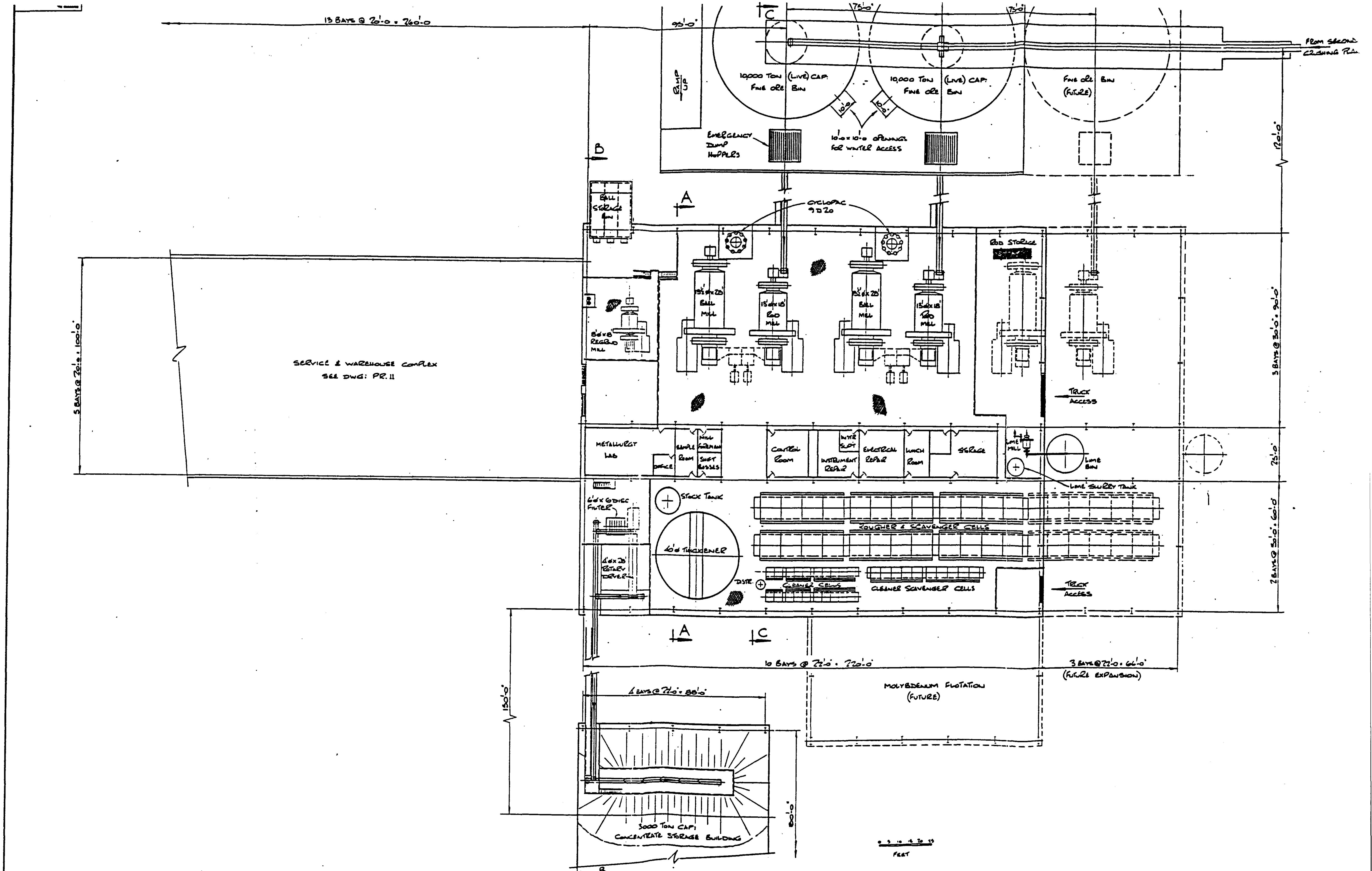
DRAWN BY: R.A. DEC/73	LOCATION: TAHTSA REACH B.C.	HUCKLEBERRY PROJECT	PROJECT No. 705
CHECKED BY:	KILBORN	SECONDARY CRUSHING & SCREENING	DRAWING NUMBER
APPROVED BY:		GENERAL ARRANGEMENT	D2-7
		PLANS ELEVATION & SECTIONS	

REFERENCE DRAWINGS — REVISIONS — DESCRIPTION — DATE — BY



SECTION		
SCALE 1/16" = 1'-0"	DATE	
DESIGNED BY:		
DRAWN BY: P.A.	NOV/73	
CHECKED BY: [illegible]	JAN/74	
APPROVED BY		

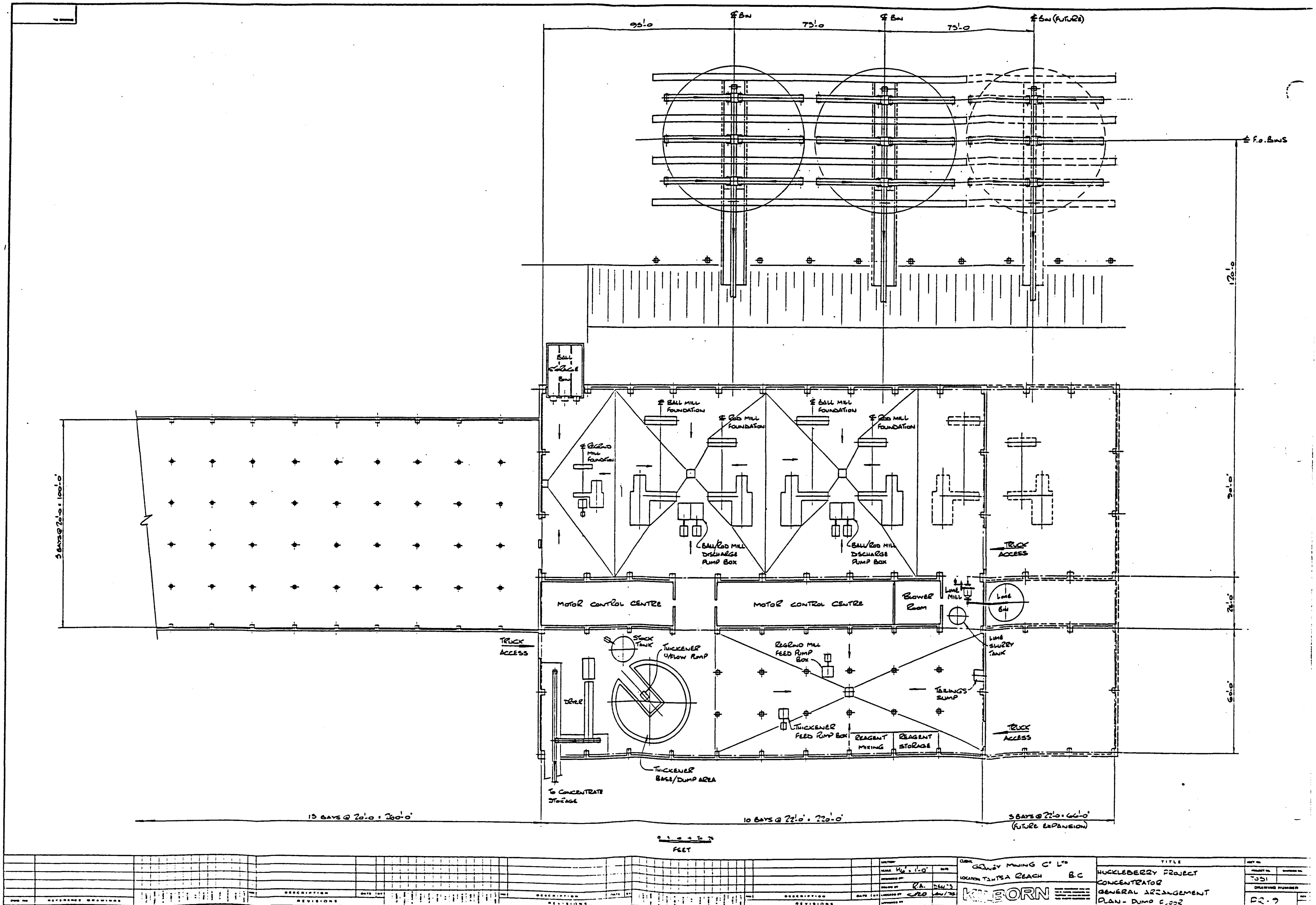
CLIENT GRANBY MINING Co LTD
LOCATION TANTSA REACH B.C.

KILBORN

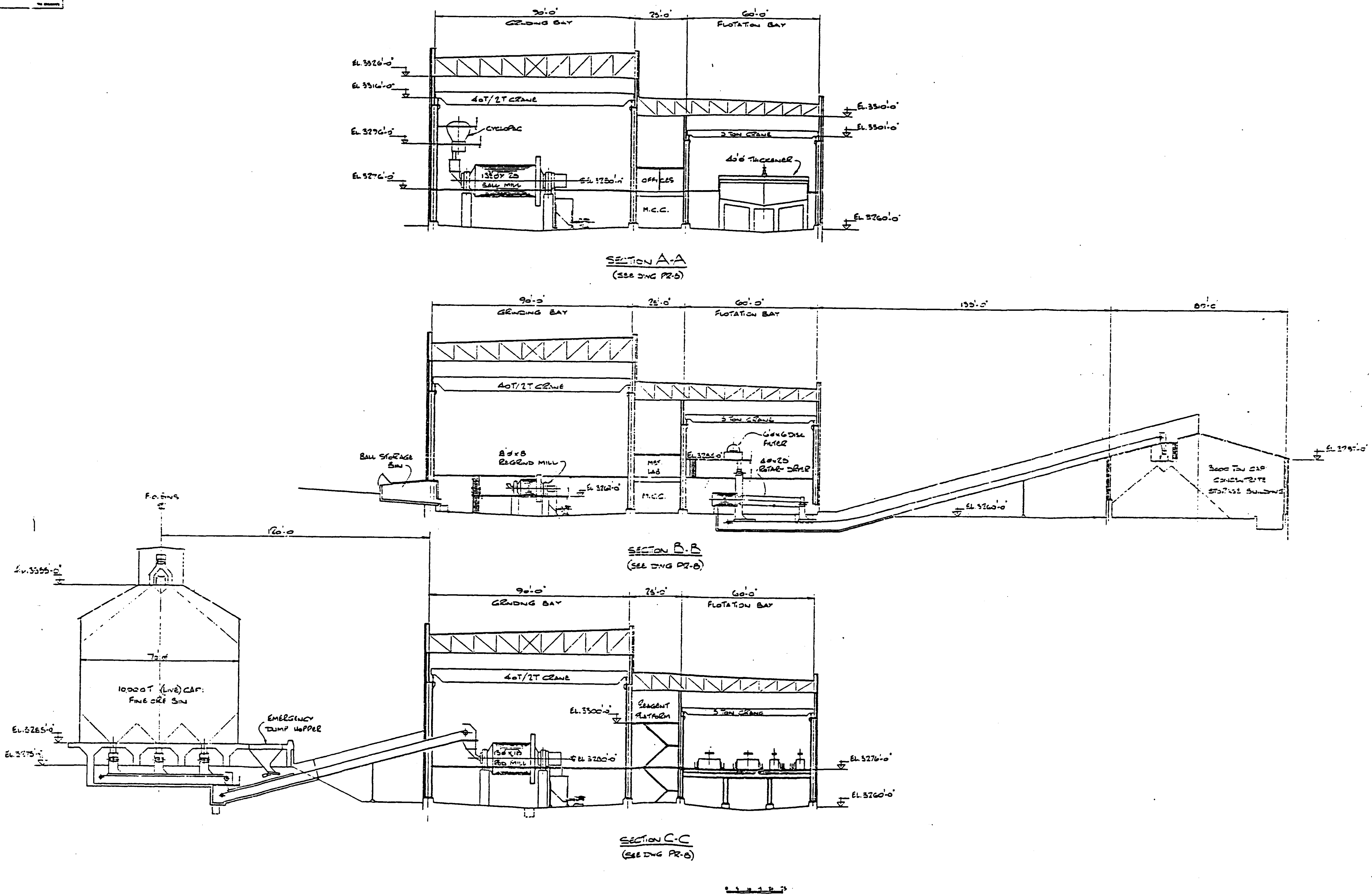
TITLE	
HUCKLEBERRY PROJECT CONCENTRATOR GENERAL ARRANGEMENT PLAN - OPERATING FLOOR	PROJECT No. 7051 DRAWING NUMBER PR-8

REFERENCE DRAWINGS

DESCRIPTION — REVISIONS — DATE — BY



℄ BIN
℄ BIN
℄ BIN (FUTURE)
95'-0
75'-0
75'-0
℄ F.O. BINS
120'-0
BALL STORAGE BIN
℄ BALL MILL FOUNDATION
℄ ROD MILL FOUNDATION
℄ REGRIND MILL FOUNDATION
BALL/ROD MILL DISCHARGE PUMP BOX
TRUCK ACCESS
MOTOR CONTROL CENTRE
MOTOR CONTROL CENTRE
BLOWER ROOM
LIME MILL
LIME BIN
LIME SLURRY TANK
STOCK TANK
THICKENER U/FLOW PUMP
REGRIND MILL FEED PUMP BOX
TAILINGS SUMP
DRYER
THICKENER FEED PUMP BOX
REAGENT MIXING
REAGENT STORAGE
THICKENER BASE/DUMP AREA
TO CONCENTRATE STORAGE
5 BAYS @ 20'-0 = 100'-0
70'-0
26'-0
60'-0
13 BAYS @ 20'-0 = 260'-0
10 BAYS @ 22'-0 = 220'-0
3 BAYS @ 22'-0 = 66'-0
(FUTURE EXPANSION)
FEET
CLIENT GRANBY MINING CO LTD
LOCATION TAHTSA REACH B.C
SCALE 1/16" = 1'-0"
DRAWN BY R.A.
CHECKED BY
KILBORN
TITLE
HUCKLEBERRY PROJECT
CONCENTRATOR
GENERAL ARRANGEMENT
PLAN - DUMP FLOOR
PROJECT NO. 7051
DRAWING NUMBER
F2-9
REFERENCE DRAWINGS
DESCRIPTION
REVISIONS



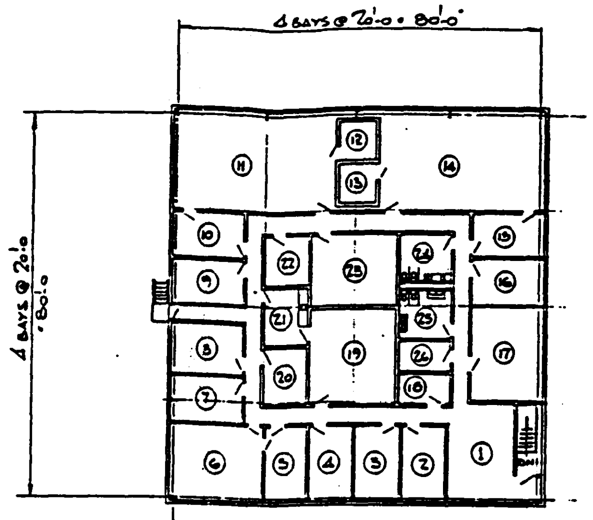
CLIENT GRANBY MINING CO LTD
LOCATION TAHTSA REACH B.C.
SCALE 1/16" = 1'-0"
DRAWN BY G.A.
KILBORN

TITLE
HUCKLEBERRY PROJECT
CONCENTRATOR
GENERAL ARRANGEMENT
CROSS SECTIONS

PROJECT NO. 7031
DRAWING NUMBER PR-10

UPPER FLOOR KEY

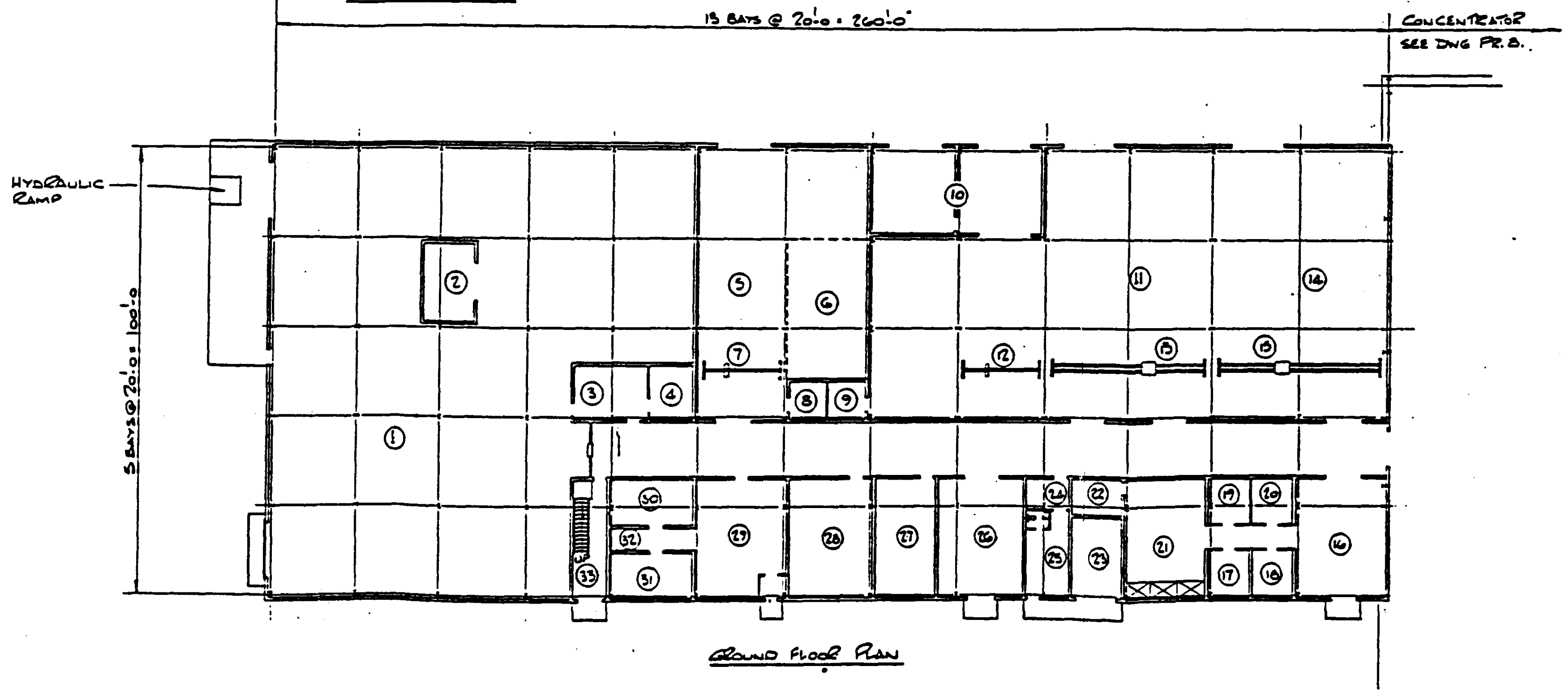
1. RECEPTION
2. SECURITY OFFICE
3. SAFETY OFFICE
4. MILL SUPT
5. SECRETARY
6. MANAGER
7. MINE SUPT
8. PLANT SUPT
9. GEOLOGIST
10. CHIEF ENGINEER
11. ENGINEERING OFFICE
12. ENGINEERING VAULT
13. ACCOUNTING VAULT
14. ACCOUNTING OFFICE
15. CHIEF ACCOUNTANT
16. PERSONNEL
17. PAYMASTER
18. TELEPHONE
19. CONFERENCE
20. STORAGE
21. KITCHEN
22. BLUEPRINT MACHINE
23. MAINTENANCE PLANNING
24. WOMEN'S
25. MEN'S
26. XEROX MACHINE



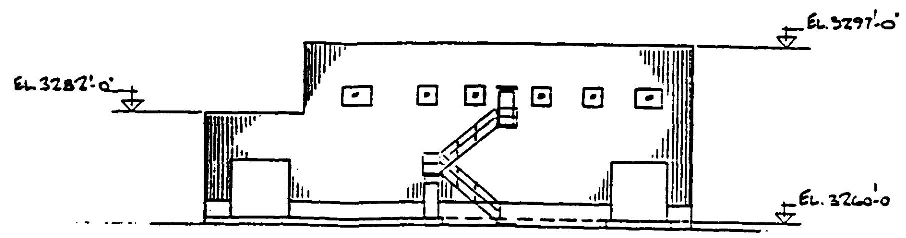
UPPER FLOOR PLAN

GROUND FLOOR KEY

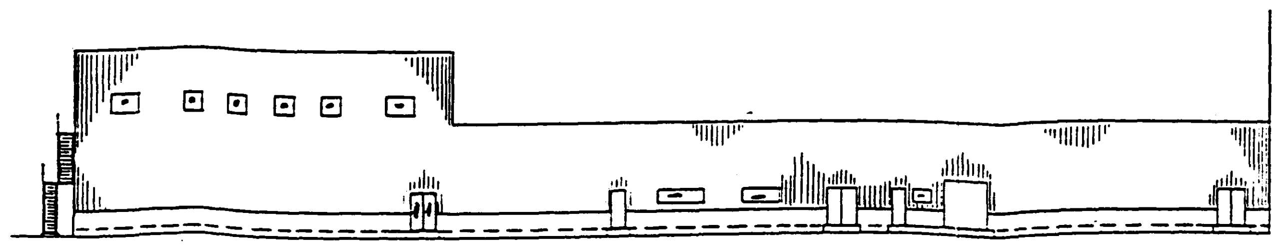
1. WAREHOUSE
2. WAREHOUSE STORAGE
3. WAREHOUSE OFFICE
4. CHIEF WAREHOUSEMAN
5. ELECTRICAL SHOP
6. FENCED STORAGE, ELECTRICAL
7. BRIDGE CRANE - 2 TON
8. ELECTRICAL SHOP OFFICE
9. MACHINE SHOP OFFICE
10. RUBBER SHOP
11. MACHINE SHOP
12. BRIDGE CRANE, 1 TON
13. BRIDGE CRANE, 5 TON
14. MILLWRIGHT SHOP
15. BRIDGECRANE, 5 TON
16. SAMPLE PREPARATION ROOM
17. BALANCE ROOM
18. ASSAYERS OFFICE
19. STORAGE
20. STORAGE
21. ASSAY ROOM
22. ATOMIC ABSORPTION ROOM
23. AMBULANCE GARAGE
24. FIRST AID WAITING ROOM
25. FIRST AID
26. CARPENTER SHOP
27. LUNCH ROOM
28. INSTRUCTION ROOM
29. LOCKER ROOM
30. WASHROOM
31. SHOWERS
32. JANITOR'S STORAGE
33. MAIN OFFICE ENTRY



GROUND FLOOR PLAN



END VIEW

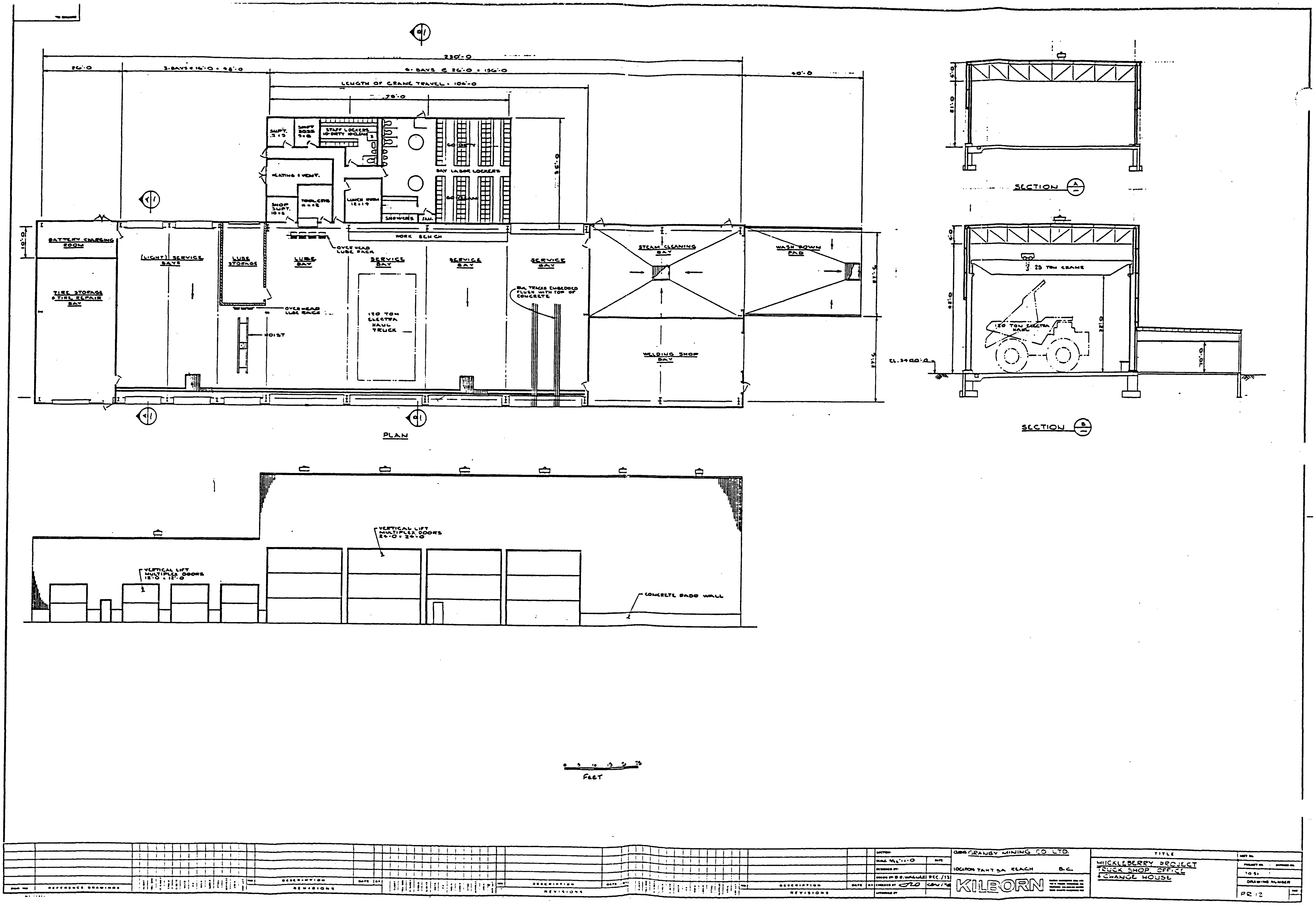


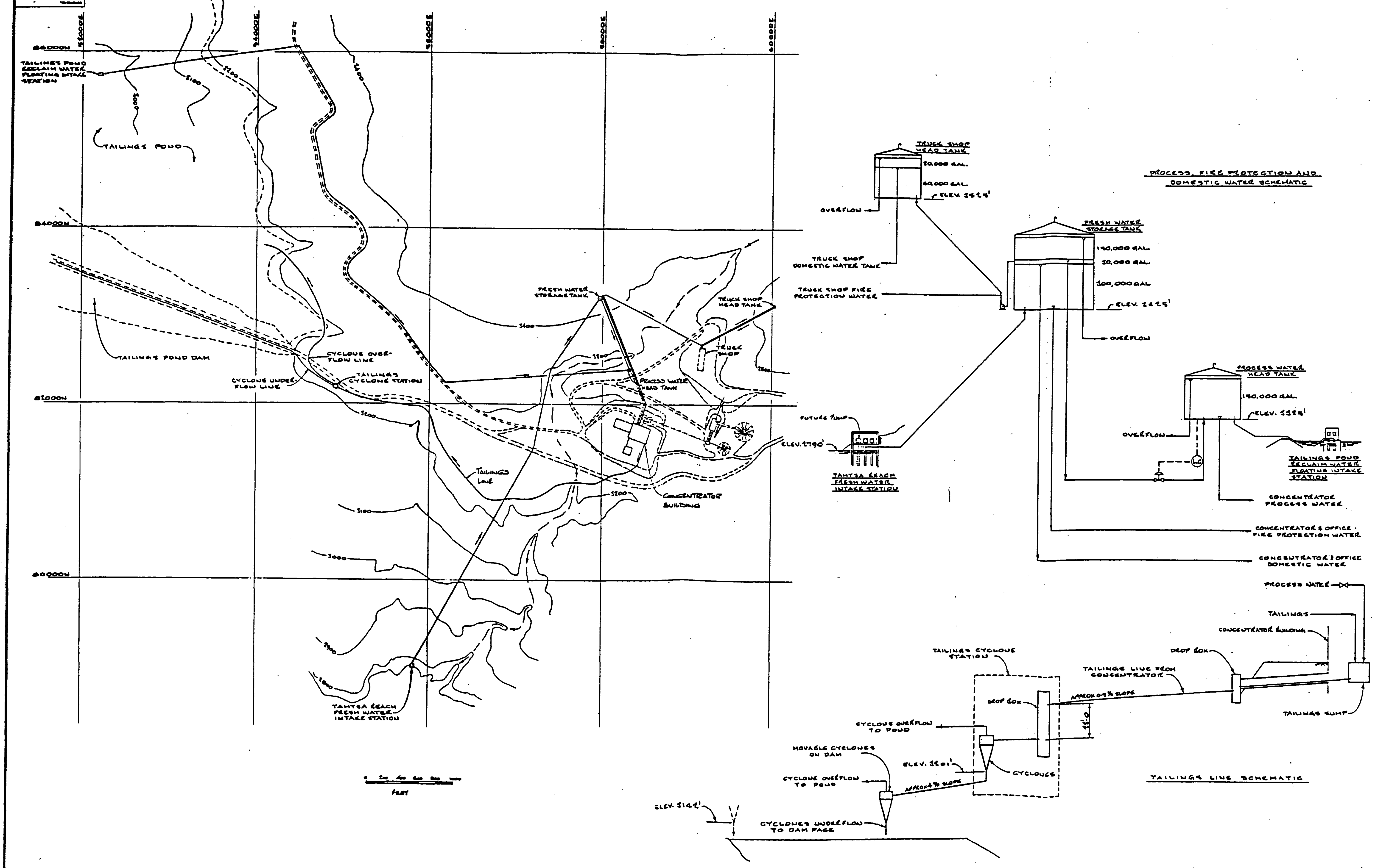
ELEVATION

FEET

SCALE	1/16" = 1'-0"
DRAWN BY	K.A. DEC/73
CHECKED BY	[illegible] JAN/74
CLIENT	GRANBY MINING Co. LTD
LOCATION	JANTSA REACH B.C.
	KILBORN
TITLE	HUCKLEBERRY PROJECT SERVICE & WAREHOUSE COMPLEX PLANS & ELEVATION
PROJECT No.	7051
DRAWING NUMBER	PR-11

DWG No. | REFERENCE DRAWINGS | DESCRIPTION | REVISIONS


230'-0
26'-0
3-BAYS @ 16'-0 = 48'-0
6-BAYS @ 26'-0 = 156'-0
40'-0
LENGTH OF CRANE TRAVEL = 104'-0
78'-0
SUPT.
STAFF LOCKERS
HEATING & VENT.
DAY LABOR LOCKERS
SHOP SUPT.
TOOL CRIB
LUNCH ROOM
SHOWERS
JAN.
WORK BENCH
BATTERY CHARGING ROOM
TIRE STORAGE & TIRE REPAIR BAY
(LIGHT) SERVICE BAYS
LUBE STORAGE
LUBE BAY
OVERHEAD LUBE RACK
SERVICE BAY
SERVICE BAY
SERVICE BAY
STEAM CLEANING BAY
WASH DOWN PAD
RAIL TRACKS EMBEDDED FLUSH WITH TOP OF CONCRETE
HOIST
120 TON ELECTRA HAUL TRUCK
WELDING SHOP BAY
PLAN
SECTION A
SECTION B
25 TON CRANE
120 TON ELECTRA HAUL
EL. 3400'-0
VERTICAL LIFT MULTIPLEX DOORS 24'-0 x 24'-0
VERTICAL LIFT MULTIPLEX DOORS 12'-0 x 12'-0
CONCRETE DADO WALL
FEET
REFERENCE DRAWINGS
DESCRIPTION
REVISIONS
CLIENT GRANBY MINING CO. LTD.
LOCATION TAHTSA REACH B.C.
SCALE 1/16" = 1'-0
DRAWN BY D.E. WAGNER DEC./73
KILBORN
TITLE
HUCKLEBERRY PROJECT
TRUCK SHOP, OFFICE & CHANGE HOUSE
7051
DRAWING NUMBER
PR 12



CLIENT: GRANBY MINING CO. LTD.

LOCATION: TAHTSA REACH, B.C.

SCALE: 1" = 400'

DRAWN BY: P.V. DEC/73

KILBORN

TITLE: HUCKLEBERRY PROJECT WATER AND TAILINGS DISPOSAL SYSTEMS LAYOUT AND SCHEMATICS

PROJECT NO. 7031

DRAWING NUMBER: PR13