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Tuesday Jan. 27, 1976

Please reply to:
Thomas Rd. Address.

Mr. George D. Ward, President,
Producers Minerals Corporation,
104 E. 40th St./New York, N. Y. 10016

Dear Mr. Ward:

I would apologize for delay, except that most of the material herein was made known to Mr. Morrow, and to you in my telephone call from Colorado Springs.

I am not the type who accepts a one day assignment as a consultant because of, regardless of experience, it is impossible to cover the client need and give him value for the dollar unless the visit is an after the fact type. In effect in my original letter to you I suggested a few days at site which was abrogated by your letter of Sept. 26th last.

Ethically speaking, and in terms of real benefit all you are getting is an opinion, admittedly with extensive background in your area of endeavor but, until down to the nitty gritty, so to speak, all you are really getting is loquacity and repetition.

I am not known for my acquiescence to, or impassiveness, involving the expenditure of several thousands of dollars whether as a consultant or in management for a corporation. My time spent has indeed been perhaps more rewarding to me perusing your report and having seen the site, because there are many hidden ideas and conclusions in my mind, without foundation because of the short visit. The study on mine planning, and the one on ore control may be of some benefit to the younger engineers whom you hire and was really the only time consuming facet of this report.

You have an admirable young man (at least by my standards) in Ralph Morrow and he should be given every opportunity to exercise his thinking, not only on paper but applied to corporate savings, in actual operation. If you feel the need for consultants in the future would suggest, for them to be of most value to PMC that they be chosen carefully as cost conscious, and at least be given time to absorb your problem areas.

CC/BCL/pt
file
R. Morrow.

Sincerely,

B. C. Lansing

1-27-76

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1.0 INTRODUCTION

This evaluation is sketchy, as only a one day visit to a property, compared to the detailed work done by PMC can be.

It would be remiss not to mention the complete cooperation of your Ralph Morrow and his dedication to this project. Many of the items herein are perhaps redundant because they were already discussed with Mr. Morrow, or had occurred to him prior to my visit on the scene.

General coverage

The survey and map work already done is above average in appearance and accuracy, including regional control etc. It is from this base that expansion, pit plans, long range plans, ore control land boundaries and overall coverage evolve, and to which a lack of such accuracy applied will multiply the errors, compounding the felony so to speak. It is refreshing to see such thoroughness.

Conclusions

The accompanying comments page for page on the "Safford Copper Project" report prepared in June 1975 are simply observations some of which are redundant and others of which have perhaps already been considered,implemented or discarded.

I have not commented on the second half of the report which refers to "Leaching of San Juan Ore" for the obvious lack of real study on a one day visit compared to your multiplicity of lab and field test work.

I must qualify this however having "accumulated doubts". No group or assemblage of groups, however skilled or scientific, will <u>ever</u> convince me that feed size of material to any leach operation is not perhaps the most encompassing factor toward ultimate return on the dollar.

This is the only area where I would use leaching experts, other than operational, with all the bacteria, black acids or others, and, a further turgescent emphasis on the scientific. This is necessary and in it's place is just as valuable as all other factors. I happen to be the exponent of practical thinking and applying everything to the dollar saved. A mining corporation trying to make a profit on a product be it uranium, copper, moly, etc. can only expend so much money comparing several theories and futuristic plans and still make a profit. When compared, all of the costs of production, and they are manifold in amount, can show areas of saving which far transcend the number of liters of this or that reagent, black or regular acid, etc. to produce a significant increase in recovery. Metallurgy is fine when all other recovery problems are solved and one must then turn to chemical application for the finite increases.

2.0 SUMMARY

1. Make a concentrated effort to capture and store run off water with dams, ponds etc., taking advantage of summer rains and run off.

2. Build these dams with leachable material sized and delivered by grade and chemical analysis, for exposure of the richest material to the most water.

3. As the rains show promise add new material and keep good product control on effluent.

4. Make a concentrated effort to supply new water such as Gila river rights, drill wells, acquire new rights if possible.

5. Completely overhaul production piping, recovery and volume physical systems, using the most up to date PVC, valves and all others recently developed for this phase of the copper leaching application.

6. Establish, by example a future personnel program with someone who knows the business to obtain, pipefitters, mechanics, electricians, dozer operators etc., along with some mine planning young types who have some imagination and can be trained.

7. Using proper screening, with a relatively small operation and the already personable attitude of Mr. Morrow you can hire people skilled in several phases, afford to pay them more money and preclude Union problems. As an example, for your type of operation there is no reason why you should not hire an operator skilled in Dozers, trucks, front end loaders, and even drills. His flexibility to work where management desires is worth more money to him, but what is more important, altho wages may be higher than the going rate such a crew overall will bring greater savings to the parent company. Without a man like Morrow and his attitude toward people I would not even suggest such a program, and you also have an area of job paucity in your favor. This paucity will not last long however when, (and not if) the price of copper increases to a just position in the world market.

Summarily then, an intriguing operation beset with some problems, none insurmountable or if they are it has not been drawn to my attention.

PRODUCERS MINERALS CORPORATION

SAFFORD COPPER PROJECT

The bulk of the assaying done by atomic absorbtion. Presumably when the mining of Peacock began there was only a small ore body delineated in scanty drill hole information, until later drilling and more accurate data obtained.

As drill programs progressed the less dependable was phased out and ultimate drilling from which the pit data into 1971 was applied determined the reserves. This was data provided almost entirely by PMC and to a lesser extent by Scruggs.

At depth there is an indicated increase in chalcocite and chalcopyrite, which along with evidence of surrounding deep activity could lead to a major ore body at depth.

History:

This proves the strategic geographical location of the PMC claims but adversely, suggests that you will be under duress by the surrounding majors with pressure to, a) sell out b) give up c) or eke some small support from the present owners who are being pressured by the past to suborn their agreement. The extent to which the owners back you and negate outside pressures will perhaps depend upon their feelings that PMC is sincere in their ultimate major potential demonstrated by increased capital and production but certainly only after proper testing and comparison of alternate methods to get "best value for the dollar .

As far as I am personally concerned, (and I do know something of the history of that general area from years back) Guy Anderson was, if deceased, or is a "crook of the first water." Any legacy he may have left would to my mind not be in the best interests of PMC.

As your experience with this group has progressed am reasonably sure that you agree with what I have said and it is certainly nothing new to your thinking, as witness the successful outcome of legal proceedings at least as of this moment.

It would be naive or showing lack of informed judgement, were I not to recognize that you are abreast of such goings on and very capable of handling same. Nonetheless, some of the past history of that area suggests the necessity of a better than average acuity.

Evidently much, or all, of the production beyond 1971 with some new material broken up to 1976 has been leaching material already mined pending outcome of lawsuits on the "black" acid etc.

So perhaps one does not get the same recovery with this acid as opposed to the pure stuff. Testimony of theoretical and farseeing scientists of one form or another may assist in the process but they do not put money in the bank, which is the name of the game.

This gets back to people like Essex, Eimon, and others involved in your litigation, and the court decision in your favor shows along with discharge of Eimon, his inadequacy not only where the parent is concerned but in his approach. For that matter, you may be presently working with him in his new capacity, none of my affair but something which seems doubtful to me.

You will of course always be plagued with those looking for personal aggrandizement, the do-gooders so to speak such as the league of women voters, Sierra Club, friends of the owners, and their relatives ad-infinitum.

Reserves and Ore Type

Computer Associates is a fine firm and no doubt dedicated. It has been my experience however, and I mean experience that the best way to satisfy yourself on the subject of reserves and mining plans for any feasibility study is to work it out by polygons on a bench to bench basis, with whatever geologic interpretation can be of benefit.

In this interest I have enclosed in the Appendix a general outline for pit design, as well as ore control, which will apply to almost any orebody. This is not new to those in the field but it may be of some value to the young people you will have to engage if you are to continue with this venture. As mentioned in the report the design engineer spends all of his time running new sections, new pit slopes, and applying all new ideas, constantly, to come up with the most economical bench height, pit slope, hole influence, etc.

If chrysocolla is prevailing it is subject to less recovery than malachite or other oxides, which may be a recommendation for at least a better than lab type comparison with leaching minus 3/8" as opposed to run of mine.

Deep drilling costs money and my suggestion is that you can get your house in order by first concentrating on recovering the surface material with minimal cost and ultimate profit.

I agree that deep drilling is warranted based upon results of surrounding property by the majors, (this is not to imply that PMC is not a major but again not relevant to the fact that one way or another if such a program is to be implemented the tremendous capital involved must be emphasized.), whether by the parent or on a partnership basis.

At the time of my visit seven leach areas were laid out with some precision for a total of some 112,000 tons which should be fairly representative, running 0.485 % copper = 543.6 tons in place.

This in itself suggests spending capital on a pilot plant operation as opposed to a lab operation for some definition on recovery relative to grind and product feed. Indeed one must remember that regardless of the price of copper, at a mine feed of 0.60% Cu. and the present market price of $0.63/lb. a one percent increase in recovery allowing for losses plus comparison with actual drill hole assays from 70% to 71% = $0.076/ton. This pays for a lot of research with something left over based upon the potential for PMC.

Checking the reserves based upon apparent recent calculations, of some 15.6M tons in the main ore body and 2.2M in the main extension, using a 1.05:1 strip ratio for the main ore body and 2.39:1 for the extension using a 0.35% cut-off, these reserves in themselves, especially based upon future copper price, should indicate detailed capital studies concerning leaching, milling, or a combination of both.

Leachability of oxide ores

Leaching, an old established process, on broken ore in place was begun in the 1920 period at Bingham Canyon and naturally has resulted in some improvement since. It is well known that a) oxygen, bacteria, temperature etc., had a major effect, b) small cost per pound of production, c) process is relatively simple but success depends upon: 1) Non-acid consuming character such as siliceous limestone as opposed to acid consuming unsiliceous limestone, 2) impervious layer under dumps, 3) permeability of the mineral zone and the ability to form sulphuric, and ferric Sulphate, depending upon the pyrite content. The ferric sulphate is an active solvent for copper in chalcocite.

The availability of adequate water with proper salt content at relatively low cost without appreciable evaporative loss (the use of the rain bird system in an area of such evaporation relevant to Safford causes more than average evaporative loss)

Finally, and perhaps most important, the distribution of the easily soluble oxides, such as malachite, cuprite, azurite, are placed to allow direct contact, soon by leaching solutions.

My impression is that you have given less emphasis to comparison of the finer feed product opposed to run of mine, and conclusions were drawn because of the depressed copper price and desire not to expend additional capital. If you are looking into the long term in situ leaching such as waste dumps from the larger open pit copper operations that is well and good. On the other hand for early recovery and profit, for which every copper company is searching today, and the deep potential more attention should be given to early recovery with a good balance sheet and showing some confidence in the future price of copper, which must be at least $0.80-0.90/lb by late 1977. This type of attitude would also be beneficial in the event you are looking for a partnership deal whether with banks or other companies in the business.

Water Supply

You are presently using water from Phelps Dodge, which is a nebulous thing because in the event they decide to go into production, which seems rather imminent because of their continuing work, they will require their own water. With the already rather substantive reserves by any standards to what extent have you made attempts at laying plans for a) using all possible natural flow areas for runoff in the rainy season, and b) estimated costs and possibility for bringing water from the Gila River rights? c) have you drilled any wells on your ground or are you contemplating same, to insure future provision of water whether by milling or leaching?

Lease Terms

The lease terms seem not unreasonable where the owners retain some some 25% net profit in the operation, especially considering this is after income tax assessment. However, the minimum monthly payment of some $1500/Mo. at no profit with operations continuing at expenses entirely the burden of PMC seems incongruous to me. You have mentioned the possibility of a modified lease, showing your concern and this may already be taken care of, along with other doubts in your own mind.

The assignment of the lease from Scruggs equal to 1-1-1/2 % of the net sale price may be to your liking, and in order as long as the net sales price discounts all production freight and indirect and general costs etc., and is not a part of some long term agreement on a minimum price.

Copper sales contract

During my tenure in England for two years and four months on the Sar Cheshmeh Project in Iran, was closely associated with Selection Trust and their Engineering Subsidiary which did some work for Southwire. For what it is worth Selection Trust is very interested in obtaining footholds in North America, which has been evidenced to me on other studies beside Iran.

If PMC is considering some joint venture type operation certainly ST would be worth talking with.

Mining Plan For Surface Oxides

1. Phase 1 It would be interesting to see a comparison between in place leaching as opposed to selected heap leaching of the crushed ores. I find it hard to accept that inplace leaching can compete with some form of crushing and pond type control in spite of the crushing costs, for an overall profit.

2. Phase 2

   Hauling the ore from the pit and placed in designated areas for leaching whether crushed or not, is good from the water control and volume control basis to determine the actual recovery in terms of grade, size and time. The old leach areas (discussed with Mr. Morrow)would seem to provide excellent impervious layers with natural flow of solution to a gathering pond near the present plant.

3. Phases 3 & 4

   These are natural evolution from plans 1 & 2, but I would certainly recommend that at least a portion of one particular mining area be reduced to plus or minus 3/8" secure in the belief that with reserves in hand, serious consideration of all factors such as recovery, capital cost, best use of water, etc. would swing the pendulum toward some crushing, or even some crushing and a small mill type operation.

You will probably eventually make the decision to drill further perimeter holes, perhaps P-36, P-55, P-56, P-59, P-62, P-39, P-63, and P-64 all drilled in the order of best geologic interest for most rapid information, based upon your already quite properly prepared data.

It would be, and has probably already been done, very easy to make an up to date comparison using your now already established drilling costs, water costs, blasting, assaying etc to insure some proper estimate of relative recovery procedures, and certainly bench planning and ore control play a major part.

The design leaching curve shows in the more refractory monzonite that the time in the field for a certain recovery compared to that in the laboratory is the same for both crushed and uncrushed.

You have no doubt conducted many tests to ascertain this assumption in calculations, and as mentioned are continuing with tests. What I am perhaps implying, based upon a complete paucity of information and time on the property, compared to your quite extensive research and practical studies, is that there does not seem to be any evidence of real ore control procedures which are paramount.

Computerized pit plans are fine but they are only general and for estimating purposes only. Such studies can only be used as aids to proper pit design, bench heights, grade control, ore waste ratios, pit slopes etc. You have, already prepared a beautiful set of maps, and assemblage of information.

I have read with diligence the rough outline of pit boundaries, projected economic results, leaching etc, along with capital projections including leasing equipment, and evidence that your plans were to make detailed design and equipment selections in late 1975.

If this has been or is being done you certainly are on the right track and properly managed, I cannot foresee anything but a good potential and profit for PMC.

Your geographical position, coupled with progress as envisaged in this report indicate a certain command of the area, and a strong bargaining position in what ever direction best suits corporate policy, if a joint venture is contemplated.

If the copper mining industry is to survive there is little question in my own mind that the price of copper will be well beyond $0.70 - 75/lb by mid or late 1977. This somewhat influences the importance of money spent now to reap the rewards later, if corporate policy and finances permit.

The second half of the report "LEACHING THE SAN JUAN ORE IN SAFFORD, ARIZONA" is very thoroughly done and reads from test data that crushing the ore, certainly has a very beneficial effect both inrecovery, and length of time for return.

Certainly I am not in a position to criticize after my short visit on all the work done, but still do suggest that having done in place leaching with no crushing long enough to obtain actual costs, has this information been compared with other methods at todays prices, to insure the very best approach for PMC ?

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DEVELOPMENT OF AN OPEN PIT LIMIT AND CALCULATION OF ORE RESERVES

DEVELOPMENT OF AN OPEN PIT LIMIT AND CALCULATION OF ORE RESERVES

The design of an open pit and the calculation of ore reserves within that pit using polygons as a tool is a step-by-step procedure. The factors that must be considered in planning an open pit are numerous and must take into account the geological characteristics of the orebody. This study is directed towards the design of an open pit copper mine, but the procedures are applicable to about any type of metallic or nonmetallic orebody.

The first step is to locate the drill holes through surveying procedures and plot them on a topographic map.

> The scale of the map should be carefully considered. It is recommended that the overall situation be examined so that the map scale is small enough to cover the immediate pit area plus possible waste dump areas and yet large enough to show adequate details such as 100- or 150-foot haulage roads which may be plotted later when mine planning becomes necessary. Often 1 inch = 100 feet is somewhat too large a scale to cover the needed geographical area and therefore 1 inch = 200 feet is used. The extent of the orebody is one of the primary determining factors.

A scale of 1 inch = 100 feet was used for the accompanying model because just the pit outline and the ore reserves were considered. A topographic map was drawn and the drill holes plotted with appropriate letter and/or numerical designation. Each hole must be identified.

The grade cutoff must be determined. The cutoff grade is determined when the net value per ton of ore (taking into account recovery at the mill and market value) equals the total cost of handling. Cost estimates and market value estimates are required. The cutoff for the model orebody was set at 0.60% copper. All material with 0.60% copper or above was considered ore, and material with less than 0.60% copper was waste.

Another determination is what bench height to use. The individual drill hole assays (5- to 10-foot samples) are usually combined (weighted by sample length) and averaged into some longer unit. These combined assays must be directly related to elevation above sea level or some basic reference plane in order to correlate assays from one drill hole with assays from another drill hole.

The planning engineer considers several different possible bank heights. The ore characteristics and the equipment to be used are both considered. These two factors usually work against each other. Present available equipment usually operates at lower cost with higher bank heights. There is no question that lower bank heights allow greater selectivity of ore resulting in higher average grades for the orebodies. Most pits use between 35- and 50-foot bank heights. The planning engineer experimentally varies the drill hole assay combination intervals both with respect to bank height and with respect to the reference plane.

In the model used in this study, the individual assays (not reported) were combined into 40-foot composites using elevations above sea level for the reference base. The overall grade of the model orebody was analyzed using a standard 40-foot composite assay and reference elevations of 5220, 5180, 5140, and 5100 and compared to the 5240-, 5200-, 5160-, and 5120-foot elevations and further analyzed using 35-foot composite assays. The final elevations used were 5240-, 5200-, 5160-, and 5120-feet above sea level. These elevations represented the elevation of the pit benches and horizontal sections were drawn on these reference planes.

After selection of the desired bank height, horizontal sections were constructed at the optimum bench elevations (see in the model maps with benches--5240, 5200, 5160, and 5120).

The best procedure in construction of these maps is to use transparent paper that can be used as an overlay and to reproduce prints upon which most of the construction work should take place. Only items that are not changeable are plotted on the original. These include the grid, title block, drill hole intercepts, and the precalculated assays.

The first step in constructing these sections is to set up a grid or reference system. Often the grid originates with 0 North and 0 East at the same point in the lower left-hand corner (illustrated in the model), preferably located well out of the area to be actively studied. From this point of origin, the grid is extended north and east. The grid designations refer to the distance or footage from the zero point. The grid is applied to the topography map and to all bench maps, and vertical sections. The grid enables the engineer to overlay the bench maps one on another and follow ore and waste trends through each bench up to and including the surface map.

Those holes which pass completely through the 40-foot bench are drawn on the section. If a hole enters the 40-foot bench interval, but does not pass through the section, it may also be plotted, but care is taken to note that it does not represent a 40-foot assay. When the time arrives to polygon the drill holes, this odd hole is considered separately and a decision made as to whether or not the geology allows whatever footage of drill hole entered the bench to represent the full 40 feet of material. Usually the footage of the hole within the bench determines whether it can be allowed or not. For example, 20 or more feet of hole probably is allowable, and less than 20 feet of hole is not allowable for 40-foot benches. If the decision is to allow it, then it is treated as any other hole; if the decision does not allow it, then the hole is ignored.

After construction of the horizontal bench sections which include the drill hole intercepts, drill hole designation, grid, title block, and the surface intercept if there is one (determined by overlaying on the topography map), the allowable horizontal influence of the composite drill hole assay is determined.

There are various procedures which can be used to determine the influence of a drill hole. Geology is always considered, particularly structural control and/or material type control. The ore may be cut off at a particular fault line or it may be confined to a particular rock type or bedding plane. These factors control any procedure for determination of assay influence and override any other process or procedure. In the model none of these geologic conditions exist and the rock is considered to be homogenous.

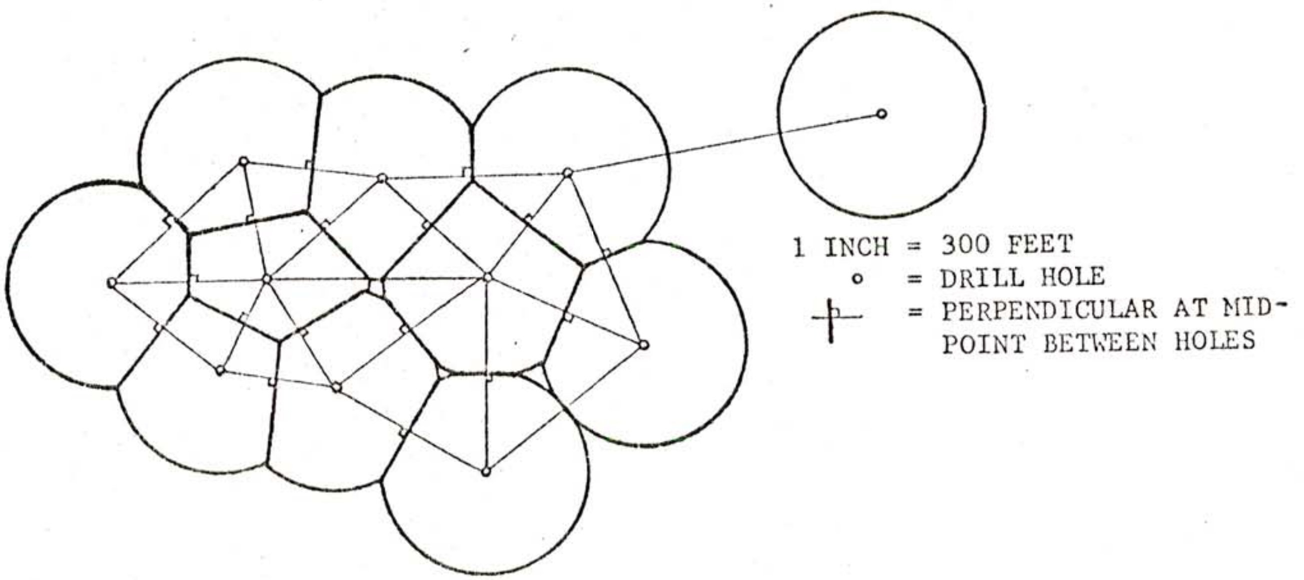
Polygon construction was used with limits set on the polygon procedure. The polygons were constructed on the horizontal bench sections. An orebody generally drilled on

200-foot centers indicates that the geologist or exploration people consider the influence of drill hole assay values of 100- to 150-feet in one direction valid. Therefore, the limitation placed on the polygons constructed in the model was that at no point must the polygon exceed 150 feet from the drill hole. The influence of the drill hole assay must not exceed this footage. This is a fairly common limit in copper and molybdenum pits; however, the distance is predetermined by the people who set the original drill hole spacing.

Determination of polygons was a matter of construction on the horizontal bench sections. A line was drawn between each hole and adjacent holes. The half-way point on each line was determined by measuring the distance between holes. To be a valid point, the distance between the two holes was not nor did not exceed 300 feet because of the 150-foot limitation.

If this test proved the point to be valid, a perpendicular line was drawn through the center point and extended in both directions. The procedure was repeated between all adjacent holes. The perpendicular lines intersected each other forming the sides and corners of a polygon as illustrated on the model horizontal bench sections. Care was taken to insure the lines were perpendicular to the original line drawn between the two holes. Two right angle triangles were used. Use of valid points resulted in polygon corners within the 150-foot distance from the drill hole limitation.

If the line between two adjacent holes measured 300 feet or more, an open-ended polygon resulted. A compass was used to draw an arc to close the polygon in the direction where no holes existed or when the holes in that direction exceeded 300 feet. A radius of 150 feet was used.



TYPICAL POLYGON PROCEDURE

When all polygons are drawn on the bench sectional maps, the planning engineer acquaints himself with the ore and waste polygons on each bench. The horizontal bench sections with the corresponding ore and waste polygons should become as familiar to him as "his right hand." He makes mental notes on location and amount of ore, location of high-grade ore, waste distribution, particularly with regards to ore location; and he puts together a three dimensional picture of the situation in his mind. Areas where ore is closest to the surface (small amounts of waste are directly above) are noted and vice versa, areas where ore with abundant waste above are noted. How well the planning engineer knows the orebody greatly influences the amount of work remaining. Previous experience in pit design helps to reduce the work load at this point, but is not necessary to finalize the study.

After the planning engineer was satisfied that he "knew" the orebody, a generalized pit outline enclosing the orebody was drawn. Usually most open pits close in circular configuration. The line was constructed on a transparent worksheet which was overlain on various benches to aid in estimating the pit outline. Only one generalized line was necessary, and it was drawn on a map with the grid system previously established.

A series of vertical radial sections were prepared. In the model, vertical section lines were drawn on the topography map. Each line represented a vertical section drawn through the proposed pit area at right angles to the pit configuration line in an attempt to show a true slope angle. This process was a repeat process; therefore, a few primary vertical sections were drawn first. As the process was repeated, more sections were added and the work refined. The use of vertical sections located radially around a pit improved the visualization of the orebody and was used to approximate the fixing of the pit limits.

The vertical sections showed drill holes, intersection of the grids, the polygon intersections, the surface intercept, and pertinent geological features. The vertical sections included in the model were colored to dramatize the ore-waste relationship. The information on the horizontal bench sections was transferred to the vertical sections.

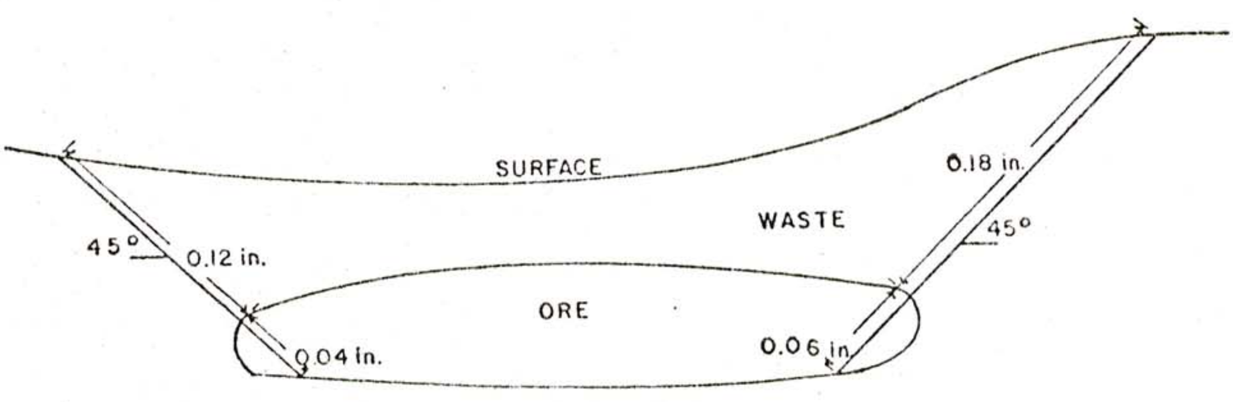
Before the pit limit lines were drawn on the sections, a break-even strip ratio was calculated and a decision on rock stability with regards to final pit slope was made. A typical 45-degree slope was utilized in the model study. A steeper slope had advantages; however, 45 degrees is a fairly common slope angle. The formula used to determine the break-even stripping ratio was:

$$\text{Break-even Stripping Ratio} = \frac{\text{Recoverable value/ton ore} - \text{Production cost/ton ore}}{\text{Stripping cost/ton waste}}$$

This ratio is applied only to the final pit limits and must not be confused with the overall ratio which is always less in order that a profit be realized. In the model a stripping ratio of one ton of ore to three tons of waste was used. Then slope lines (in blue color on the model vertical sections) were drawn. The slope was

45 degrees and dipped inward towards the center of the pit.

The locations of the 45-degree lines were determined by trial and error. The purpose was to enclose as much ore as possible and exclude as much waste as possible. Measurements along the 45-degree line must show the relationship of the ore-waste break-even stripping ratio. For instance, in the model the one ton of ore to three tons of waste strip ratio required that the ore along the 45-degree line was not less than 1/3 of the total length of the line. In other words, the approximate pit limit was defined as that line on the vertical section which had a 45-degree slope and was located where the break-even stripping ratio occurs (in the model three lengths of waste to one length of ore).



TYPICAL SECTION

After determining the pit limit lines using the vertical sections, the location of the pit limit was transferred to a transparent horizontal map. This was accomplished by locating (using the grid lines as a reference) a point (commonly called the toe of the bench) on the 45-degree line and transferring these toe points to the horizontal map. After plotting and identifying these points, the transparent map was laid on each horizontal bench plan and a pit limit drawn for each bench using the transferred toe points as guides. Care was taken to not close the bench lines within the designated slope desired or, in other words, the lines were not drawn closer than 40 feet for 40-foot bench heights using a 45-degree slope. The lines were at times more than 40 feet apart to accommodate the elimination of waste as long as it did not eliminate ore on a bench below.

The new pit had a different configuration than that first estimated, and some of the vertical sections were no longer at right angles to the pit sides. Therefore, new vertical sections were constructed at right angles to the new pit, and the whole process repeated. The method was repeated again and again until the planning engineer was satisfied that the results show an optimum economic pit. The final test was the calculation of ore reserves.

The transparent final pit outline map was superimposed on each horizontal bench map and the polygons measured with a planimeter which can be purchased from Los Angeles Scientific Instrument Company, Los Angeles, California. Waste and ore polygons inside the pit limits were measured and tabulated along with the ore assays. The procedure was to measure each polygon in terms of square inches. The work was checked by measuring all polygons on a bench and adding the square inches to compare against a planimeter measurement of the entire bench as outlined by the pit limit line. In the model calculated ore reserves were:

Polygon Measurements

Bench	Total Square Inches	Ore (Square Inches)	Grade (% Cu)	Waste (Square Inches)
5240	0.22	-	-	0.22
5200	66.52	25.97	0.75	40.55
5160	118.58	86.73	0.85	31.85
5120	86.33	72.19	0.82	14.14
Total	271.65	184.89		86.76

The 5240- and 5200-bench polygon measurements were adjusted because the average bench heights were two feet and 20 feet, respectively. They were less than 40 feet because of the surface configuration. The above square inches for these two benches were calculated by multiplying the actual measurements by 2/40 and 20/40, respectively.

Stripping ratio = 86.76 units of waste/184.89 units of ore = 0.47/1

Pit average grade of ore (weighted average) =

Bench	Tons	x	Grade
5200	25.97	x	0.75
5160	86.73	x	0.85
5120	72.19	x	0.82
	184.89	@	0.824

The bench weighted average grade illustrated above under "Polygon Measurements" was calculated in the same manner as the pit weighted average grade. Individual polygon measurements and grades were used instead of the bench measurements and grade.

A tonnage factor for a square inch and a 40-foot bank height was calculated and applied to the measurements above. Using an assumed 2.12 tons/cubic yard and realizing that the above measurements were taken from the model maps with a 1 inch = 100 feet scale, the tonnage factor was calculated:

1 square inch = 100 feet x 100 feet x 40-foot high bench
= 400,000 cubic feet

400,000 cubic feet/27 cubic feet per cubic yard
= 14,815 cubic yards

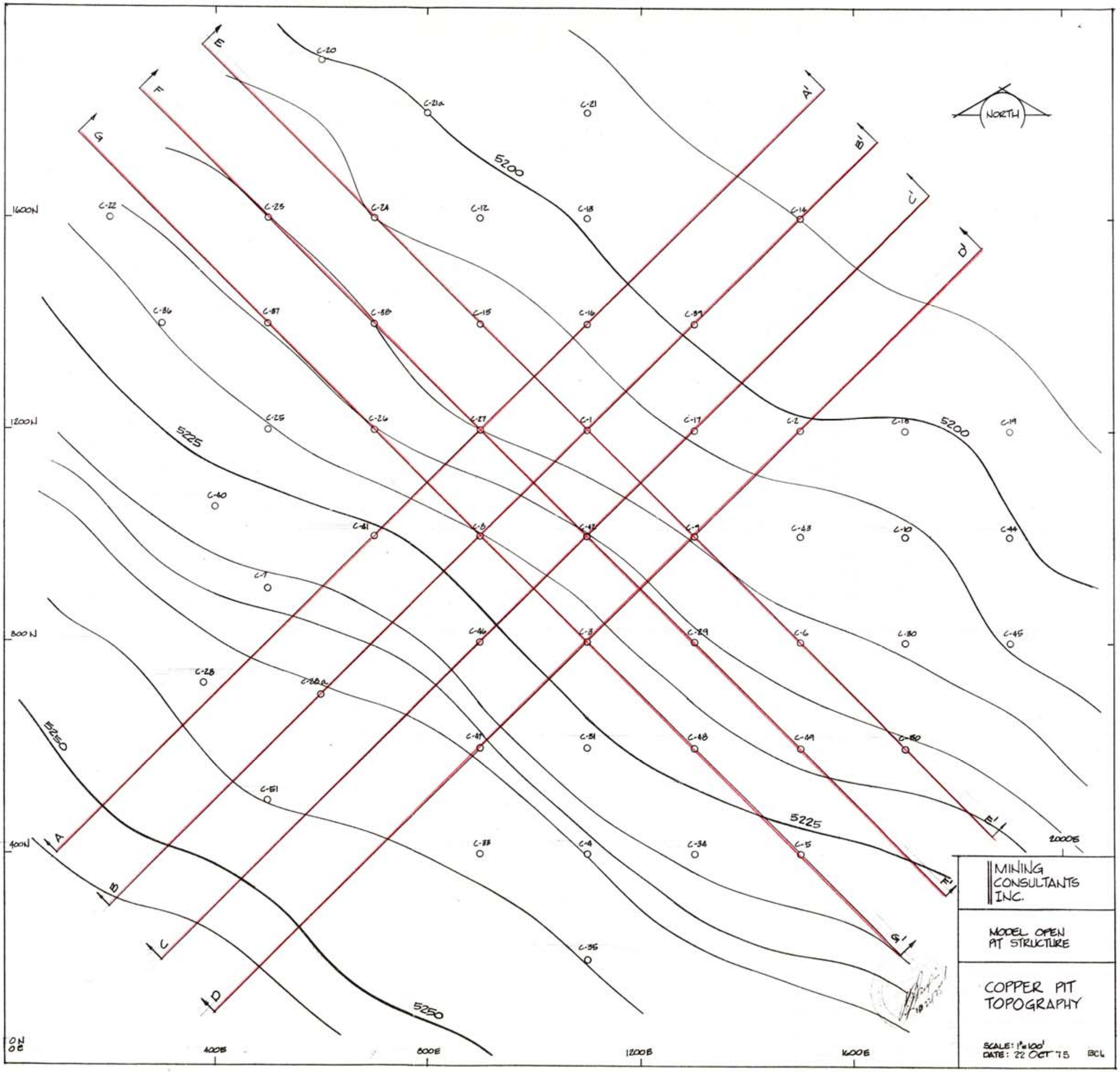
14,815 cubic yards x 2.12 tons per cubic yard
= 31,408 tons

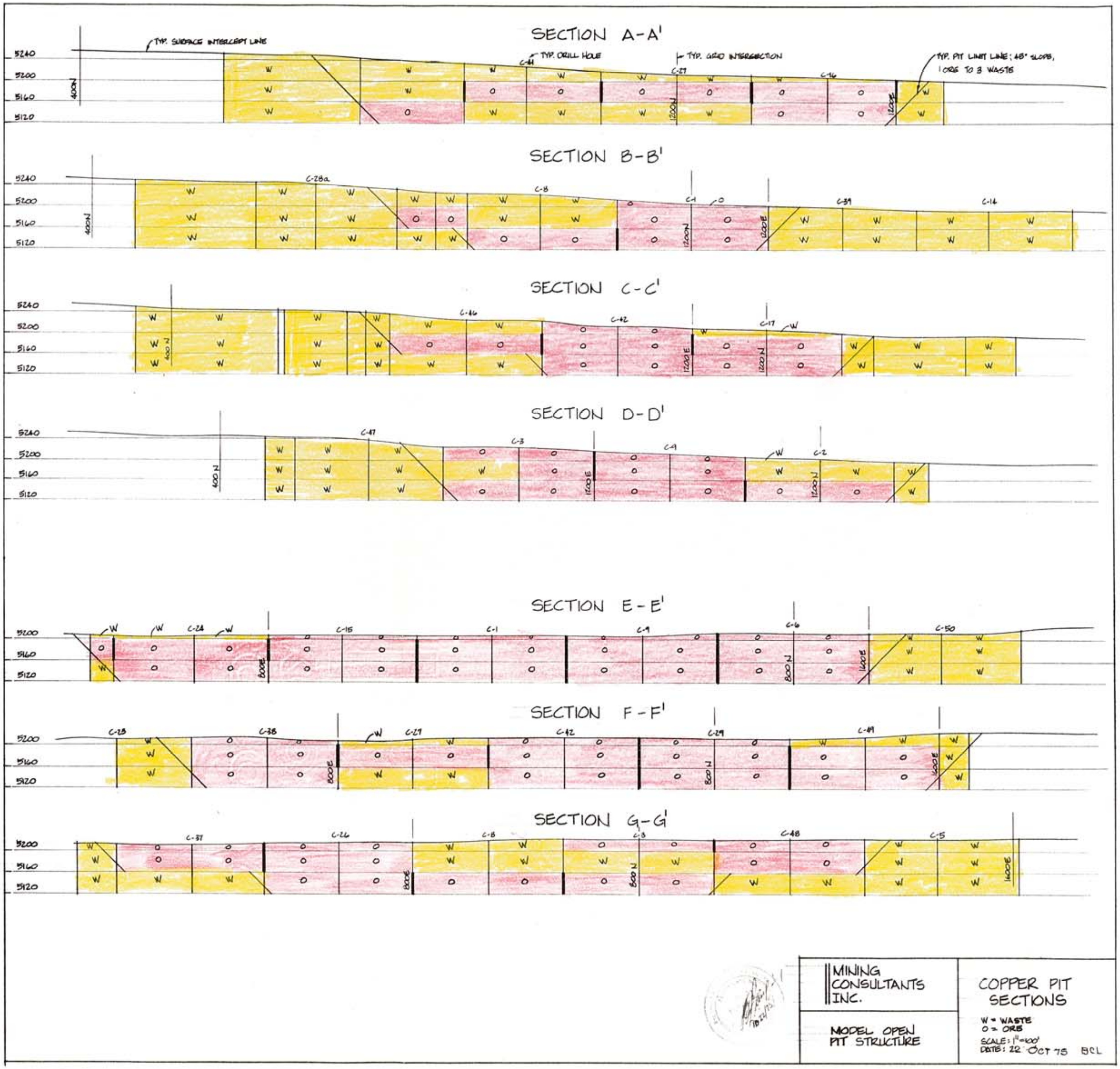
therefore

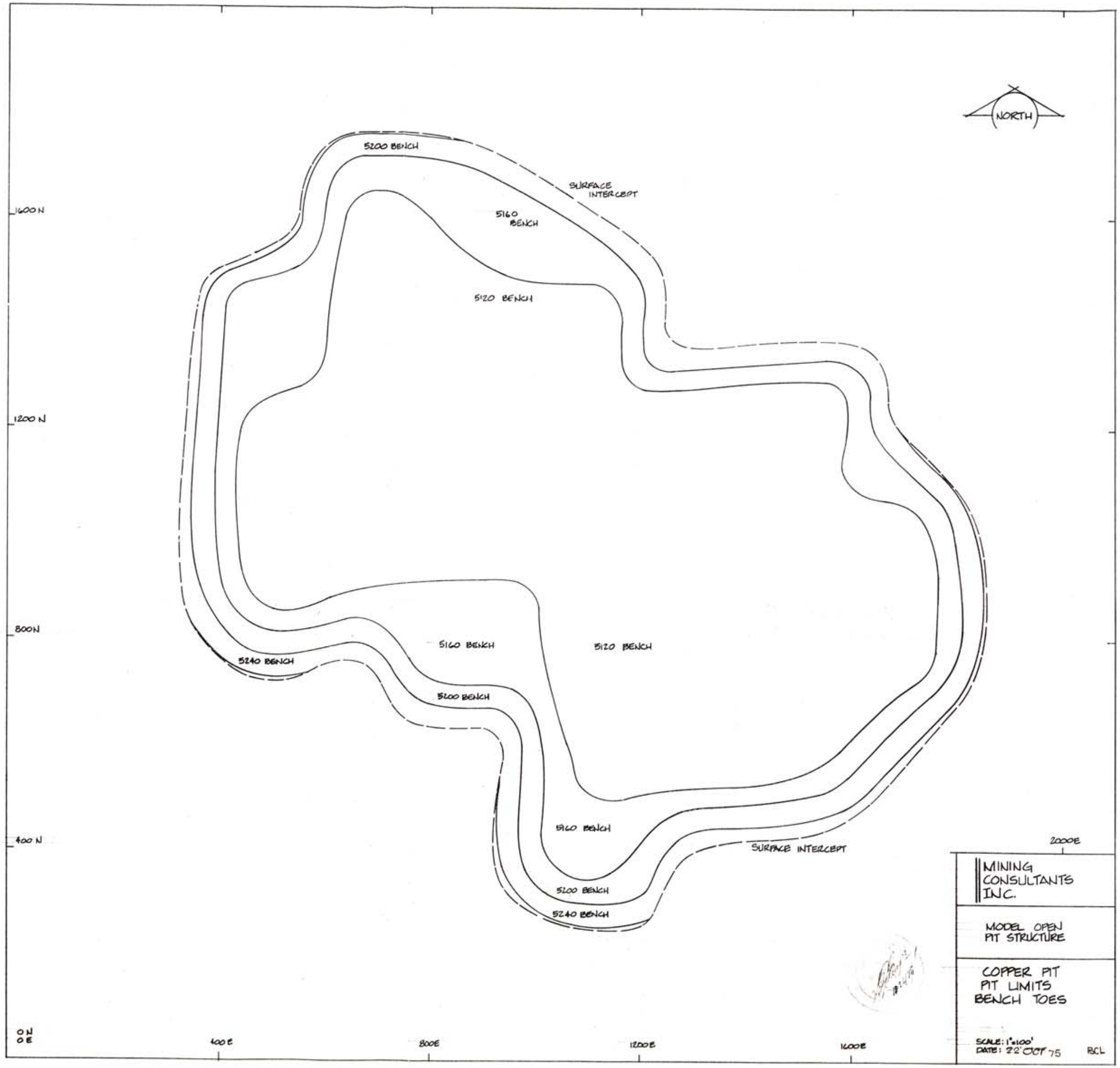
1 square inch represents 31,408 tons of material and multiplying this factor times the above square inches results in pit tonnages of:

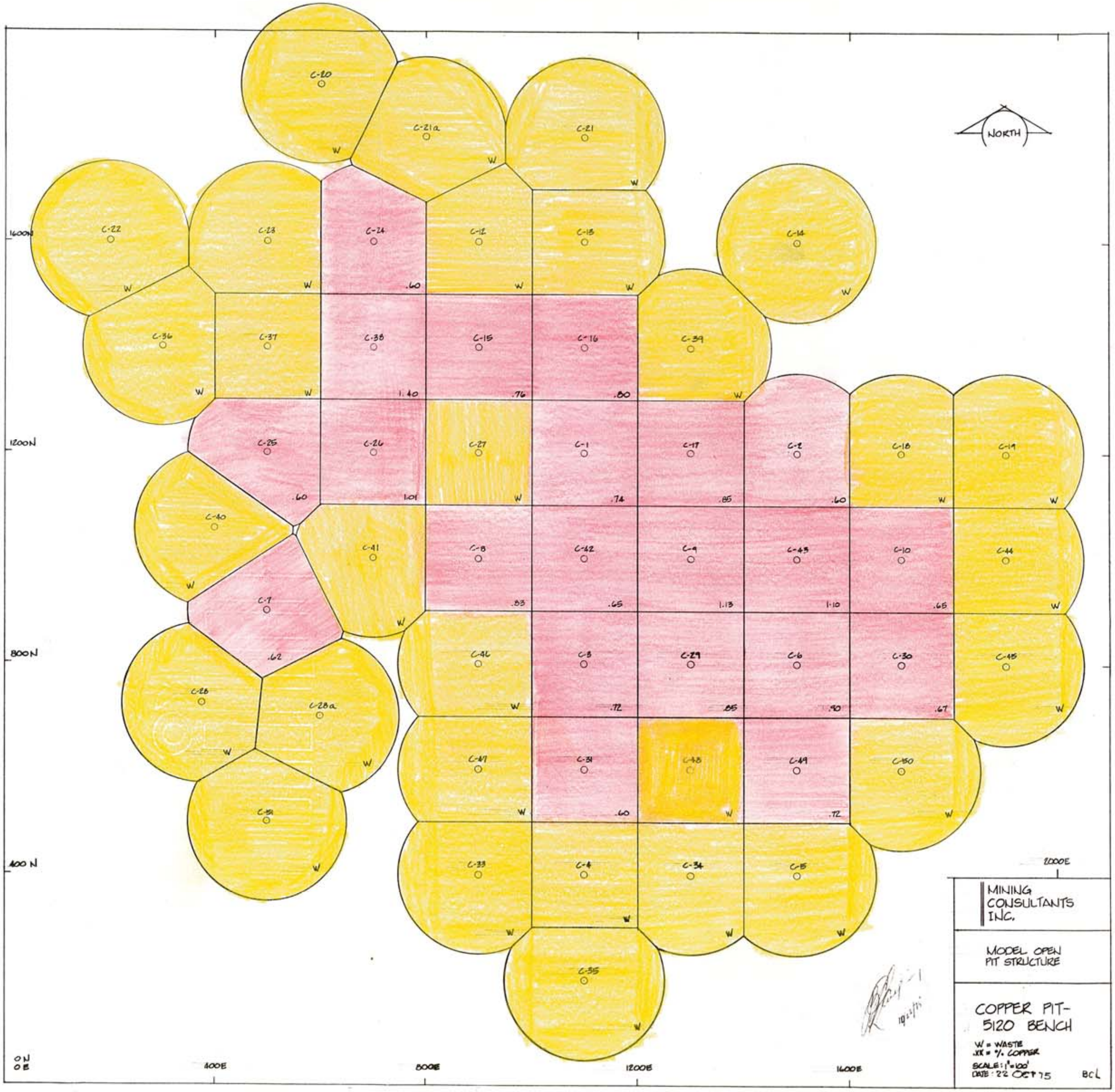
Bench	Total Tons	Tons of Ore	Grade (% Cu)	Tons of Waste
5240	6,910			6,910
5200	2,089,260	815,666	0.75	1,273,594
5160	3,724,361	2,724,016	0.85	1,000,345
5120	2,711,453	2,267,344	0.82	444,109
Total	8,531,984	5,807,026	0.824	2,724,958

The grade and the tons were for the 0.60% copper cutoff grade. The tons of ore and tons of waste figures would change if the cutoff grade is changed. It is possible to change the cutoff grade and calculate new tonnages for this pit. Of course, when the ore and waste tonnages change, so does the stripping ratio. The proper way to calculate ore reserves using a different cutoff grade is to redesign the complete pit using the new criteria.

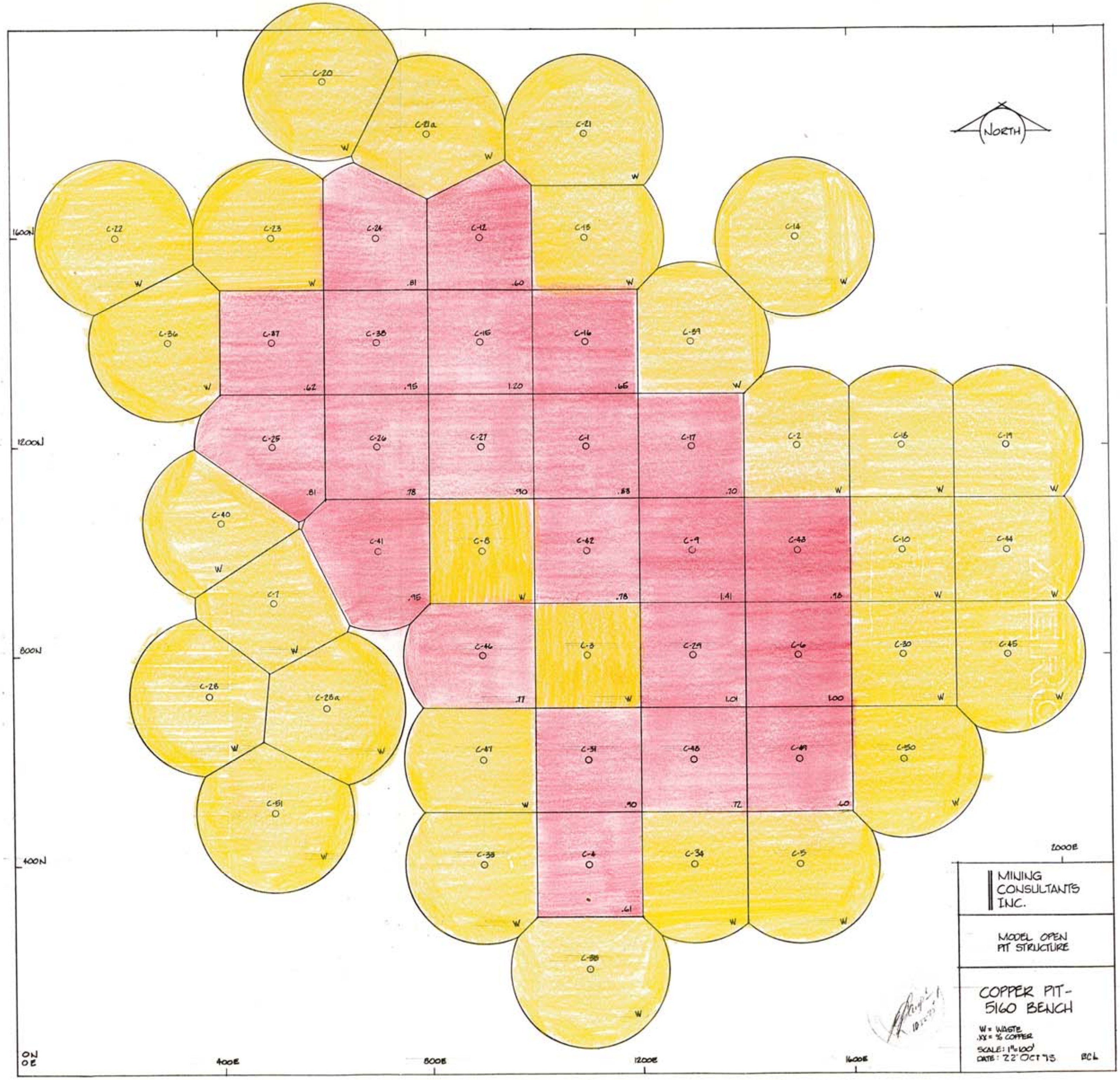

NORTH
1600N
1200N
800 N
400N
0N 0E
400E
800E
1200E
1600E
2000E
5200
5225
5250
A
A'
B
B'
C
C'
D
D'
E
E'
F
F'
G
G'
MINING CONSULTANTS INC.
MODEL OPEN PIT STRUCTURE
COPPER PIT TOPOGRAPHY
SCALE: 1"=100'
DATE: 22 OCT 75
BCL

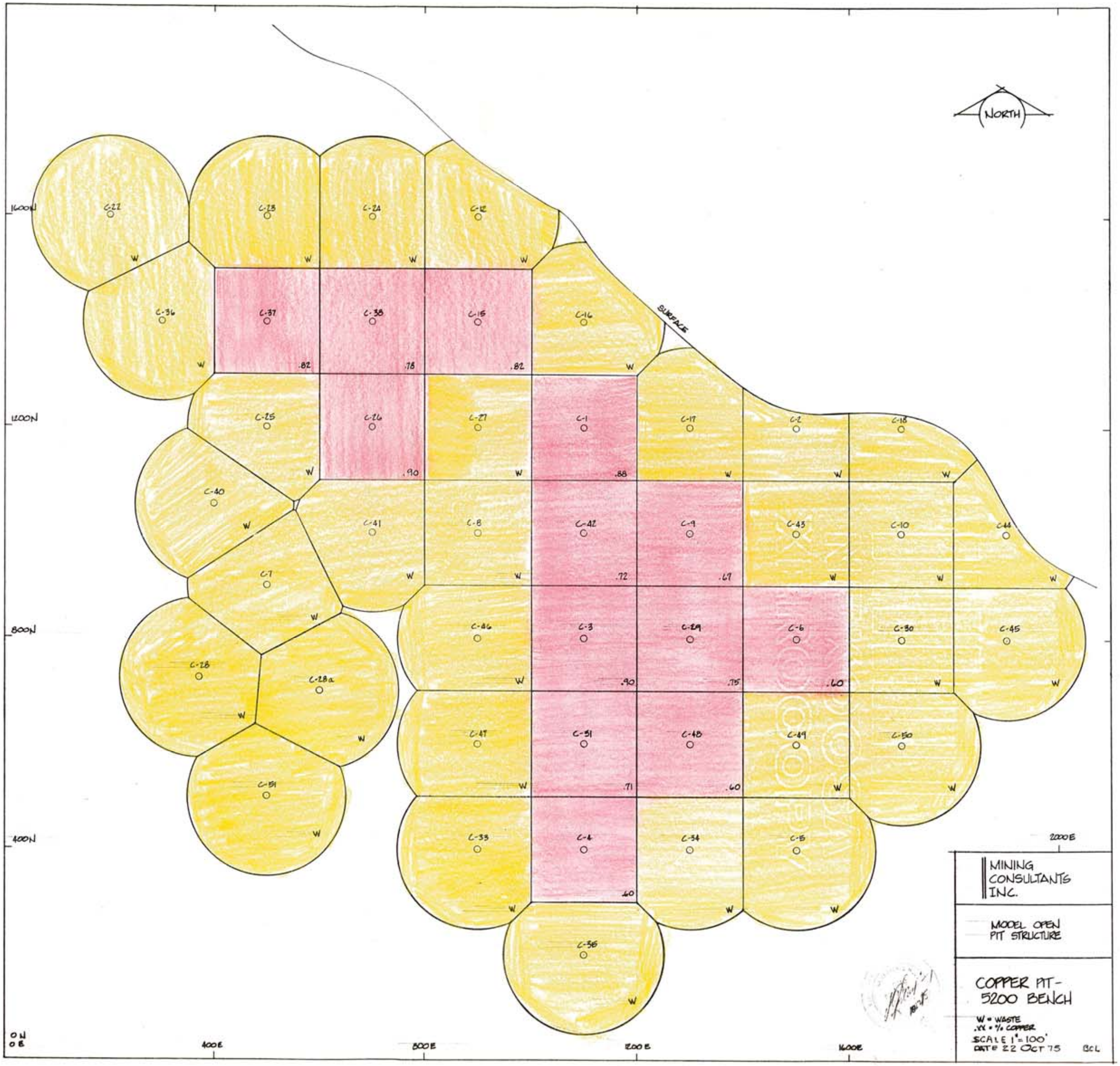

SECTION A-A'
TYP. SURFACE INTERCEPT LINE
TYP. DRILL HOLE
TYP. GRID INTERSECTION
TYP. PIT LIMIT LINE; 45° SLOPE, 1 ORE TO 3 WASTE
SECTION B-B'
SECTION C-C'
SECTION D-D'
SECTION E-E'
SECTION F-F'
SECTION G-G'
MINING CONSULTANTS INC.
MODEL OPEN PIT STRUCTURE
COPPER PIT SECTIONS
W = WASTE
O = ORE
SCALE: 1"=100'
DATE: 22 OCT 75 BCL


NORTH
5200 BENCH
SURFACE INTERCEPT
5160 BENCH
5120 BENCH
1600 N
1200 N
800N
400 N
0 N
0 E
5240 BENCH
5160 BENCH
5120 BENCH
5200 BENCH
5160 BENCH
SURFACE INTERCEPT
5200 BENCH
5240 BENCH
400 E
800E
1200E
1600E
2000E
MINING
CONSULTANTS
INC.
MODEL OPEN
PIT STRUCTURE
COPPER PIT
PIT LIMITS
BENCH TOES
SCALE: 1"=100'
DATE: 22 OCT 75
BCL

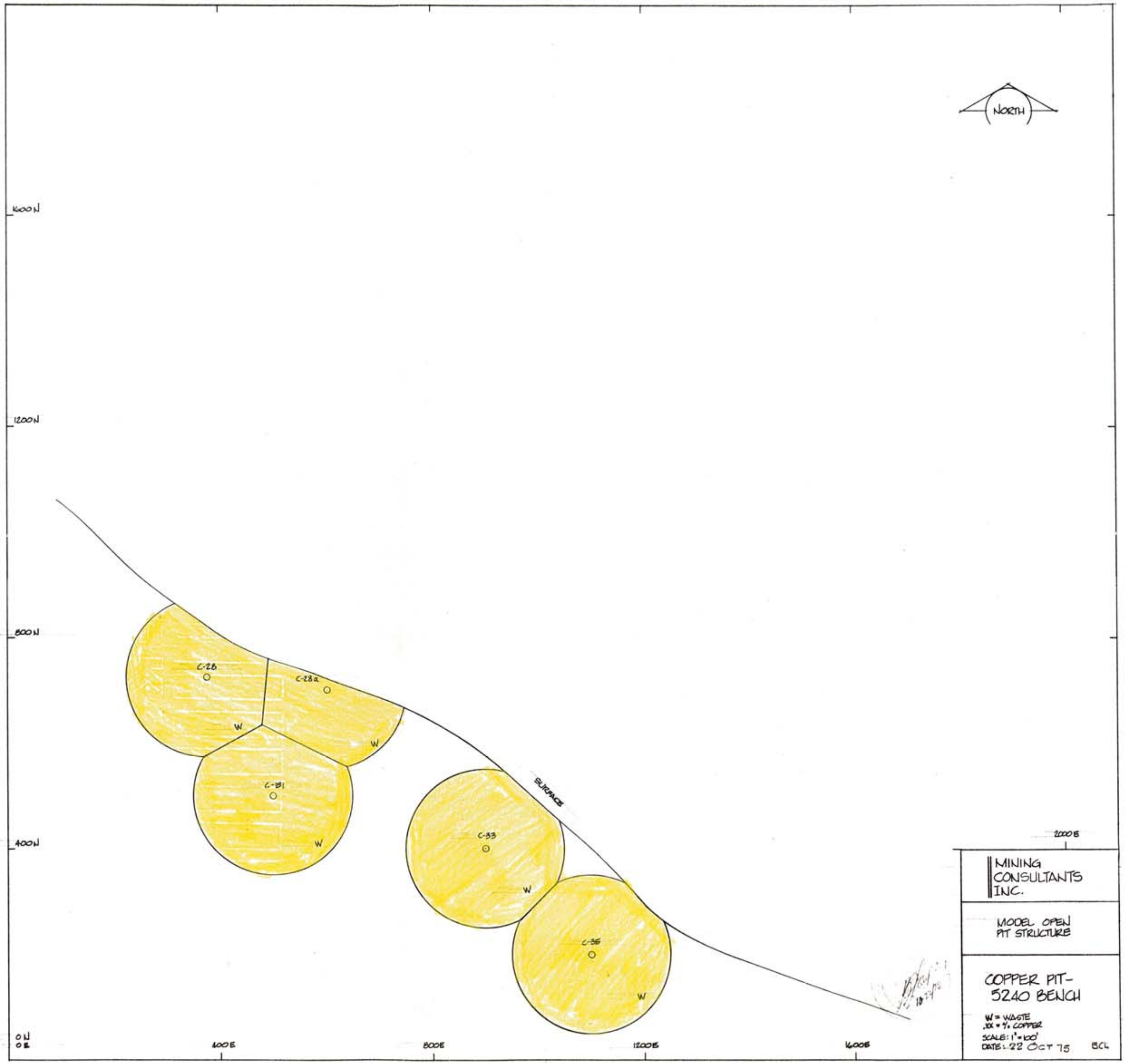


NORTH
1600N
1200N
800N
400N
0N
0E
400E
800E
1200E
1600E
2000E
C-20 W
C-21a W
C-21 W
C-22 W
C-23 W
C-24 .60
C-12 W
C-13 W
C-14 W
C-36 W
C-37 W
C-38 1.40
C-15 .76
C-16 .80
C-39 W
C-25 .60
C-26 1.01
C-27 W
C-1 .74
C-17 .85
C-2 .60
C-18 W
C-19 W
C-40 W
C-41 W
C-8 .83
C-42 .65
C-9 1.13
C-43 1.10
C-10 .65
C-44 W
C-7 .62
C-46 W
C-3 .72
C-29 .85
C-6 .90
C-30 .67
C-45 W
C-28 W
C-28a W
C-47 W
C-31 .60
C-48 W
C-49 .72
C-50 W
C-51 W
C-33 W
C-4 W
C-34 W
C-5 W
C-35 W
MINING CONSULTANTS INC.
MODEL OPEN PIT STRUCTURE
COPPER PIT- 5120 BENCH
W = WASTE
.XX = % COPPER
SCALE: 1"=100'
DATE: 22 OCT 75
BCL


NORTH
1600N
1200N
800N
400N
0N
0E
400E
800E
1200E
1600E
2000E
MINING CONSULTANTS INC.
MODEL OPEN PIT STRUCTURE
COPPER PIT-
5160 BENCH
W = WASTE
.XX = % COPPER
SCALE: 1"=100'
DATE: 22 OCT 75



NORTH
SURFACE
1600N
1200N
800N
400N
0N 0E
400E
800E
1200E
1600E
2000E
C-22
C-23
C-24
C-12
C-36
C-37
C-38
C-15
C-16
C-25
C-26
C-27
C-1
C-17
C-2
C-18
C-40
C-41
C-8
C-42
C-9
C-43
C-10
C-44
C-7
C-46
C-3
C-29
C-6
C-30
C-45
C-28
C-28a
C-47
C-48
C-50
C-33
C-4
C-34
C-5
.82
.78
.82
.90
.88
.72
.67
.90
.75
.60
.71
.60
.60
W
MINING CONSULTANTS INC.
MODEL OPEN PIT STRUCTURE
COPPER PIT- 5200 BENCH
W = WASTE
.XX = % COPPER
SCALE 1" = 100'
DATE 22 OCT 75
BCL



NORTH
1600N
1200N
800N
400N
0N
0E
400E
800E
1200E
1600E
2000E
C-28
C-28a
C-31
C-33
C-35
W
SURFACE
MINING CONSULTANTS INC.
MODEL OPEN PIT STRUCTURE
COPPER PIT-
5240 BENCH
W = WASTE
.XX = % COPPER
SCALE: 1" = 100'
DATE: 22 OCT 75
BCL

OPEN PIT QUALITY CONTROL

TECHNIQUES

Quality control of an open pit ore body is achieved by following four basic steps:

1. Establishing sample stations or reference points on the pit benches
2. Taking representative samples
3. Preparing the samples for assaying
4. Assay recording and interpretation.

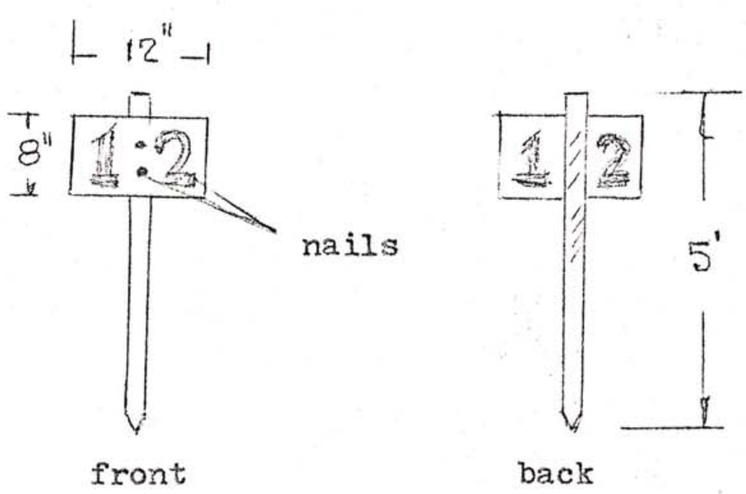
Establishing sample stations on a pit bench is done for the purpose of separating ore from waste and/or separating various grades of mineral content.

The spacing of sample-station markers usually does not exceed 100 ft. Geology of the immediate area is used to limit the spacing. Minimum spacing is determined by shovel limitations and is seldom less than 25-30 ft. for large shovels. One procedure for establishing sample stations is to place markers approximately every 50 feet along the crest of pit benches. It is not critical where the markers are originally located because they will be adjusted when the blast drill-hole and bench assays are known. The marker must be placed near enough to the crest so that it can be read from the bench below.

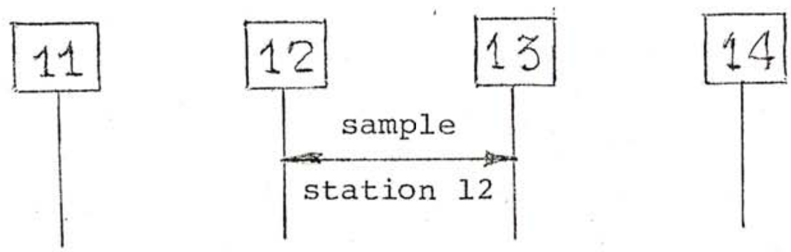
One type of marker which serves the purpose is constructed using a five foot wooden stake with a large square of white cardboard nailed to the top. The cardboard is thick enough to prevent bending and breaking from weather conditions.

Sample station numbers are painted on both sides of the cardboard, using a stencil (at least six-inch high numerals so they can be seen from a distance), and fast drying bright orange or red spray paint. Painting the numbers on the markers using a small brush works, but the numbers are usually not as readable.

Sample station numbers run consecutively beginning with number one on each bench. An example of a sample station marker is shown below with approximate dimensions.



A marker usually denotes the beginning of a sample station. For example, the area between markers 12 and 13 shown below is sample station 12:



This procedure for sample station location prevents confusion.

When the sample-station markers are established on the area or bench to be mined, the next step is to collect assay samples from the blast-drill hole cuttings.

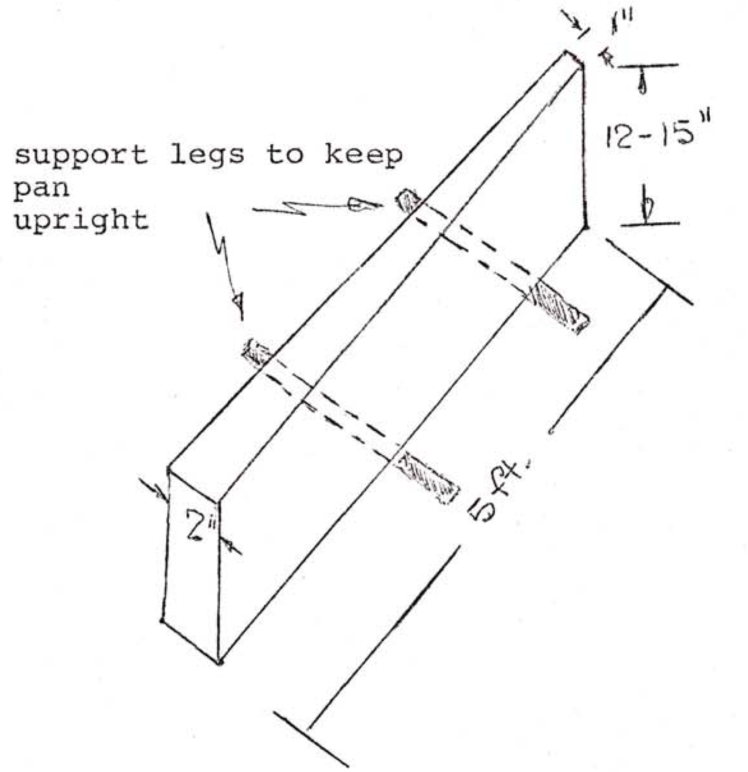
Tools and materials required for this job include (1) a long-handled, pointed-nosed shovel, (2) canvas sample bags with tie strings at the top capable of holding 10-15 pounds of material, and (3) pencil and small identification tags (1-inch x 2-inch piece of paper for each drill hole samples). Leather gloves are needed to sample several holes.

Drill hole cuttings are sampled very carefully. Using the shovel, remove and discard a pie-shaped section (approximately the width of the shovel head) from the mound of cuttings surrounding each hole.

Concentrating on a 3- or 4-inch area to the right or left of the pie-shaped section, skim and discard that portion of cuttings from the top of the mound that represents subgrade material. A 40-ft. high bench usually is drilled at least 44-ft. to insure proper toe breakage when blasting. Cuttings from the bottom 4 ft. of the hole (the top 1/11 of the mound of cuttings) is discarded to guard against sample contamination. A 3- or 4-in. thick vertical slice is then loaded into a canvas sack and identified using a piece of paper showing the date, bench elevation or designation, sample-station number, and drill-hole number. The drill-hole number was previously written by the driller on a small wooden stake after the hole was drilled and stuck into the pile of cuttings.

Another method of taking drill hole samples is to use a wedge-shaped, thin-metal pan. The narrow end of the pan is placed next to the drill stem with the long axis parallel to the radius of the drill hole. As the hole is being drilled, cuttings blown from the hole land in the pan. Once the hole is drilled to grade, the pan full of cuttings should be pulled away from the drill stem. The contents of the pan are then poured into a sample sack and identified.

A properly fabricated pan is longer than the radius of the conical mound of drill cuttings and with high enough sides to prevent spill over. A five foot long pan, one inch wide at the narrow end, two inches wide at the other end, and with sides 12 to 15 inches high is ample for holes up to 12-in. in diameter and 40-ft. deep. An illustration of a pan is shown below:



The pan method of sampling drill hole cuttings is not always as reliable as the trench method described earlier. During periods of strong winds, the drill hole cuttings are sometimes blown to one side or another bypassing the pan. Also, pans get damaged from rough treatment and sometimes lost.

After the drill-hole samples are gathered and identified, they are taken to a sample building and prepared for assaying. Sample preparation of drill-hole cuttings includes splitting, crushing, drying, and pulverizing.

Sample splitting is a method of reducing the original sample to a fraction of the original weight and volume without altering the percentage of mineral content.

First pour the drill cuttings from one hole (10-12 pounds) into a pile on top of a table. Mix the pile thoroughly with the fingers and divide the pile into two halves. Discard one half. This process is repeated until a 1-lb. sample remains.

The 1-lb. sample is run through a cone or gyratory crusher two or three times (Denver Equipment Company makes several sizes) to reduce the cuttings to approximately minus 1/4-in. in size. The cone crusher also splits the sample. Discard half the sample each time it is crushed.

The remaining 2- to 4-ounce sample is poured into small metal pans (aluminum 3 in. x 3 in. x 8 in. or similar) with the original identification tag and dried over electric or gas hot plates for 15 or 20 minutes to remove moisture.

The sample is then poured into a pulverizer grinder to reduce the particle size to at least a minus 200 mesh.

At this point the pulverized sample is screened using a 200-mesh screen. Any particles that do not pass through the screen when using a rapid vibrating motion are returned to the pulverizer grinder for further reduction. After pulverizing, the sample is placed in a small sample envelope, sealed, labeled, and sent to the assay lab.

Sample preparation rooms are very dusty because of all the grinding equipment. A duct system where the air and dust is removed by a heavy duty electric fan should be placed over each crushing and grinding machine. Grinding personnel are required to wear safety glasses and respirators at all times. The cone crushers create considerable fly (small rock particles traveling at high velocities) if the rock being crushed is fairly hard. Protective hearing aids such as ear muffs or ear plugs are also worn.

After assaying each sample, the results are recorded in a log book and taken to the field for drill-hole correlation and sample station adjustment.

The next step is to group together the ore and waste holes into separate sample stations. Some waste holes can be included in a sample station of ore if the average assay of all the holes within the station is above the cutoff grade.

Any holes that appear to have an assay out of the ordinary (a great deal higher or lower copper content that the surrounding holes) are carefully examined. If no physical differences such as rock type or mineralization are apparent in the hole cuttings, another drill hole sample is taken and a new assay determined.

After all drill hole assays are field checked and sample stations relocated, the sample crew proceeds with bank sampling.

Bank sampling requires a minimum of two people no matter how few samples are to be taken. One man gathers the sample while the second man stands away from the bank and watches for falling rock. The second man (generally called the lead sampler) also guides the sampler to where the sample station begins and ends by noting the sample station markers above.

A bank sample is taken as high as possible upon the toe (fallen rock piled up along the face of the bank). Fist-size rocks are pulled from the face of the bank and from the toe. Fines (smaller size material) are included in the sample. A 10-pound sample is gathered from each sample station.

Areas within sample stations that look irregular are sampled separately. Drill holes with wide spacing may completely miss some geologic structures such as dikes and faults.

When all bank samples are gathered (a new sample should be taken from every station mined in the last 24 hours) they are taken to the sample building and prepared for assaying in a similar manner as the drill-hole samples.

Bank samples require primary crushing (Massco 4 in. x 6 in. jaw type crusher) for reduction of the rock to a minus one inch. After primary crushing, the remainder of the preparation is exactly the same as for drill cuttings, splitting, secondary crushing in cone crushers, drying, and pulverizing.

Once the bank samples are assayed, the results are logged according to sample stations and in a manner so the bank assays and drill hole assays can be compared simultaneously. An example of a log sheet for bank and drill hole assays is:

Sample Station	Bench Elevation - 6200 Bank Assay	Old Drill Hole Assay	Broken Drill Hole Assay	New Drill Hole Assay
6	65-63-61-~~42~~-53	61^3	60^2	55^3
7	34-29-32	25^2	32^3	34^2
8	55-66-68-~~99~~-72	53^3	65^3	70^3
9				43^3
10				41^2
11			35^3	
12		26^3		
13				

In the assay recording shown, the bank assays determine if the sample station is ore or waste. The drill hole assays may confirm or not confirm the bank assays.

The last three columns to the right of the log page are used to record drill hole assays. The column to the extreme right (new drill hole assay) is used to record the average copper content of all drill holes within a sample station that have not been blasted. For example, in sample station six, 55^3 means 3 holes that average 0.55% copper. An additional hole drilled in station six that assayed .35 would change the 55^3 to 50^4 ($55 + 55 + 55 + 55 = 200 \div 4 = 50$ or 50^4).

The drill hole assays initially entered with pencil in the extreme right column (new drill hole assay) is transferred to the middle column (broken drill hole assay) when the holes are blasted. The right column is then erased and remains blank until new holes are drilled in that sample station. Hole assays that are in the middle column are transferred to the left column (old drill hole assay) when the broken muck is mined out.

In station 6 the numbers 65 - 63 - 61 - 42 represent previous bank assays taken as the shovel advanced into the bank. Note the diagonal line drawn through 42 in sample station 6. This means that the assay interpreter thought 42 was too low since the last bank assay was 61 and the broken drill hole assay was 60. The drill assays (old, broken and new) indicate the grade is dropping, but probably not as low as 42. A new bank sample in station 6 produced a 53 assay. This is closer to the 60 broken drill hole assay and 55 new drill hole assay; therefore, it is considered valid.

Note the diagonal line drawn through 99 in sample station 8. The drill assays do not indicate any material that should run higher than 70 so a new bank sample was taken. The new assay of 72 is more in line with past bank samples, and present broken and new drill samples, therefore it it accepted.

If the cutoff grade is 60 (.60 of 1% copper) then sample station 8 is mined for ore at a .72 heading, and sample stations 6 and 7 are mined for waste at a .53 and .32 heading respectively. This type of assay recording is kept for each bench or working level.

MINE QUALITY CONTROL--ASSAY REPORT

Date ____________________

Bank SAMPLES

How Reported % X-Ray

DESCRIPTION	Sta.	Copper	Iron	Moly	Gold	Silver
B-5650 Est.	1 139	2.00 1.55	2.4 2.8	003 017		
B-5700	58 58ck 59	.92) .91 .90) .44	1.4 1.7 1.4	008 003 039		
B-5750	71 74 74ck	.32 .66) .64 .63)	1.8 2.5 2.1	008 016 013		
B-6050	18 19	.34 .33	3.6 7.3	012 011		
B-6100	25	1.32	10.4	000		
B-6300	55 57	.05 .03	4.3 4.2	000 001		

MINE QUALITY CONTROL
ASSAY REPORT

JOHN DOE CORPORATION

ROTARY DRILL SAMPLES

HOW REPORTED % Elec.

Date	Bench Area	Sta. #	Hole #	Cu. %	Fe. %	Mo. %	N - S Coord.	E - W Coord.	Elev.	Conc. %	Rec. %	Code
1/22/75	6000ISLA	37	11	.18								
"	"	37	12	.32								
"	"	32	13	.19								
"	"	38	14	.19								
"	"	38	15	.25								
"	"	38	16	.20								
"	"	38	12	.27								
"	"	39	18	.32								

OR 1976 - AREAS - January 6, 1976

RENTAL RATES/MONTH		
HRS	352 HRS	528 HRS
,250	$ 2,000	$ 3,200
,600	2,560	4,096
,825	2,920	4,672
,550	4,080	6,528
,250	5,200	8,320
,075	6,520	10,432
,750	9,200	14,720
,500	18,400	29,440
,370	$ 2,192	$ 2,507
,955	3,128	5,000
,270	3,632	5,811
,760	4,416	7,070
,460	7,136	11,420
,000	11,200	17,920
,985	$ 3,176	$ 5,082
,425	3,880	6,208
,325	5,320	8,512
,700	10,720	17,152
,250	$ 2,000	$ 3,200
,840	2,944	4,710
,750	4,400	7,040
,595	5,752	9,203
,080	8,128	13,000
,390	11,824	19,000
,750	15,600	24,960
,500	$12,000	$19,200
,500	15,200	24,320
,500	$10,400	$16,640

as term of rental,
above figures are
lease-purchase or
e option, Caterpillar
% of rentals apply,
thin 12 months of
lly 1% of purchase

REMARKS

One - Gardner-Denver Air Trac, Model ATD-3700 with 12 foot chain feed and PR-123J Independent Rotation Drill

Price; F.O.B., Salem, Virginia:------------------- $49,295.00
Less 10%:--------- $44,365.00

One - Gardner-Denver Portable Air Compressor, Model SP750 Rota-Screw, mounted on four pneumatic tires and powered by an eight cylinder Cummins Engine, Model V-903-C265.

Price; F.O.B., Quincy, Illinois:------------------ $43,713.00
Less 10%:--------- $39,341.00

If the above units are purchased out-right; Net 30 day Terms will apply on the discounted prices. On a Lease Purchase Option, the following Terms will apply:

Model ATD-3700, Single Shift = $1,350.00 Per Month
Double Shift = $2,025.00 Per Month
Triple Shift = $2,700.00 Per Month
Model SP-750, Single Shift = $1,050.00 Per Month
Double Shift = $1,575.00 Per Month
Triple Shift = $2,100.00 Per Month

If the equipment is purchased within three months, 100% of the accrued rentals will apply to the Purchase Price, if purchased within six months, 90% of the accrued rental will apply. If the equipment is purchased after six months, but, within twelve months of shipment date, 80% of total accrued rentals will apply.

They presently have in Denver two Model ATD-3700 Air Trac's and two Model SP750 Compressors, used, but, only have 300 hours on them.

Price Per Air Tracs:------------------------$42,795.00
Price Per Air Compressor:-------------------$38,713.00

The same rental rates and lease-purchase option as described above will apply.

One - Gardner-Denver Model G7B, Universal Bit Grinder.

Price; F.O.B., Job Site:---------------------------$ 3,147.00
Less 10%:----------$ 2,832.00
Rental Rate: = $275.00 Per Month

The following Accessory Equipment will be on an out-right purchase.

One - CL5-1762V9 Shank:---------------------------- $ 57.65
One - CL5-1712VJ, 12 ft. Drill Rod:---------------- $ 142.05
One - CL5-1700V, Coupling:------------------------- $ 22.95
One - 3" Timken Button Bit:------------------------ $ 73.10
One - 3-1/2" Timken Button Bit:-------------------- $ 92.30
One - 3" Timken "X" Bit:--------------------------- $ 73.10
One - 3-1/2" Timken "X" Bit:----------------------- $ 92.30
One - 40 Lb. Bucket Thread Lube:------------------- $ 30.80
One - 55 Gallon Rock Drill Oil:-------------------- $ 136.00