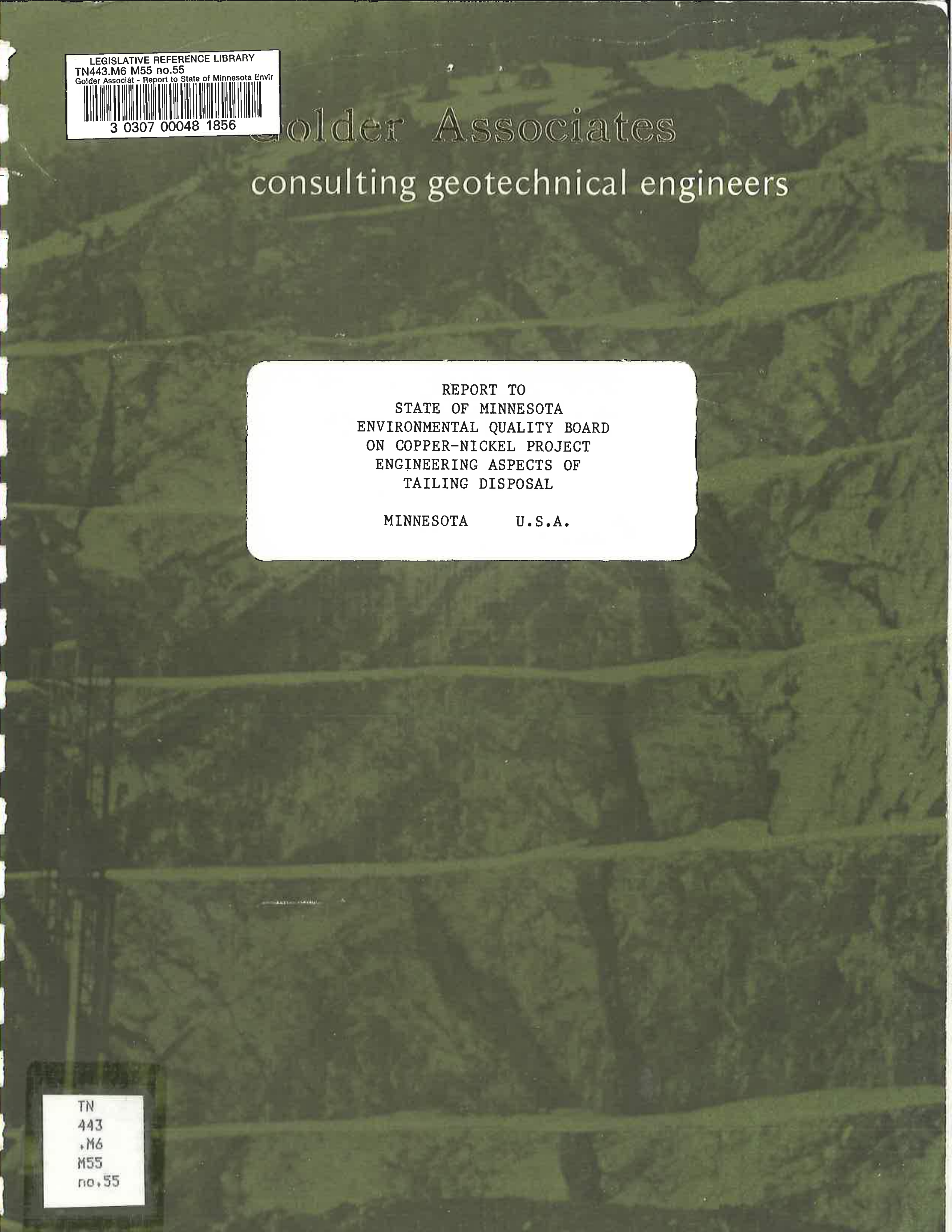

Golder Associates

consulting geotechnical engineers

REPORT TO
STATE OF MINNESOTA
ENVIRONMENTAL QUALITY BOARD
ON COPPER-NICKEL PROJECT
ENGINEERING ASPECTS OF
TAILING DISPOSAL

MINNESOTA U.S.A.

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ABSTRACT

A study is made for planning purposes of the geotechnical engineering aspects of tailing disposal for potential copper-nickel mining in northeastern Minnesota. Consideration is given to tailing embankment design, seepage control, reclamation, fugitive dust, and cost of pumping tails to distant locations. Characterization is made of the region for tailing disposal around the belt of mineralization, and a few examples are given of possible tailing pond layouts. Guidelines for the establishment of regulatory procedures are indicated.

1.0 INTRODUCTION

This report discusses waste disposal systems for potential copper-nickel mining in northeast Minnesota. The belt of mineralization trends SW to NE and lies east of the Mesabi Iron Range, west of Lake Superior and south of the Boundary Waters Canoe Area (BWCA). The study was authorized by the Regional Copper-Nickel Study under the Minnesota Environmental Quality Board (MEQB).

The report address the following four items:-

a) Siting and Design Criteria for tailing disposal systems, including reclamation.

b) Methods of seepage control.

c) Methods of fugitive dust control.

d) Capital and operating costs of long distance tailing pumping lines.

The work has been handled jointly through Golder Associates Inc., by Golder Brawner & Associates Ltd. in Vancouver, B.C., together with Kilborn Ltd. of Toronto, and NUS Corporation, Maryland. Items a) and b) above have been studied by Golder Associates with the exception of the section on reclamation performed by NUS. Item c) has been studied by NUS. Item d) has been the responsibility of Kilborn.

The full report is issued under the cover of Golder Associates, but the separate sections prepared by Kilborn and NUS are included unedited by Golder Associates.

2.0 CHARACTERISTICS OF TAILING DISPOSAL SITES

Selection of areas for disposal of mine tails should be based on the following considerations:-

a) Economical distance for pumping along tailing lines.

b) Hydrological boundaries.

c) Geotechnical criteria of stability, foundation, underseepage, and local availability of embankment construction material.

a) Length of Tailing Lines

A recent survey in Canada showed that the average distance between the mill and disposal area was about 1-1/4 miles. The maximum economic length for pumping tails from a mill to a tailing pond is assumed herein to be about 10 miles. On the assumption that a mill could be up to 4 miles away from a mine, this gives a total radius of 14 miles from the mine to the tailing pond. MEQB have used this as a preliminary guideline to define the boundaries of the tailing disposal study area.

Within this broad guideline the outer boundaries of the area that could be considered for potential tailing disposal are defined more precisely by the hydrological and geotechnical criteria discussed below. In practice the economics of pumping tails will dictate that the tailing ponds should be situated as close as practicable to the mills, unless of course, the economics of construction of retention embankments is an over-riding factor. Furthermore, there will clearly be an aesthetic advantage, and in general, less risk of pollution and land disturbance by keeping the areas occupied by the mine, mill and tailing disposal as compact as possible. Construction, operation and maintenance of tailing lines and tailing ponds necessitate the building of access roads, which inevitably disturb adjacent land.

The economics of length of tailing line are assessed in detail by Kilborn/NUS in Appendix C.

b) Hydrological Characteristics

Surface water considerations are dealt with in this section. Ground water and underseepage are considered under Section c) - Geotechnical.

It is prudent to keep discrete tailing disposal areas within the confines of major water course and watershed boundaries. Existing lakes greater than 1 sq. mile in area and located entirely within a single watershed could be useful disposal basins, but there may be conflicts with other interests in the lake, this likely being more so the larger the lake.

Tailing disposal lines are seldom foolproof, and for this reason the crossing of major water courses, such as the St. Louis, Embarrass, Partridge, Kawishiwa, Dunka or Stoney Rivers, by tailing pipelines should be avoided other than in their upper reaches. If a river crossing is unavoidable, emergency dump valves designed to operate in case of power failure or line blockage should be incorporated. These valves should be located at nearby low points, perhaps in a natural basin large enough to contain the spillage and preclude it entering the river until the line can be repaired.

Since blockage, breakage and leakage of tailing lines is commonplace, these factors should be considered when siting ponds. If a fail-safe tailing system is the objective, the ponds should be sited such that tails lost from the line at any point remain within the catchment basin of the tailing pond. This usually means that the tailing pond should be located as close as practicable to the mill, especially in flat lying regions. Obviously a tailing pond and a mill located in different watersheds is not a fail-safe system.

Management of water, both decant water and pond underseepage, together with natural run-off, can be facilitated by designing and operating discrete tailing disposal areas such that they fall entirely within the confines of at least major watershed basins. Basically, there is no strictly engineering reason why tailing disposal sites should not straddle watersheds, but criteria may well be established by virtue of different levels of allowable pollution from one basin to the next. For example, standards to the north and to the south of the Laurentian Divide would likely indicate separation of tailing ponds. The only purely engineering consideration would therefore be in the cost of incorporating different seepage control measures.

The major watersheds, the zone of mineralization, and the layout of possible mine sites are shown on Figure 1.

An obvious primary boundary is the Laurentian Divide, which separates water flowing north to Hudson Bay from that flowing south to the Great Lakes Basins. Within the area considered for tailing disposal, the gathering ground north of the Divide lies entirely within the Rainy River watershed. In so far as the Laurentian Divide is roughly orthogonal to the belt of mineralization and the study area, it ought not to be too difficult to site tailing ponds entirely on one side or other of the Divide. Also, a large part of the western boundary of the area for tailing disposal could conveniently be defined by the NE-SW trending arm of the Laurentian Divide between Biwabik and the West Two and East Two Rivers watershed.

The secondary watershed boundaries could be used to subdivide the tailing disposal region into 13 areas as follows:-

A. St. Louis River Watershed

- i) Upper and lower reaches of the main stem of the St. Louis River (2 Areas)
- ii) Embarrass River
- iii) Partridge River
- iv) Water Hen Creek
- v) Whiteface River
- vi) Cloquet River - marginal and omitted.

B. Rainy River Watershed

i)	Kawishiwi River	
ii)	Bear Island River	
iii)	Stoney River	
iv)	Dunka River	
v)	Omaday and Bogberry Lakes	small but significant watersheds adjacent to the orebodies
vi)	Keeley Creek	
vii)	The small unnamed bog area to the N of the Dunka River watershed	
viii)	Pike River	marginal and omitted
ix)	West Two and East Two Rivers	

While the boundaries of these secondary watersheds should not be treated as rigid criteria for subdivision of tailing areas, they will prove useful in assessing any particular site. In redefining the outer limits for the potential tailing disposal areas, the eastern margins of Whiteface River and Stoney River watersheds can be adopted, as can the western margin of Bear Island River watershed.

Similarly, within the watersheds the main stems of major rivers will delineate boundaries for tailing ponds, as well as route selection for tailing lines as mentioned previously. Dumping of tails in a basin sited across an existing river, even in its upper reaches, may well involve expensive river diversion works, the long term stability of which needs careful consideration; in time after mining has ceased the river will likely try to return to its original channel, which is now through the tailing pond. Once again, however, it should be pointed out that from a strictly engineering mechanics standpoint there is no reason other than siltation, why tails should not be discharged directly into a river. Historically, and even quite recently, this was the common method, because of cheapness. Nowadays, pollution considerations almost invariably eliminate such a choice. One such major copper mine that uses river discharge of tails is Bougainville mine in New Guinea. In this case, rivers carry the tails to swamplands where it settles out. A major engineering consideration in this scheme was the severe seismicity of the region which was said to be a threat to normal embankment ponds. However, earthquake considerations would not be a valid argument in Minnesota.

c) Geotechnical Characteristics

Useful regional surficial geology is contained in the preliminary report by the U.S. Geological Survey, "Physiography and Surficial Geology

TABLE 1

Physiographic Provinces

Province	Remarks
(Location indicated on Figure 2)	
A. Shallow bedrock-Moraine	Ridges of exposed bedrock with peat bogs and wetlands between. Moraine till thin and generally < 10 ft. thick, apart from 100 mile swamp adjacent to Mesabi Iron Range (E). Some outwash sand and gravel.
B. Drumlin-bog (Toimi Drumlin Field)	Low elongated ridges or hills of till aligned roughly NE-SW, and up to 75 ft. thick. Probably some ridges have cores of bedrock. Lowlying ground between ridges infilled with peat and some clay and silt.
C. Embarrass and Dunka Rivers Sand & Gravel Plain	Flat lying ground with thick deposits of permeable outwash sands and gravels. Depth to bedrock generally in excess of 50 ft., and > 200 ft. in places along the Embarrass River.
D. Outwash-Moraine Complex	Small discontinuous outwash plains of sand and gravel located between hills and ridges of till moraine.
E. Embarrass Mountains (Mesabi Iron Range)	Taconite mining region of high bedrock relief. Iron mining precludes use for Cu-Ni mining tailing disposal.
F. Seven Beaver - Sand Lake Wetland	Flat lying, peat covered lakeland. Seven Beaver, Sand, Round, Pine, Stoney, and Long Lakes in this area. Big Lake shown in adjacent area A.
G. Aurora - Markham Till Plain	Flat marshy land underlain by reddish clay till. Depth to bedrock in excess of 150 ft. on W limit. Granular, pervious, and moraine deposits in places. Elsewhere, peat over soft unconsolidated glacial lake sediments.

of the Copper-Nickel Region, Northeastern Minnesota". This information is helpful for general planning needs, and a summary is shown on Figure 2. Site specific proposals would, in general, need a more detailed soils investigation to supplement this data.

The physiographic and surficial deposits of the region have been divided by the USGS into 7 areas. These are shown in Table 1, together with remarks on the character of the soils that are likely to be significant from an engineering aspect.

From an engineering standpoint, at least with respect to foundation conditions and control of underseepage, the copper-nickel belt and adjacent provinces A and B to the east should provide more attractive sites for tailing disposal than the provinces C and G to the west, where the iron tails are disposed. The high ground of the Mesabi Iron Range, Province E, separates the provinces C and G to the north and south, with the exception of a small lobe of province C at the eastern end of the Iron Range. The less attractive feature of provinces A and to a lesser extent province B, is the scarcity of native construction materials. Province F is either lakeland or peat bog, which is unattractive as a foundation material. Furthermore, province F is located in general about 10 miles away from a potential copper-nickel mine, and for this reason it may well be economically unattractive to dispose of tails so far away. Province D contains considerable thicknesses of outwash materials along the Stoney River. To the north, south and east, however, the province is in general bounded by ridges of terminal or recessional moraine, which could provide useful impermeable barriers for tails retention.

Good materials for embankment construction are well graded sand and gravel, and glacial till. Representation of all particle sizes comprise a well graded material; fine particles fill the void spaces between larger particles, which in turn fill in between still larger particles. A good glacial till is similar to a sand and gravel, but contains about 20 per cent more fines of silt and clay, which makes the material relatively impervious in a dense compacted state. Well graded materials are preferable because they are easier to compact and they possess good shearing resistance. In the case of sand and gravel, seepage can take place without piping or internal erosion occurring. The undesirable characteristic of uniformly graded fine silty sands, such as tailing material, is their susceptibility to erosion and piping. Well graded glacial tills have the property of being self-healing if cracks should develop in an impervious zone, for example as a result of differential settlement.

In general, it can be said that provinces A and B, the largest areas, would involve few underseepage problems and provide good impermeable construction materials in those areas where the moraine till and drumlin till are found. Province B could utilize the drumlin topography. Province D is similarly attractive, but underseepage problems might be encountered near the Stony River. Provinces C and G lie over deep sediments, and suitable native construction materials may not be in easy reach. Also, control of underseepage may be more costly. Province F is in general, a remote lake wetland which is unfavourable for tailing disposal because of soft foundation conditions, and paucity of suitable borrow material for dyke construction.

d) Subdivision of Tailing Disposal Region

About a third of the region for copper-nickel tailing disposal as defined by the MEQB's boundaries, includes the area of taconite mining and its associated tailing disposal areas. Gere (1970) has indicated the magnitude of iron tailing disposal problem, by pointing out that 30 billion tons of iron tails may need to be disposed of in the future. He reasons that this could create piles on average 15 ft. thick, over an area of about 1,000 sq. miles of land. Even if the average thickness of tails is increased to 70 ft., an area of about 200 sq. miles would be needed still; this is equivalent to somewhat more than the combined areas of provinces C, E and G within the western boundary of the copper-nickel area.

In the case of the copper-nickel mining, 1 metric ton of tails requires about 0.905 cu. yds. of storage volume. Hence, for the given mine model scenario consisting of 6 open pit mines producing 20 x 10^6 metric tons/year and 7 underground mines producing 12.35 x 10^6 metric tons/year, all for 30 years, a total of 6.2 billion tons of tails would be produced, and a total area of land of 77 sq. miles would be needed for tailing storage to an average thickness of 70 ft. This value does not include the areas occupied by embankments constructed of native borrow or waste rock. Hence, we can see that somewhat more than 80 sq. miles of land would be needed, which is less than the combined areas of provinces A and B within the tailing disposal study area. The greater land area required for storage of equal tonnage of copper-nickel tails compared to iron tails, results from the lower specific gravity of the copper-nickel ore.

Obviously, one way to reduce the land area needed is to increase the thickness of the tails by increasing the height of the retaining embankments. However, the largest area with the lowest embankment will provide the maximum ratio of tails storage volume to retaining embankment volume, and hence, minimum cost to a mine operator faced with the cost of constructing the embankments. Theoretically, for a constant storage volume, and for a varying thickness of tails, h, (assumed equal to the height of the retaining embankment) on level ground, the storage to embankment ratio varies according to $1/h^{3/2}$. Clearly the area needed for a constant storage volume on the same ground varies according to $1/h$. In general, the height of tailing retaining embankments is not limited by stability requirements that cannot be overcome by engineering design. Very high embankments on flat terrain may be objectionable however, from a visual aspect. Furthermore, embankments that protrude above the natural topographic features could have a marked influence on the storage to embankment ratio, especially if natural hills and ridges are used as part of the retaining embankments. This will be apparent by considering how ponds might be constructed in the drumlin province B, if the drumlin ridges are utilized as segments of the embankments.

Bearing in mind that the maximum local relief is not more than about 100 ft. anywhere in the area, other than in the Mesabi Iron Range, it is reasonable to assume that the average thickness of tails would be of the order of 70 ft.

Apart from the above considerations, the area of pond required for adequate clarification of the water prior to reclaim or discharge to stream depends on the settling rates of the solids. This is difficult to assess theoretically. The fine sand particles settle quickly. Fine silt and clay sized particles, < 2 microns, produce turbitity. Wind and wave action inhibits settling. In general, a few days are required for clay particles to settle out. The process can be accelerated by the addition of flocculent agents. Rules based on experience with fine grind metal mine tails have been developed. In general, the pond size should provide 5 days retention time for the water, and a pond area of 15 acres (0.0234 square miles) per 1,000 tons/day of tails produced is considered adequate. For the tonnages of the mine model scenario above, for clarification purposes alone, the minimum pond areas for open pit and underground mines respectively, would be 1.3 and 0.8 square miles for each mine. For ponds of 70 ft. average thicknesses, the above criteria would be met by distributing the tails between about 7 ponds or cells each of the above minimum area for each mine respectively over a 30 year life.

Based on the above economical, hydrological, and geotechnical criteria, the region for tailing disposal can be subdivided and ranked as shown on Figure 3. The ultimate boundaries of the area have been refined and, in a few places, drawn in, but in general the band width of 28 miles has been retained. The region has been subdivided into 8 areas, of which 6 are suitable for tailing disposal.

Ranking of the areas has been based on the following reasoning:-

Class I Area	-	meets all criteria of short distance from mill to tailing pond, favourable topographic features, lies within a tributary river watershed, lies within zones of stable foundation soils and easy seepage control, does not overlap iron mining.
Class II Area	-	fails to meet one of the above criteria for a Class I area.
Class III Area	-	fails to meet two or more of the above criteria for a Class I area.

Boundaries for the classifications coincide in part with either watershed boundaries, rivers, BWCA limits, or topographical - surficial geological boundaries.

The reasons for the ranking of the areas are given in Table 2, following:-

TABLE 2

Ranking of Areas for Tailing Disposal

Area	Comments
I-A I-B I-C	Shallow to deep relatively impervious strata, within short distance of potential mine site, and favourable topography for construction of tailing ponds.
II-A	Within major watershed of St. Louis River, or too far from mine site.
II-B	Deep pervious deposits of Dunka Basin or too far from mine site.
II-C	Deep pervious deposits of Embarrass Basin, or too far from mine site.
III-A	Unfavourable topography, and overlaps with iron mining.
III-B	Unfavourable lakeland topography, deep pervious deposits, and too far from mine site.

Excluding the area occupied by Birch and White Iron Lakes in the north of the region, the usable Class I area in total is 296 sq. miles. More than twice this amount of Class II Area is available for more costly tailing disposal systems. The minor amount of Class III Area is largely unsuitable for tailing disposal.

3.0 DESIGN AND CONSTRUCTION OF TAILING PONDS

a) General

The design and construction of tailing ponds is basically much the same as any other problem in Civil Engineering in so far as it is a compromise between safety, cost, and time. The basic questions might be summarized as follows:-

i) How much stability or seepage control is required?

ii) For how long is it required?

iii) How much will it cost?

In response to the last question iii), it is obvious that tailing disposal generates no revenue, and the cheapest solution is the most attractive to mine management. While a systems approach to waste rock and tailing disposal is to be preferred, the cost of hauling waste rock to distant tailing sites is prohibitive; for this reason the separation of waste rock dumps and tailing ponds is commonplace. In theory, however, the integration of these sites could minimize both pollution control requirements and also the land area taken up by the system.

The first two questions i) and ii), are intimately related. In general the shear strength properties of tails derived from igneous rocks, which may be used in embankment construction, do not deteriorate with time. However, short term stability, that is during the life of a mining operation, may give way to instability on abandonment if measures are not taken to prevent erosion of slopes. Similarly, if it is necessary for seepage control measures to operate for many years after abandonment, either for stability reasons or to prevent pollution, it is no use having underdrains that might clog or plastic liners of a type that could deteriorate after 30 years usage. On the other hand, long term measures could be prohibitively expensive, and so once again the systems approach is called for; it may well be possible to do away with the need for impermeable liners by suitable treatment of effluents in the mill circuit.

To those familiar with conventional water storage dam design, tailing embankments differ in three respects:-

i) For the most part, rather than water, the substance stored is soft, loose comparatively impervious tails. As deposited they have little or no strength, being in a semi-fluid state. With time they can consolidate to a moderately strong material. Hence, in the long term the thrust against the retaining embankments can diminish to insignificant amounts.

ii) It is common practice to build a large part of the retaining embankments using the coarser fractions of the tails. Separation may be effected by differential sedimentation after spigotting the tails, or by cycloning. Tails are far from ideal embankment building materials, but they are usually the cheapest available. The main disadvantages of tails are their high susceptibility to both internal erosion in the dyke by piping, and external erosion and frost action on the outside slopes, and difficulties in achieving satisfactory compaction. In addition, loose saturated tails can liquefy under sudden shock loading, flowing and losing virtually their entire strength. In general, this only leads to problems in earthquake regions, although blasting very close by could also lead to liquefaction. Depending on the magnitude of the blasting, tailing ponds located greater than 1,000 ft. away should be unaffected.

iii) The embankment construction is carried out largely by the mining operators as part of an ongoing tailing disposal program. The embankment is usually raised progressively as storage requirements increase. This is more economic in terms of cash flow than building the entire embankment at the start of operations. In general, however, it is true to say that the mining operator in charge is less skilled in earth work construction than an experienced civil engineering contractor.

b) Site Investigation and Construction Material Survey

Regional surficial geological mapping of the area is useful for planning purposes, but detailed design should be preceeded by site specific field and laboratory soils investigation. Soil studies for agricultural or forestry purposes, while useful in preliminary assessment of subsoil classification, are usually of little value in determining engineering properties for design purposes. Detailed airphoto studies are very helpful in the initial appraisal of a site. Field soil investigations can be carried out by diamond, churn, rotary or percussion drilling machines, hand or power augers, and tests pits or trenches usually excavated by machine. Bedrock and soil exposures should be mapped. Details of drilling and test techniques can be found in Hvorslev (1949) and in textbooks on soil mechanics; good coverage is given in the CANMET Pit Slope Manual - Waste Embankments (1977), listed in the References.

Investigations should be directed toward an evaluation of the existing foundation conditions beneath the proposed tailing pond and its surroundings, and toward an assessment of suitable retaining embankment construction materials.

Soils deposited by glacial action, which characterize Minnesota, can be notoriously varied materials. Grain size distributions and the degree of packing of soil particle structure can vary widely. Buried bedrock channels, sometimes filled with boulders can occur. As a result, general rules for the amount of investigation at any particular site cannot be laid down. The person responsible for the design should carry out sufficient investigation to assure himself that the pertinent engineering soil characteristics have been found. This is more likely to be successfully accomplished by an engineer trained in soil mechanics. The engineer should be prepared to follow up the initial soils investigation by field inspection during construction. Foundation excavations can reveal soil conditions different from those indicated by a few exploratory boreholes. For this reason the engineer should be prepared to modify his design if necessary at the construction stage. This is the standard approach in earth dam design, and planning agencies and regulatory bodies should appreciate this philosophy.

In approving a proposed design, regulatory personnel should assess the adequacy of a site investigation with respect to the following:-

i) Distribution, depth to bedrock, grain size, and in situ density of foundation soils.

ii) Shear strength properties. Values for granular soils can often be estimated in this respect, while cohesive soils such as silts and clays will usually require laboratory testing on representative samples taken carefully and tested at their undisturbed density.

iii) Consolidation properties of cohesive silt, clay or peat soils usually requiring laboratory oedometer tests.

iv) Identification of the ground water table, and estimation of the in situ permeability of soils. Ground water baseline data for pollution control requirements should also be determined, that is the pH and level of contamination before construction commences. Field permeability testing down a borehole is usually preferable to laboratory measurements of permeability of foundation deposits. The permeability of remoulded soils proposed for construction of embankment can be assessed from the results of laboratory tests.

Typical values for soil properties such as density, shear strength, consolidation, and permeability can be found in the CANMET manual referred to above.

Bedrock geological investigations should determine whether faults occur that might give rise to leakage. This would be more important in areas where the bedrock surface is exposed or at shallow depth in the area of the pond. The seismicity of the Minnesota region is minor, and this need not be considered in designing retaining embankments.

Materials for use in retaining embankments can be classified as follows:-

i) Native borrow materials excavated and hauled to the tailing pond site, or in some cases, excavated from within the area occupied by the pond prior to impoundment.

ii) Mine derived - sand from tails,

- overburden and waste rock from mine excavations, or blast furnace slag.

Native borrow materials should be investigated for gradation characteristics, the volume of material available, and the ease of excavation with respect to position of the water table, and diggability. Glacial tills can be so hard as to require blasting and ripping, so that although it might be ideal in all other respects, the cost of excavating such a deposit might preclude its use. Similarly, coarse granular deposits where the water table is close to the ground surface can be difficult and expensive to win. Test pits and trenches are an excellent means of investigating prospective borrow materials. Laboratory testing, in addition to any of the items listed above and deemed necessary, should include compaction tests for moisture - density control during construction. If 'impervious' soils are to be used, permeability tests should be performed. Where ever possible, test techniques should be in accordance with the standard procedures as specified by the American Society for Testing and Materials.

The use of the coarser faction of tails in the construction of retaining embankments is common practice. Quite often, however, insufficient material is available for the entire embankment and it is necessary to supplement the tails with zones constructed from native borrow. In general, the structurally useful part of the tails is the sand greater than the No. 200 mesh size (0.074 mm). In the sample from the MEQB pilot plant test, this represents about 50 per cent of the tails, see Figure 4. The gradation is an average gradation for present day copper mine tails. This may not always be so in the future, since lower grade ores necessitate finer grinding resulting in the production of less usable sand faction. The volume of sand required must be assessed stage-by-stage in terms of the method of construction. The downstream method of construction requires far more material than the upstream method.

Where large volumes of waste rock are available this is an ideal source of stable material for building rock fill retaining embankments. Commonly stripping ratio, that is the volume of waste rock and overburden to the volume of ore, is about 2 or 3 to 1, and thus, large volumes of potential construction materials are usually available. Crushing the rock to smaller sizes is in general not required, but carefully built graded filters are needed to guard against migration of tails through the retaining embankment. Hence, coarse sands and fine gravels are needed in smaller quantities, and these must be either found locally or manufactured. Similarly, impervious material to seal dykes may have to be imported.

The possible use of slag as a construction material depends on the character of the ore and the process used for the complete recovery of the metal. To our knowledge, metallurgical slags have not been used in tailing dam embankment construction. Blast furnace slags resulting from the separation of iron from its ore have been used as cementing agents for bricks, road base stabilization, and cements. Currently, however, most blast furnace slags are disposed of rather than utilized. Nickel slag from Sudbury, Ontario has been successfully used as base courses for highway construction and as ballast for the Canadian National railway. A well-documented paper on the use of slags is that by Emery (1975). In general, it would seem that unless alternative sources of embankment material are unavailable, smelter slag can be better utilized elsewhere. Ground copper slag and cement mixed with rockfill have been used as underground backfill at Mount Isa Mines in Queensland, Australia.

c) Tailing Embankment Design

The method of design of a tailing embankment is an iterative process of trial, analysis, and redesign, generally following the steps outlined below:-

i) Estimate the long term storage volume needed for the mine tails.

ii) Select a storage area and using topographic maps calculate storage volume as a function of elevation of pond; estimate the ultimate height of the required embankments.

iii) Select a trial embankment cross-section that incorporates the most economic and readily available fill materials. If tailing sand is proposed, its availability in sufficient quantity when needed is critical to its use.

iv) Perform a soil mechanics stability analysis to determine the factor of safety of the trial section.

v) If the calculated factor of safety is unnecessarily high or too small modify the section, repeating the stability analysis and section modification until a satisfactory result is achieved.

vi) Decide on methods of seepage, decant and run-off control.

vii) Prepare detailed construction drawings and specifications for foundation treatment, fill placement, and water control measures.

Storage volumes can be calculated from mill tonnages and the dry density of in situ tails, which can be assumed for planning purposes in this study to be 90 lb./cu. ft.

The topographic maps of the U.S. Geological Survey to scale of 1:24,000 and 1:62,500 at contour intervals of 10 ft. are useful for planning. For detailed design studies, larger scale maps at 1:2,000 or 1:5,000 and smaller contour intervals of 2 or 5 ft. are required, and these can be prepared by photogrammetric mapping.

Economy dictates that embankment slopes should be as steep as possible commensurate with stability. Stability will be governed by the shearing resistance of both the embankment fill and the underlying foundation soil; one or the other will be the governing criterion. In some cases it will be economical to excavate weak foundation soils of shallow extent. In both the fill and the foundations pore water pressures will influence stability. For initial planning purposes, if weak foundation soils are non-critical, the following gradients can be assumed for the outside slopes of retaining embankments:-

i) Free draining lightly compacted sand and gravel, or mine waste rockfill, use 1.5 horizontal to 1.0 vertical.

ii) Well compacted cohesive glacial till, use 2.0 horizontal to 1.0 vertical.

iii) Tailing sand lightly dozed into place, use 3.0 horizontal to 1.0 vertical.

No definite values can be quoted where weak foundation soils are present. Each case must be assessed on its own merits, but values in the range of 4.0 to 6.0 horizontal to 1.0 vertical might be appropriate. Tailing embankments, however, are often raised very slowly and this fact can be put to good use by consolidating weak underlying silts, clays or peats. In many areas of northeastern Minnesota, soft soil foundation conditions are quite common. An example of tailing dam construction on a peat foundation is described by Taylor and D'Appolonia (1977) for the proposed new taconite tailing pond at the Fairlane Plant south of Aurora. A full scale test fill was constructed to determine the strength behaviour of the peat, and the results might be applicable for nearby potential copper-nickel tailing disposal. The test section indicated that strain compatibility between the peat and the overlying tailing embankment fill would not be a problem after placement of the initial lifts. The proposed taconite tailing dyke will be instrumented to monitor behaviour of the peat during the on-going construction phases; pore water pressures will be measured using piezometers. The results will be compared with design assumptions made in stability analyses as well as to control the rate of fill placement. In this proposal the initial outside slope will be reached fairly early during the life of the operation, thus, facilitating vegetation and final abandonment and reclamation. Finally, the soft subsoils left in place and consolidated will provide low permeability barriers for control of leachates. Critical in this design, however, is the availability of usable tailing sand, and the stability of foundations of the dykes at the time that storage demands dictate dyke raising.

Stability analyses should account for the shear strengths and densities of both the foundation and embankment soils, and pore water pressures must be included, estimates of which can be made using flow nets. Consolidation theory can be used to estimate pore water pressures in compressible foundations. Such analyses are routine soil mechanics exercises, but the various methods are too lengthy to describe here. Reference can be made to any good textbook on soil mechanics such as Terzaghi & Peck (1967), or to the CANMET Manual (1977). These analyses should be carried only by persons well versed in the engineering mechanics involved.

Freeboard for retaining embankments should be based on the maximum flood flow and tailing water storage capacity, and also on wave run-up assuming that water is ponded against the embankment. Rip-rap can be placed on the inside face to minimize wave run-up, and to reduce wave erosion and details are given in the CANMET Manual (1977). Minimum freeboard should be about 3 ft. above the maximum estimated water level during flood and/or peak tailing water storage. The embankment crest width is another related design detail. It may be dictated by the minimum allowable distance for percolation of water through the embankment. Usually, however, it is based upon the minimum width for operation of construction equipment, which is commonly about 20 ft.

As stated previously, tailing disposal is an added cost to mineral production, and consequently the cheapest disposal method is sought. Generally tails cost little to use and traditionally, the upstream embankment method has been adopted; this is illustrated in Figure 5a. The starter dyke is constructed from local borrow material. Subsequent dykes are built from tails spiegotted off the existing crest and dozed up to form a new crest. The prime requirement in an embankment of course is to keep the crest above the level of the pond. With the upstream method this is comparatively easy to accomplish with minimum earth work. However, using the upstream technique the upper dykes are built over partially consolidated tails and slimes sedimented in the pond. Depending on the degree of consolidation of the slimes this may involve more or less risk of a complete slope failure through the slimes beyond the limits of the retaining dykes, as shown on Figure 5a. In practice the degree of consolidation, and hence the shear strength of the slimes depends on the under drainage; if slimes overlie pervious foundation soils such as an alluvial sand and gravel deposit and these are not sealed, the slimes can consolidate under downward seepage forces and become quite stable in a short time. Even so, there is a limited height to which an embankment constructed by the upstream method can be raised before failure takes place. For example, if the slimes have an average shear strength of 500 psf and the outside slope of the pond is raised at a slope of 3 horizontal to 1 vertical, stability analyses indicate that the maximum height attainable would be about 100 ft. for the embankment lying directly on firm foundations. If the embankment were underlain by deep deposits of relatively soft foundation soil the corresponding maximum height to which an embankment could be raised before failure occurred might be as little as 40 ft. Finally, although Minnesota appears to be free from earthquakes, the risk of failure of an upstream embankment is much higher where there is any chance of liquefaction occurring. Under such conditions the strength of the slimes temporarily drops close to zero, and hence along a potential slip surface little or no shearing resistance is offered. The upstream technique is really only suitable for low tailing embankments located in areas where the consequences of failure are minimized; in general they do not meet todays standards of safety and pollution control.

To overcome these shortcomings, the downstream embankment method of construction has evolved along lines similar to conventional water storage earth dams. All dam building is carried out downstream from the starter dam, as shown on Figure 5b. Although tails may still be used, the coarse sand is usually separated from the slimes by cycloning, as shown on Figure 6. Waste rock and, or native borrow material may replace or supplement the cycloned sand in the subsequent dyke rises. The base of the embankment is often provided with a drainage layer. In addition, the embankment may be compacted using heavy earth tamping machinery. Compaction should always be carried out if cohesive borrow material is used, but it may or may not be necessary with free-draining cohesionless soils depending on the shear strength needed for safety in the design. To achieve a given factor of safety with a cohesionless embankment material, it may be less costly to flatten the slope by placing a little more material rather than to use compaction throughout. High quality compaction cannot be achieved under severe freezing conditions, and it should never be attempted at temperatures less than 20 degrees F.

Although the downstream method has greater stability than the upstream method, as a comparison of Figures 5a and 5b will show the downstream approach uses far greater quantities of material in the embankment. As a result careful planning is needed, and usually greater expense is incurred. Because the embankment is built of free-draining sand it is necessary to seal the inside face either with a beach of slimes or a zone of impervious borrow material. This lowers the phreatic surface through the embankment and lessens the danger of a piping failure by backward erosion of the sand. It also meets pollution requirements where the seepage of water or passage of fines through the embankment is prohibited. The downstream method is imperative for all major tailing embankments in order to ensure acceptable standards of safety.

A variation on the downstream approach is the centerline method illustrated in Figure 5c. Instead of the crest of the embankment moving progressively downstream with subsequent rises, the crest rises vertically. This approach has the main advantage of requiring smaller volumes of fill to raise the embankment to a given height; consequently it can be raised quicker and has less trouble staying above the pond without resorting to the use of native borrow material. The centerline method, however, can run into trouble by failure of the inside face of the embankment into the pond. This can arise if the inside face encroaches too far too quickly onto the slimes. Even so, a minor failure may not be of serious consequence. Only if a major failure occurred, which allowed the pond to breach the embankment, would the result be damaging by allowing effluent and tails to escape. Careful observation of settlement and slumping of the embankment during construction usually indicates whether such a failure is imminent. If this condition is approached, stopping construction temporarily and allowing the underlying slimes time to consolidate and strengthen will usually be all that is necessary. Failing this, the crest of the embankment can be staggered downstream to obtain the necessary degree of stability.

In any area such as Minnesota where liquefaction of embankments under earthquake loading is not likely to be a problem, the requirements for compaction of sands need not be stringent. In general, free-draining sands can be placed by hydraulic methods to acceptable densities, without the need for compaction by mechanical equipment; in situ densities should be checked, however, by field tests during construction, and the criterion of a relative density of not less than 50 to 60 per cent should be applied. Relative density has a precise soil mechanics meaning based on the void ratio, e, of the soil. Void ratio is the ratio of the volume of the voids to the volume of the solids in the soil. Relative density of a cohesionless soil is defined as:-

$$RD = \frac{e_{max} - e}{e_{max} - e_{min}}$$

where:- e_{max} = void ratio of soil in its loosest state

e_{min} = void ratio of soil in its densest state

e = in situ void ratio of soil.

Relative density can therefore range from zero for the soil in its loosest state to 100 per cent from the soil in its densest state. In assessing and comparing relative densities care should be taken to ensure that the loosest and densest states are determined according to standard test methods.

Sands placed hydraulically compact well if placed above the water table and downward seepage is maintained. The use of mechanical compaction equipment adds substantially to the cost of constructing a tailing embankment.

Where tailing sands are used for embankment construction, the method of separating the sand from the slimes is either by spigotting the tails from the crest of the embankment, as used in the upstream method, or by cycloning as used in the downstream and centerline methods. During spigotting the coarsest particles of sand settle out by gravity close to the point of discharge, and the fines and slimes flow into the pond. This can be used for the downstream method where the sand is coarse and the yield is high; the sand is then dozed onto the outside face of the embankment. Cyclones, either in single or double stages, separate sand as underflow from slimes as overflow. This leads to a higher yield of sand and greater potential for embankment raising. Double stage cycloning is usually necessary if the tailing contains a high percentage of clay sized particles. No data are at present available on the actual clay content of the tails for the Cu-Ni project. Cyclones are commonly located at intervals along the crest of the embankment.

It is important to be able to determine the yield of sand usable for embankment building. What is usable sand is a design decision. For any given tailing gradation the yield of pure clean sand will be less than if some fines can be included. A design calling for a very free-draining embankment may have to exclude fine sand. However, a broader gradation of well-compacted sand can produce a very dense and stable embankment. Fines are defined as soil particles, either silt or clay, smaller than the No. 200 U.S. Standard sieve size (0.074 mm). Typically in new copper mine tailing embankments, the fines content has been about 10 to 12 per cent for single cycloning, and about 3 to 7 per cent for double cycloning. There is some evidence to suggest that where sand is placed directly on the embankment from a cyclone underflow without segregation of fines by hydraulic filling, the allowable fines content may be as high as 20 per cent; this still yields a sand sufficiently permeable to stabilize quickly. If the sand is hydrauliced any distance, the fines tend to separate out and to form thin but highly impermeable seams, which impede downward flow of water through the sand and leads to perched water tables. For a given tailing gradation and particular cyclone characteristics, the manufacturers of the equipment can estimate the sand yield, and it can be substantiated by prototype field tests. Finally, in estimating the availability of usable sand for construction phasing, allowances must be made for down time and winter conditions when hydraulic operations and compaction of sand are not possible. During severe freezing conditions below about 10 to 20 degrees F, the entire tailing product is discharged unseparated into the tailing pond. These conditions ought not to apply for more than about 2 to 3 months of the year at the Cu-Ni project.

Open pit mining operations commonly produce large volumes of waste rock. The rock waste can often be used to construct very stable rockfill embankments for tailing impoundment. This can have valuable advantages for pollution control by concentrating the waste rock and the tailing product at one location. However, because of the coarse gradation of the rockfill, and its high permeability, usually it is necessary to design a zoned embankment to prevent leakage of effluent and migration of the tails through the voids in the rockfill. Such an arrangement is shown typically on Figure 7a. Depending on the gradation of the rockfill and the filter zones, and on the quality of the seepage water, the impervious zone may be unnecessary. In such a design the tails are not utilized and they are simply spigotted directly into the pond. This approach has an added advantage in that rockfill construction can be carried out successfully in severe freezing conditions if compaction is not specified. A disadvantage may be the added cost for extra haulage and placement of the rockfill to a suitable tailing disposal site compared to the distance to convenient waste rock dumping site. Also, the scheduling of the open pit excavation has to be studied carefully beforehand to ensure that construction of the rockfill embankment stays ahead of the tailing pond impoundment; this may involve the incorporation of a higher starter dyke constructed of native borrow material. Alternatively, a hybrid design can be adopted using cycloned tailing sand in the inner portion of the embankment and waste rock in the outer portion. A typical design is shown on Figure 7b.

Conventional tailing disposal is by perimeter spigotting from the retaining embankments. An alternative arrangement, which is amenable to flat lying terrain such as northeastern Minnesota, is central discharge. This is a relatively new and untried concept, although mines at Timmins, Ontario have used it since 1965. Its main proponent is Robinsky (1975), who claims that it is not only less costly to construct and operate than the traditional approaches, but also that it provides better solutions to environmental problems such as seepage control, dam stability and ease of abandonments. The method is shown diagramatically on Figure 8. The tails are discharged from a central location and allowed to fan out in all directions to form a cone, the discharge point being raised from time to time as the cone builds up. An average slope of the cone of 3 to 4 per cent seems possible by maintaining the water-solids ratio of the tails to about 2:1 by weight, or by use of a thickener. In reality the surface of the cone should be slightly convex, since the coarser particles that settle out first will stand at a steeper slope than the fine particles settling out further away. A low impervious perimeter dyke is required to contain the mill water for reclaim. The dyke need only be about 10 ft. high and thus, the volume of embankment building can be greatly reduced; waste rock with an inside clay seal would be one solution. The method seems relatively easy to operate and regulate, because no mechanical handling of the tails is involved and only a limited length of header piping has to be raised compared to the traditional methods. For low perimeter retaining embankments, the volume of tails contained is obviously very large for an area say 2 miles in diameter. This is because the surface of the tails rise in the form of a cone above the level of the pond; in the traditional methods the tails are generally below the level of the retaining embankment. With the cone projecting above the pond, however, the problems with fugitive dust may be more severe on abandonment; because of the projecting cone the tails cannot be maintained saturated to prevent this. During operating continual rotation of the discharge point might wet the tails sufficiently to abate the dust problem. Experience of winter operation has in some cases been unfavourable. Ice builds up in the cone and the sand sloughs out on melting in the spring, resulting in flatter angles and reduced storage volumes. The long term stability of the slopes should be excellent however.

d) <u>Methods of Seepage Control</u>

Control of seepage from tailing ponds is important for three reasons:-

i) To maintain the egress of polluted water to within acceptable limits.

ii) To ensure stability of retaining embankments and their foundations.

iii) To reclaim water for mill processing.

The first rule in pollution control is to direct all uncontaminated water around a tailing pond. This reduces the amount of polluted water requiring treatment, and hence reduces the treatment costs. While large amounts of seepage may be treated, at an acceptable cost by collecting, recycling and neutralizing during the operating life of a mine, after abandonment the cost can be excessive. At mines where acid generation in metallic sulphide ores is produced this can be even more of a problem. Some sources suggest that acid production depends on oxygen in air reaching the tails, and that one way to prevent this is to flood the tailing pond permanently on abandonment. To maintain water in such ponds, impervious retaining embankments have to be built and these can be costly. Another approach is to design low dams under small hydraulic gradients so that seepages are limited to acceptable amounts. Whatever the approach it is necessary first to decide on acceptable limits of pollution tolerance and then to incorporate measures into the system designed to meet these limits. Pollution standards and water quality fall within the realm of a chemist; control measures can be handled by engineering design.

The water balance for a tailing pond is determined from a knowledge of surface inflow, seepages, precipitation and evaporation, and the quantity of water from the mill. Soil mechanics analysis can be used to determine seepage quantities.

Reclaiming water from tailing ponds is important for both pollution control and water conservation. Clearly, the ideal arrangement is a closed system, but very few mines ever achieve it. Studies by the U.S. Bureau of Mines indicate that nation-wide, the copper mines recirculate only up to about 50 per cent of tailing pond water, although in the northern states the quantity is higher perhaps up to 75 per cent. The problems that arise are build-up of reagents affecting the flotation process, excess precipitation into the pond, and costs of pumping and piping. Quite naturally a mine superintendent likes to use fresh water only in his mill. In theory, however, treatment of reclaimed water for mill usage ought to be no more costly than treating polluted water before it is discharged to the environment. The amount of excess water from precipitation obviously increases for shallower ponds of greater areal extent, thus influencing tailing pond design.

An assessment of seepage through the embankment and foundations is an essential step in the design of any tailing pond. If the seepage is allowed to pass through the body of the embankment, the shear strength of the materials is reduced and a flatter outside slope must be used. If seepage exits on the face or downstream of the toe, and the hydrostatic pressures exceed the weight of the soil above, there is a danger that piping may develop. This can lead to progressive backward erosion and subsequent collapse. This mechanism is potentially very dangerous and has been a major factor contributing to the failure of earth dams. Particular care must be taken to prevent seepage and piping along any culvert that may be located through or under the embankment. Installing culverts and piping through embankments is not good practice.

The flow of water and piezometric pressures in a porous media may be evaluated using a flow net. This is a grid formed by the intersection of two sets of orthogonal lines. One set of lines, defined as flow lines, represents the loci of seepage flow through the soil. The other set, defined as equipotential lines, represents the loci of points having the same pressure head; these are the piezometric contours. Typical flow nets are shown on Figure 9. Flow nets may be developed using graphical procedures, electric analogs, models, or finite element or finite difference methods; details are given in the CANMET Manual (1977). The rate of seepage through an embankment can be calculated using the flow net, coefficients of permeability and the seepage head.

The quantity of seepage that escapes from a tailing pond depends very much on the nature of the foundation soils, on details of embankment construction, and on the method of deposition of the tails. The principle of seepage flow is illustrated on Figure 10a. Clearly, however, if water ponds against a pervious embankment as in Figure 10b the water escapes through the embankment, and the rate depends upon the hydraulic gradient and the embankment permeability. Alternatively, if slimes are beached against the embankment and water is ponded against an impervious barrier such as a drumlin of till at the opposite end of the pond, the seepage path is downward through the tails, as on Figure 10c. In this latter case if the foundations are a more pervious outwash sand and gravel, the seepage after an initial blinding layer has been placed can be calculated simply from the equation of Figure 10a and is of the order of 3×10^{-5} U.S. gal/min per sq. ft. area of pond (850 U.S. gals/min per sq. mile); the hydraulic gradient is slightly greater than unity and the coefficient of permeability of the slimes is assumed to be 2 x 10^{-6} cm/sec. This is a reasonable value for the overflow fines from cycloned tails based on experience at copper mine operations in British Columbia. The time required for water to seep through the tails depends on the velocity and the length of the drainage path. Thus, in the above example of downward vertical seepage the transit time for vertical seepage through the tails would be 176 x H days, where H is the thickness in feet of the tails. If the foundations are of uniformly lower permeability than the tails, till or compressed peat for example, the seepage will be less. The seepage is controlled by the rate at which water can flow through the foundation soil, tails, or the embankment, as in Figure 10d; in this case the use of a flow net analysis might be appropriate. If for example the area underlying the pond were uniformly covered by a thick deposit of either peat, silty clay or clay till, whose coefficient of permeability might be about 1 x 10^{-7} cm/sec, the seepage out of the pond would be at least an order of magnitude less, say about 50 U.S. gals/min per sq. mile of pond area. These seepage figures should be used only to indicate the order of magnitude of the quantities. Because of the extreme variation in permeability of glacial deposits, more precise estimates could only be made on a site-specific basis.

An interesting analytical study of seepage through tailing embankments has been reported by Mittal and Morgenstern (1976) based on analyses of the large tailing embankment for the Bethlehem Copper Mine in British Columbia. The tailing embankment is constructed of waste rock, and tailing sands are deposited directly against the inside face. The embankment is about 5,000 ft. long built across a valley, and has a maximum height of about 300 ft. The quantity of seepage was calculated to be 135 U.S. gal per min which is quite low, but compares very closely with measurements of discharge collected downstream of the impoundment. It is pointed out by the authors that this low rate of seepage is unlikely to cause any serious problem, and thus it would have been difficult to justify the cost of an impervious seal in this case. If on the other hand tails had been discharged from the head of the valley so that free water ponded permanently against the face of the rockfill embankment, the seepage would have much higher and a seal might have been needed. This case illustrates how the method of operation can have a significant influence on the rate of egress of water and hence, on the design requirements for seepage control.

If the studies indicate that seepage pressures through the embankment should be reduced to improve stability, it is usual to provide drains beneath the downstream slope. The drainage system may consist of granular blankets, strip drains, or of drainage pipes. The type of drains will depend on the availability of suitable drainage materials, potential seepage volume, and foundation conditions. Drains will control piping and seepage from the slope and will allow steeper slopes and less embankment fill. In addition, they will negate the possibility of freezing of the slope, which impedes seepage, and they will minimize the development of ice lenses and subsequent surface sloughing during the spring thaw.

If perforated pipe drains are used they must be designed to withstand the total vertical load. The perforations should be placed down and the perforation diameter should not exceed 0.5 of the 85 per cent finer size (D_{85}) of the surrounding drainage soil. Since pipe drains cannot be repaired, they should not be used where moderate to high settlement could damage them. Also, the material with which the pipes are constructed must preclude any possibility of deterioration by corrosion.

Blanket or strip drains are laid down prior to placement of the embankment. If the volume of seepage is expected to be moderate to heavy, blanket drains are preferable. If the potential seepage is small, strip drains of pervious material may be sufficient. If the foundation is a source of seepage, caused by artesian conditions or consolidation drainage due to increasing the height and weight of the embankment, the drainage layers must be designed to carry this volume of water.

The thicknesses of the drainage layers will be a function of the seepage volume and soil gradation in the embankment and the foundation. They should never be less than 12 inches and preferably should be at least 36 inches thick. The final embankment should incorporate a coarse toe drain to ensure free drainage and to control piping.

Chemical tests should be performed on the embankment and drain materials, and the seepage water to ensure compatibility; the drains should not comprise carbonate rocks if the seepage water is acidic.

The drainage layers must be carefully designed and constructed if they are to function satisfactorily on a long term basis. Their capacity should be overdesigned in the event that leakage develops. Where the embankment contains zones of material having significantly different gradation, or where the gradations of the foundation and embankment materials differ markedly, the zones of different material must be separated by filter zones to prevent piping and subsequent subsurface erosion. The filter must meet two requirements; it must be more permeable than the adjacent finer soil that it is protecting so that it will drain freely, and it must have a gradation designed to prevent passage of the soil particles into the drainage layer. Particular care must be taken that segregation does not occur during construction. Details of filter design criteria can be found in standard soil mechanics textbooks, or in the CANMET Manual (1977).

Where the gradation differences are great, two or more filter layers may be required to meet filter criteria.

Where the embankment is constructed using the downstream method it may be necessary to seal the inside face with an impervious membrane or zone of fill to reduce seepage from the pond through the dyke. It is preferable to deposit the slimes on the inside face rather than to deposit them in the upstream area of the pond with the pond water directly against the inside face of the retaining embankment. In the latter case a flatter inside face is required to allow for the potential of suddenly drawing down the pond water level.

If artesian pressures develop in the foundation or below the toe of the embankment there is a danger of piping and instability developing. This problem can usually be controlled by the installation of pressure relief wells in the foundation located at the toe of the dyke. The spacing, depth and design of the wells is dictated by the soil stratification, permeability and water pressures. Experience documented by the U.S. Army Corps of Engineers (1963) for hydro and storage dams is invaluable in determining design and construction requirements. Alternatively, it may be possible to control potential erosion and piping by construction of weighted filters extending outward from the downstream toe.

The amount of seepage through the foundations can be reduced by placing an impervious blanket on the inside of the pond. This may consist of slimes or of fine grained overburden. If the pervious soils in the foundation are shallow, a core trench backfilled with impervious soil may be used. Where the dam is located on an impervious foundation or an impervious geologic barrier outcrops downstream a small collection dam may be constructed downstream, and the water that seeps through the embankment may be returned to the tailing pond by pumping.

Where the tailing pond is located over thick pervious deposits, positive pollution control can be accomplished by developing a system of injection and pumping wells downstream of the retaining embankment. The injection wells are located downstream of the pumping wells. Uncontaminated water is pumped into the injection wells and the contaminated water is extracted from the pumping wells. By maintaining the piezometric elevations at the injection wells moderately above those at the pumping wells, a hydraulic barrier will develop which will cut off the escape of tailing pond seepage. Methods of seepage control are summarized on Figure 11.

Grout curtains have been used partially to intercept seepage on many earth dams. Because of the variable success and usually high cost, this method of seepage control is not recommended for tailing disposal. Other methods that may warrant consideration for special conditions include sheet pile cut-offs or slurry trenches. The bentonite-slurry trench method of producing an impervious cutoff wall requires excavation of a trench to bedrock or to an underlying impervious soil deposit. The material on each side of the trench is retained in position by stabilizing the excavation with a heavy slurry of bentonite, which is poured into the trench as excavation proceeds. When the trench excavation has progressed sufficiently so that backfilling operations will not interfere with excavation, the bentonite slurry is progressively displaced with impervious fill material to form the cutoff wall. Slurry trench walls are dug with a large clam shell machine and backfilled with either a soil-bentonite or cement-bentonite mixture. The maximum depth of installation is generally limited to about 90 ft. by the capacity of the digging machines. An example of a cement-bentonite slurry trench wall used as a foundation cut-off beneath a tailing embankment was described in the Engineering News-Record (1976). The project was at Cleveland Cliffs Iron Company's Tilden Mine near Marquette in the upper peninsula of Michigan. Costs for this wall, 2 ft. thick, were quoted as $3.64 per sq. ft. for walls up to 40 ft. deep and $4.10 per sq. ft. up to 80 ft. deep.

Increasing use is now made of plastic liners to seal tailing ponds, particularly if naturally occurring impervious clay soils are not readily available. A useful study on various types of liners including natural and treated soils, asphalt treatments, and plastic materials was carried out by the U.S. Environmental Protection Agency (1975). The report discussed costs, methods of installation and deterioration of plastic membranes. If plastic or bituminous liners are designed as stabilizing features in a retaining embankment, great care has to be taken in installation and protection against long term deterioration; excessive settlement can stretch and tear a membrane, likewise clumsy handling with machinery can puncture a plastic liner. The recent failure of the Elbe

Seiten Canal in West Germany was caused by fracture of an impervious bituminous membrane resulting from excessive differential settlement. The canal had been in operation for less than 6 months, and the total damage was estimated at $100 M. The canal is being repaired with an expensive non-tearable polyvinyl chloride membrane called folie. However, the cost for even 5 mm thick material is about $20 per sq. m.

An interesting use of a PVC membrane in a tailing disposal system in Minnesota is at the Minorca Taconite mine owned by Inland Steel Mining Company. The use of a plastic membrane in this instance is intended to be to limit the amount of seepage, and the design has been described by Gubbe (1977).

Whatever method of seepage control is adopted it is important to monitor leakage and, more particularly for stability purposes, to measure pore water pressures. This can be carried quite simply by the installation and reading of piezometers, a typical example of an open standpipe instrument being shown on Figure 12; other types such as pneumatic and electrical piezometers can be used. Readings should be transmitted promptly to the designer for review and assessment on a regular basis, and if any unusual change in the readings occurs he should be notified immediately. This may seem an obvious statement to make, but readings often accumulate in files without interpretation, and in some instances failures have occurred through neglecting to notify a designer. He after all is the person who appreciates fully the significance of the readings. Special precautions should be taken to protect instrumentation from vandalism and from damage by construction equipment.

Of special interest in the copper-nickel region of Minnesota is the construction of tailing ponds on peat. It is quite feasible to found embankments on peat, but organic soils should not be included in the body of the embankment. Peat is a highly deformable material even after consolidation under high loads. The permeability of fibrous peat in its natural state can be quite high in an unloaded condition, but permeability decreases dramatically under loads such as that imposed beneath the base of an embankment. In general, conventional methods of seepage control are acceptable but with certain precautions. Because of the high deformability of peat, rigid vertical cut-offs and pressure relief wells can be fractured by load being transferred from the embankment to the cut-off or wells. Also, the use of thin impervous clay zones to control seepage through embankments founded on peat should be avoided unless the clay is well graded, such as a till; if cracking occurs in the impervious zones as a result of excessive settlements of the peat, well graded material are self-healing and the cracks fill up with soil.

In constructing embankments on peat, trees and shrubs should be cleared from the base area, but the surface layer of live peat need not be removed. The initial fill is usually placed by end dumping to form a mat on which equipment can operate. Drainage ditches can be located around the area to lower the water table and to assist in stabilization of the peat. Alternatively, construction can be facilitated by working in the winter, removing the snow cover to allow the peat to freeze. If excavation below the water table is to be carried out this is easier during the winter. Providing operations are carried out continuously working the soil all the time, fill can be placed and lightly compacted down to temperatures as low as 10 degrees F. In general, where stage construction procedures are used, there is less chance of foundation failure in the peat if the first stage is built when the peat is frozen.

An entirely different method of dealing with peat is to either remove it by blasting or to displace it by deliberately loading it at a sufficiently high rate as to induce instability. These techniques are common in highway construction practice, but with tailing embankments built slowly they may be unnecessary.

The benefits of permanent seepage control from a water quality standpoint, employing sealed ponds that do not allow for drainage and consolidation of the tails, should be weighed against the loss in land usage resulting from the softness of tails in the pond. Downward seepage, on the other hand, promotes rapid consolidation of the tails. Thus, the ground becomes firm and useable. Moreover, in this condition it easily becomes capable of supporting a layer of waste rock, about 3 to 5 ft. thick, which is a positive, permanent and relatively maintenance-free method of fugitive dust control, see Appendix B.

To meet quality standards the underseepage can be minimized by using a number of small ponds in sequence rather than a single large pond. This would result in a need for extra retaining embankments at an added cost, but it may be cheaper than using elaborate seepage control measures. Once an allowable level of seepage egress has been determined based on water quality standards the maximum size of pond can be calculated using an estimate of the likely underseepage. Such an approach, however, is very site-specific and a generalized approach could be totally misleading in practice, but to illustrate the concept an example is given in Section 5d.

e) Decant Systems

The method of decanting effluent water or reclaiming it from a pond can have an important influence on the safety of the tailing embankment.

The traditional method is to use a concrete decant tower raised in sections, with a discharge pipe located beneath the base of the embankment. The height of the inlet on the decant tower can be adjusted to provide adequate depth of water for clarification purposes. Apart from possible freezing of the decant pipes the method is simple to operate. They do, however, have some serious drawbacks, and many failures have occurred in the past. Pipes installed through or at the base of embankments represent potential paths for erosion of material along the outside surface of the pipe; in extreme cases this can lead to a failure of the embankment by the backward erosion process known as piping. The danger of this happening is more severe if the embankment is constructed of uniform sized fine material such as tailing sand. Structural failure of towers and decant pipes can also occur, because of differential shear movements in the tails and differential settlements.

A more favourable method of decanting water is by means of a floating pumphouse or syphon pipes located over the crest of the embankment. A pump requires a source of electrical energy, which can fail. A syphon although cheap to install, loses its prime if the pond water level falls too low and at low flows in cold weather freezing can occur.

Simple rock lined weirs notched into the crest of the embankment are a simple and fool proof arrangement particularly suitable for use in cold weather. They are reconstructed on the crest of the embankment or on the abutment with each raise in height of the impoundment. This is generally quite economical except for high embankments. Weirs also provide excellent permanent decants for the tailing pond on abandonment. They can be incorporated of course, in an embankment on completion to replace a pump or syphon system.

f) Use of Tailing Material and Tailing Disposal Areas

A positive use of tailing material, which is at the same time a method of disposal, is mine backfill. If abandoned open pits are in close proximity to current milling operations, tails can be dumped into the pit. Provided ground water pollution hazards do not exist, this is a feasible method of reclaming the area occupied by the abandoned open pit. Because of the bulking of tailing material, however, the tonnage of tails that can be storage will always be less than the tonnage of waste rock and ore removed from the pit. Whether all the tails can be stored depends on the output of the current operation. If the tails can drain and consolidate, and if the water can be reclaimed, the area on abandonment may be usable land. This approach could be a positive use of a tailing disposal area where nearby mines are operated in sequence.

Tailing sand has been used as backfill and underground mine support on a number of projects, where it has had advantages in providing improved ground control during extraction, and in reducing fire hazard. The prime disadvantage, however, is the sterilization of material of possible future mineral potential. In this respect open pit disposal is less of a disadvantage. Tailing sand is used where underground mining employs cut and fill or square-set stoping methods. Careful soil mechanics studies can lead to the utilization of tailing sand as a support material of quite acceptable strength. Desliming of the tailing mill product by cycloning is almost invariably required to yield a backfill having adequate strength and drainage properties. For this reason only about 50 per cent of tailing material is usable as backfill. It is usual to mix a cementing material with the tailing sand prior to placing it as a backfill. This may be portland cement and, or, in some instances, pulverized smelter slag. The copper mining operation at Mount Isa Mines in Queensland, Australia is an example of the use of smelter slag, tails and quarried rockfill, as mine backfill support. The cementing agent increases the strength of the backfill and hence, its supporting capacity. In considering the use of tailing sand as underground mine backfill, the overall tailing disposal system has to be considered quite carefully; the tailing slimes left behind still need to be disposed. If the only readily available retaining embankment material happens to be tailing sand, its entire use as mine backfill may be uneconomical in the overall mining operation.

Tailing usage as backfill in civil engineering projects has been given considerable study, but at the present time it is not common. The reasons are mainly a matter of economics. Mining and milling operations are generally far from potential markets for backfill and thus, haulage costs become prohibitive. In addition the volumes of tailing materials produced far exceeds local demands for fill. Finally, the engineering characteristics such as fineness and poor drainability, together with undesirable chemical properties often make tails ill-suited for general civil engineering purposes. Despite these shortcomings, tails have found usage in highway and railroad fills and base courses. An example is the large-scale usage for a section of highway in Idaho, where a tailing pond happened to be conveniently located and no other suitable borrow fill was nearby, see Pettibone and Kealy (1963). It is understood that taconite tails have been used in Minnesota for highway construction.

Other possible uses of mine tails are as processed materials, chiefly building stones, glass and ceramics; this is principally in the research and development stage and will not be discussed herein, other than to say that again potential supply is likely to far exceed demand.

Little study has been given to usage of land on abandonment of a tailing disposal area. In principal, however, once consolidation of the tails is complete its use from a foundation standpoint is in no way inferior to any other land underlain by firm to soft ground. In some ways it is superior, because the knowledge of the existing ground conditions is probably much better than a comparable area of natural ground. An operating mine could well find the use of a conveniently located abandoned tailing pond acceptable for warehouses, equipment storage or other moderately lightly loaded buildings. In the Minnesota copper-nickel region, however, unless the pressure for land use changes dramatically it is difficult to visualize the land having more than amenity value once mining ceases. Further consideration of this is given in the section of the report dealing with reclamation.

g) Construction Control and Regulatory Inspection

Clearly, careful control of construction in the field by inspection and supervision is as essential to success as office design and analysis; this applies to whatever method of tailing embankment design is adopted. An engineering factor of safety in a design analysis has no meaning in reality unless the conditions assumed in the design are realized in the field during construction and operation. This fact cannot be over-emphasized. It should underlie all regulatory procedures for inspection and control of tailing impoundments. Accurate records of construction work should be kept. Rather than risk misinterpretation or an oversight, it is advisable for the engineer responsible for the design to make regular visits to the field to observe the work in progress. Such visits are required more frequently in the early phases of construction, when key factors such as foundation excavations are underway; these may well reveal conditions not indicated by borings carried out during the preliminary site investigations. When the construction procedure becomes routine and well understood by the mine operators, the visits of the design engineer can be less frequent, perhaps 2 or 3 times a year at his discretion.

The mine personnel must be familiar with the intent of the designer. If any doubt arises they should call in the designer. In reality construction personnel cannot divorce themselves from the responsibility for a safe structure. A designer cannot be held entirely responsible for a failure arising from unforseen circumstances during construction if the information is not relayed to him at the time. If he is to be so held responsible, he has no option other than to be on site full time, in which he might as well assume control of construction. This point is laboured because of the great variety of soil and ground conditions that can be met in geotechnical engineering. Break down of communication between design and construction personnel is probably at the root of by far the majority of failures in earthwork engineering.

Mine personnel should record such facts as grain size distribution of sands, percentage recovery of sand from the tailing, permeability and density of the sand in place, and make regular observations of instrumentation such as piezometric levels. Two critical items are the rate of rise of the pond and the rate of rise of the crest of the embankment. Because the embankment construction is usually a slow ongoing process throughout the life of the mine, the design may require review from time to time; this may lead to revision of construction procedures perhaps resorting to the use of borrow material to supplement tailing sand. Clearly, changes in the orebody, the milling operation, adjustments to the volume of reclaim water can all lead to changes in the required construction procedures. This requires that the design be flexible. In contrast to a conventional water storage dam, which is not put into use until construction is completed, a tailing embankment is usually constructed and put into operation simultaneously. Only in rare circumstances, perhaps where the embankment is built entirely of borrow material, is the construction completed prior to operation. This latter approach may lead to a low operating cost, but it involves a high initial outlay of capital.

4.0 COSTS OF TAILING EMBANKMENTS AND SEEPAGE CONTROLS

Engineering construction costs are very difficult to assess on a generalized basis. Each site and each installation has costs peculiar to the location and the techniques adopted. Haulage, loading and any processing required is a significant and sometimes the major part of fill costs. Techniques requiring a specialist contractor to perform the work, perhaps using special equipment not available locally, can involve very large fixed mobilization and demobilization costs; the cost of such work therefore varies widely on a unit basis depending on the total quantity involved.

The cheapest approach to embankment building is to use tailing sand. At Brenda Copper mine in British Columbia, the cost of the embankment, which will ultimately utilize about 32 million cu. yds. of tailing sand, is said to be not more than 5 cents/cu. yd. The tails flow under gravity from the mill to a cyclone station located on one abutment, and from there are distributed hydraulically to the embankment. Double stage cyclones are used, and the first stage works by gravity alone; the second stage consumes some energy. The only real costs are labour, the cyclone equipment and some power to operate the second stage cyclone, and some dozing of cells. Compaction is not used on the sand. Waste rock was used in a starter dam, and some local impervious borrow in an upstream blanket, and for filters and underdrains. The cost of 5 cents/cu. yd. applies only to the tailing sand, and it is based on costs of equipment about 7 years ago.

Very often a mining company can construct an embankment at significantly lower costs than by employing an outside contractor. Apart from contractors profit, the mining company may be able to build an embankment with equipment that is being under-utilized elsewhere in the mining operation. Also differences in union agreements between inside and outside construction workers can have a significant influence on labour costs; again mine labour may be utilized say indoors in inclement weather which is a saving compared to what it would cost a contractor.

Where tails are used for construction it is reasonable to assume that total embankment costs will be about 15 to 20 cents per cu. yd. including small starter dams and underdrains, personnel and dozing equipment. These costs are exclusive of tailing transportation costs discussed elsewhere in this report, and any measures for foundation seepage cutoffs.

If tailing sand is not the main element in the embankment construction, fill has to be excavated, hauled, loaded, processed and placed. If the fill is waste rock from mine excavations, it would normally be disposed of as close as possible to the mine site; hence the only real extra costs incurred by using it in tailing embankments are overhaul, processing and placing. The overhall is the extra distance from the waste dump site nearest to the mine to the tailing disposal site. Processing is any sorting of unwanted oversized rock pieces by passing the material over a large screening grid, commonly called a grizzly. Placing is any extra work involved in spreading by dozer and compaction over and above the work involved in dumping the rock, which is cost that is entailed anyway. Loading should not be a cost charged to tailing disposal since this cost is unavoidable in disposing of the rock waste.

If borrow material is required, borrow pits must be investigated, and a determination made of whether processing is required. Excavation costs have to be estimated based on whether the material can be dug by loaders above the water table or by dragline below, or whether scrapers can be used. If a gravel has an excessive fines content, it can be very difficult to handle if it has to be excavated from below the water table; the material is sloppy and refuses to drain easily. Scrapers are low in loading costs, but their haul distance is restricted to about one mile. If loaders alone cannot dig material, ripping may be required and in the extreme, with very hard till, blasting may have to be resorted to. Normally, blasting would be avoided because of the prohibitive cost, and alternative sources of borrow material would be investigated. Finally, haul roads must be laid out and the haul distances and grades determined. Haulage cycles can then be calculated and hourly production figures established; these will control equipment selection.

The relationship between bank, loose, and embankment yardages has to be determined before take-off of quantities can be made. Bank cubic yards refers to the volume of the material in place in the borrow pit and which has to be loaded. Loose cubic yards is the volume of the material as hauled from borrow pit to the site. Embankment cubic yards refers to the volume of the material placed and compacted in the embankment, and it is the figure on which payment is usually based. The relationship between bank, loose and embankment unit volumes, assuming that no processing is carried out, can vary between 1: 1.16: 0.88, respectively, for clean, wet sand and gravel, to 1: 1.33: 1.00 for a dense clay till. Processing may consist of scalping, screening, washing, blending or any combination of these processes, which clearly can affect very considerably the relationship between the yardages. Impervious borrow material may require moisture adjustment to bring it to the optimum condition for compaction.

Scheduling of embankment construction has to be planned so that all excavated material can be hauled directly from the point of excavation to the tailing embankment and used without being placed in storage stockpiles, since this would be an expensive extra handling item. The thickness that each layer of fill can be placed in an embankment can vary between 6 inches for cohesive material used in impervious zones to 24 inches for granular pervious materials. The thickness also varies depending on the type of compactor used and on the number of passes of the compactor. Fill spreading on the embankment may be done either with crawler or wheel dozers, with blades mounted on the front of compactors, or with motor graders. The amount of spreading needed depends on the type of haulage equipment used.

Compaction of the embankment material can be specified by one of two methods. In the end result specification, the required density and moisture content is called for and the responsibility is placed on the contractor to determine the type of compactor and the number of passes of the equipment that are needed to achieve this result. In the method specification, the type of compactor, and the number of passes are specified and the designer accepts responsibility for the result. Compaction equipment varies from heavy rubber tired rollers and sheeps foot rollers for cohesive soils to lighter vibratory rollers, grid rollers and crawler tractors for granular fills.

All equipment has what is termed an equipment availability of perhaps 70 to 80 per cent of the total possible time, the balance being used for servicing and repair. Additional equipment must also be provided for haul road building and maintenance.

Finally, waste rock embankments will probably need filter zones which usually require very careful screening and blending of either sand and gravel or rock fines or possibly crushing of rock waste.

The foregoing discussion is presented to show some of the variables that can enter into embankment fill costs. Clearly, only a range of cost is strictly applicable. Nevertheless, to guide the MEQB for comparison purposes in the planning process the following costs are given based on 1976 dollars.

Items for haulage are broken down so that construction using borrow fills from remote sources can be assessed.

i) Excavation and loading of sand and gravel, or till as borrow material can be assumed to cost $1.30 per cu. yd., assuming that the material can be dug and loaded by a front end loader or dragline. If ripping, or light blasting is needed to remove hard till, the excavated cost should be taken as $1.80 per cu. yd.

ii) Because waste rock from the mine is a necessary excavation whether the material is used or not, the cost to tailing disposal should be assumed to be zero, unless processing or special blasting techniques are used to yield a material of a more suitable gradation for embankment construction.

iii) Haulage costs from borrow area to embankment site or overhaul of waste rock should be taken at $0.50 per cu. yd. per mile of haul.

iv) Placement and compaction of fill in layers in a tailing embankment should be taken as:-

Waste rock	- $0.50/cu yd	(assumes no compaction needed)
Sand & gravel	- $0.30/cu yd	(assumes only light compaction by dozing and spreading)
Cohesive clay till	- $0.80/cu yd	(assumes good compaction to produce impervious seal)

The above costs are based on the assumption that the work would be of high quality performed by a skilled independent earthwork contractor. Mobilization of equipment, camp costs, and so on, are inclusive, but it is assumed that a sizeable piece of embankment construction not less than about 1.0 million cu. yds. would be undertaken.

Methods of seepage control can be costed using the above figures if the cutoff is to be a shallow trench backfilled with compacted till, or if an upstream blanket is employed. A specialized technique such as a bentonite slurry trench cutoff may be used. For pollution control purposes such walls can be quite thin, perhaps no thicker than the width of trench in which a digger can operate, say to 2 to 3 ft. For planning purposes where deep cutoffs are envisaged a cost of $5.00 per vertical sq. ft. of wall should be used.

The cost of plastic liners for impervious seals depends upon the type of material used, the thickness of the membrane, and the method of placement. Useful figures based on 1972-73 costs are given in the EPA report (1975). For planning purposes a PVC membrane can be assumed to cost

$0.010 per sq. ft. per mil thickness. Thus, if a membrane 20 mils thick is used the cost of the sheet would be $1.80 per sq. yd. The cost of Hypalon, which is more durable, should be taken at $0.014 per sq. ft. per mil thickness. In addition to the cost of the plastic, an installation cost of at least $0.50 per sq. yd. should be included. If the membrane is to function as a truly impervious seal, joints have to be sealed and it has to be protected with about a 6 to 12 inch layer of fine soil; for proper functioning very careful handling and construction is needed and sufficient monies should be allowed for in any contract bid. Normally the lining would be used only to seal the inside face of the embankment, but if the pond were underlain by pervious deposits or the tails were designed to remain permanently flooded, the lining might be called for over the whole area.

5. EXAMPLES OF TAILING POND LAYOUTS FOR THE CU-NI PROJECT

A "cookbook" for the design of tailing disposal systems will in most instances be misleading and in some cases dangerous; designing such a facility is a specialized engineering and mining function. The preceding sections are not intended to be a step-by-step approach to the design of tailing ponds. Rather they are intended to help formulate useful guidelines for regulatory agencies. They are not intended to replace engineering know-how, but to help regulatory agencies to be able to judge whether a proper engineering approach has been undertaken in any particular case.

In addition the document ought to be useful for planning and economic trade-off studies. In this respect, a planner ought to be able to use the document to produce a number of hypothetical designs for a particular mine scenario. The following examples illustrate how such designs might be undertaken.

a) Water Hen Creek Basin

This area is at the southern end of the belt of mineralization, south of the St. Louis River and in the drumlin province. An open pit mine is considered yielding 20 x 10^6 metric tons per year, and producing 96.82 per cent tails for a period of 30 years. A tailing disposal system is examined in Class I region in the upper reaches of the South Branch of Water Hen Creek.

It seems likely that a pond can be developed by construction of dyke segments spanning the gaps between drumlins as shown by the outline ABCDDE in Figure 13. It also appears that a single large pond could be built within this area. It will be assumed that the criterion has been set that no seepage should emerge downstream of the pond into Water Hen Creek Basin. Therefore, the arms of the embankment BC, CD, DE, and EA should be impervious; this might be achieved by constructing these sections of the embankment from impervious glacial till. It can be assumed that the drumlins are composed of impervious till, but this would need to be verified by site investigation if a specific proposal is made. Seepage could be allowed to escape through the arm AB of the embankment, and it could be constructed from tailing sand. This section of the embankment is at the upstream end of the basin, and underseepage would tend to migrate downslope toward the open pit where it could be collected and either reused in the mill, or treated and discharged into the St. Louis River. Peat encountered in the foundations of the embankments would be excavated along the short portion of the arm CD of the embankment, but it would be left in place along AB and consolidated under the gradually increasing weight of the embankment constructed of the tailing sand.

The first step in the analysis of such a scheme would be to calculate the storage volume that is required.

The total tails produced would be 581 x 10^6 metric tons, and assuming that the dry density of the tails in 90 pcf, the total storage volume required would be 14.22 x 10^9 cu. ft. Allowing for about a 10 per cent contingency we should plan for a storage volume of 15.5 x 10^9 cu. ft.

To calculate the available storage volume within the area of the pond the prismoidal formula can be used. This is explained by reference to Figure 14. A prismoid is a solid whose ends are parallel and whose sides are plane or warped surfaces. It can be shown by means of calculus that the correct volume of a prismoid is:-

$$V = L/6 \ (A_1 + 4A_m + A_2)$$

where:-

L = the distance between the two parallel bases whose areas are A_1 and A_2

A_m = a section midway between the two end bases and parallel to them.

A_m is not an average of A_1 and A_2, although strictly each of its linear dimensions is an average of the corresponding dimensions of A_1 and A_2. In the following usage it is accurate enough to take the parallel bases A_1 and A_2, and the midway section Am as the horizontal areas enclosed by the pond at contour intervals of 10 ft. The prismoidal formula can be applied repeatedly to successive areas provided that the total number of areas are odd numbers. The reason for this will be apparent in the following calculation.

A maximum pond thickness of 80 ft. might provide adequate storage, and so the calculation will be made between the minimum elevation of 1510 and a maximum elevation 1590. The areas enclosed within the pond at the successive contour intervals are measured using a planimeter, and the calculation is laid out in the table below. The scale of the plan in Figure 13 is 1:62500, and for the particular planimeter that we used the factor that the planimeter readings have to be multiplied by is 67.275 in order to yield the areas in square feet.

Elevation ft.	Planimeter Reading	Area x 10^4 sq. ft.	A_1 and A_5	$4A_m$	2 2A3 4
1950	353	$A_1 = 23745$	23745		
1980	343	$A_{m_2}^{1} = 23078$		92312	
1570	323	$A_2 = 21732$			43464
1560	300	$A_{m_3}^{2} = 20182$		80728	
1550	249	$A_3 = 16749$			33498
1540	188	$A_{m_4}^{3} = 12648$		50592	
1530	86	$A_4 = 5780$			11560
1520	31	$A_{m_5}^{4} = 2088$		8352	
1510	4	$A_5 = 269$	269		

$$\Sigma = 24014 + 231984 + 88522$$
$$= 344520 \times 10^4$$

Now the prismoidal formula calculates the volumes between A_1 to A_2, A_2 to A_3, A_3 to A_4, and A_4 to A_5, hence the length L in the formula is twice the contour interval, i.e. 20 ft.

Applying the formula repeatedly the total volume is:-

$$V = L/6\ (A_1 + 4A_{m_2}^{1} + A_2 + A_2 + 4A_{m_3}^{2} + A_3 + A_3 + 4A_{m_4}^{3} + A_4 + A_4 + 4A_{m_5}^{4} + A_5)$$

$$= L/6\ (A_1 + 4A_{m_2}^{1} + 2A_2 + 4A_{m_3}^{2} + 2A_3 + 4A_{m_4}^{3} + 2A_4 + 4A_{m_5}^{4} + A_5)$$

$$= 20/6 \times 344520 \times 10^4 = 11.484 \times 10^9 \text{ cu. ft.}$$

The volume that we have calculated falls considerably short of the required volume, hence for the same area of pond the maximum elevation will have to be higher. We might try another 20 ft. thickness of pond up to elevation 1610. The same procedure is used again applying the prismoidal formula to calculate the additional volume from elevation 1590 to 1610. The calculation for the additional volume is shown in the table below.

Elevation	Planimeter Reading	Area x 10^4 sq. ft.
1610	370	$A_o = 24892$
1600	366	$A_{m_1}o = 24623$
1590	353	$A_1 = 23745$

$$\text{Additional volume } V = L/2\ (A_o + 4A_{m_1}o + A_1)$$

$$= 20/6\ (24892 + 4 \times 24623 + 23745) \times 10^4$$

$$= 20/6 \times 147129 \times 10^4 = 4.904 \times 10^9 \text{ cu. ft.}$$

The total volume that could be stored would be $(11.484 + 4.904) \times 10^9$

$= 16,388 \times 10^9$ cu. ft.

This is now a little more storage than we need, and a pond to elevation 1605 might be adequate. However, we need freeboard, and therefore embankments built to elevation 1610 would be satisfactory. This would yield an average thickness of tails of about 61 ft.

To determine the volume of embankment fill that is required we must assume cross-sections for both the part built from tailing sand, and that built from impervious till. Arm AB of the embankment built from tailing sand will lie on weak peat foundations, which we have assumed will be left in place and consolidated under the weight of the embankment as it is gradually raised. To ensure stability of the embankment the outside face will need to be quite flat, and a slope of 4.0 horizontal on 1.0 vertical has been taken. Stability analyses would be needed to verify this slope angle in a specific proposal. The other arms of the embankment might be constructed of compacted glacial till. Assuming that weak material is excavated from the foundations, the outside slope would be 2.0 horizontal on 1.0 vertical. The embankment cross-sections are shown on Figure 15.

By estimating and measuring the equivalent lengths of the embankments and using cross-sectional areas calculated from Figure 15 we have estimated the following volumes of embankment fill:-

i) Arm AB, tailing sand, volume = 9.3×10^6 cu. yds.

ii) Arms BCDEA, compacted glacial till, volume = 9.5×10^6 cu. yds.

If we assume that tailing sand costs $0.20 per cu. yd., Section 4.0 of this report, then the cost of Arm AB of the embankment would be about $1.86 million. The glacial till could be won by excavating those portions of the drumlins that lie within the area of the pond. Thus, haul distances would be short, and the cost might be say 20 cents per cu. yd. on average. The cost of the till therefore is estimated to be $2.30 per cu. yd., and the cost of the Arms BCDEA of the embankment would be about $21.8 million. The total cost of embankment would be $23.7 million.

Clearly, the use of compacted glacial till is expensive, especially since large portions of the embankment utilize the existing drumlin ridge topography. The cost of the tailing disposal pond would be 4.08 cents per ton on average.

An alternative approach might be to use waste rock from the mine and an impervious plastic seal for Arms BCDEA of the embankment. In this case the outside slope of the embankment could be steepened to 1.5 horizontal on 1.0 vertical, and the volume would decrease to 8.3 x 10^6 cu. yds. Assuming that some processing of the rock waste is needed to remove oversize material, and the cost is \$0.20 per cu. yd., and assuming a mean haul distance of 3 miles at \$0.50 per mile per cu. yd., the cost of the rock waste would be \$1.70 per cu. yd. The cost of the rock waste embankment therefore would be \$14.1 million. If 4.725 x 10^6 sq. ft. of 20 mil thick PVC membrane were used to seal the embankment at a cost of \$0.25 per sq. ft., including filter material and installation, an additional cost of \$1.2 million would be incurred. The total cost of the alternative sealed rock waste embankment would therefore be \$15.3 million, and a saving of \$6.5 million is indicated compared to the use of glacial till. A hidden saving would also arise from the reduction in land area used for the disposal of additional mine rock.

b) Big Lake

This example is put forward to illustrate the possible use of a ready made disposal basin. The site is fairly remote being 7 miles from the belt of mineralization; it was included in Class I area, however, because that lake lies within the Partridge River Basin and within the arc of terminal moraine ridges. The other lakes in the vicinity are not favourably located with respect to hydrological and geotechnical features.

The surface area of Big Lake is 33.5×10^6 sq. ft. The elevation of the surface of the water is shown to be about 1,690 ft., but the depth is not indicated. Assuming, however, that the average depth is 15 ft., the storage volume just up to surface water level would be about 0.5×10^9 cu. ft. Apart from the pumping costs, this amount of storage would be cost free. Using the terminal moraine ridge bordering the SE shore of the lake and building a low retaining dyke around the NW shore, the storage volume of the area could be increased. If the lake were to be enclosed by dykes to elevation 1750 ft. an additional 2×10^9 cu. ft. of tails could be stored.

However, the SE side of the lake is close to the boundary of the Partridge River and the main branch of the St. Louis River watersheds. Therefore to avoid contamination of the St. Louis River and possibly Seven Beaver Lake, an impervious barrier might be needed along this side of Big Lake.

In view of the cost of constructing such a barrier, using either glacial till or plastic membranes, the use of Big Lake for other than a minor volume of tailing storage may be unattractive, especially if pumping costs are high.

c) Dunka River Basin

This area should prove very interesting for tailing disposal systems. We have ranked the area as Class II, because the basin contains thick deposits of outwash materials, which could lead to seepage control problems. The study by the U.S. Geological Survey suggests that the Dunka River Basin is probably an infilled pre-glacial valley, tributary to the

Embarrass River. The infilling outwash sediments, probably sand and gravel, are indicated to be up to 70 to 90 ft. in thickness based on drill hole records.

Of particular interest is the drainage from the Dunka Basin, all of which seems to exit through the narrow gap in the Giants Range in the NW corner of the area, see Figure 17, Point A; the river flows north into Birch Lake via Dunka Bay. A detailed hydrogeological investigation would reveal whether the basin lies entirely within bedrock and impervious glacial till. If this proves to be the case, the only exit is the narrow gap in the Giants Range, and a unique opportunity would exist for seepage control. By constructing a bentonite slurry trench cutoff wall in the outwash sediments in the gap at Point A, all underseepage could be arrested. Subsurface water in the level swampy ground east of the river would be virtually stationary, and the only major egress from the basin would be surface water in the river channel; this could be monitored quite accurately.

Such an arrangement might allow a number of separate tailing ponds to built in the Dunka River Basin without the need for special underseepage control measures at the ponds. Provided that the hydraulic gradient from a pond to the nearest point on the channel of the Dunka River was small, the flow of effluent to the river would be small, even though the outwash sediments are permeable. A low hydraulic gradient could be achieved either by keeping the elevation of the pond low with respect to the river, or by situating the pond as far away from the river as possible.

Tails might be disposed of by the central discharge method. A pond could be situated in the sector of land bounded by the two roads, as shown on Figure 17; with center at C, a radius of 5,000 ft., and assuming a mean slope for the discharged tailing of 4 per cent, the elevation of the point of discharge would be 1750 and the corresponding maximum thickness would be 200 ft. The volume of tails stored by this means can be calculated using the formula for the volume of a cone:-

$$V = 1/3 \Pi R^2 H$$

where:-

R = the radius of the cone and H is the height.

The available storage volume by this means would be somewhat less than 5 x 10^9 cu. ft., allowing for some loss in storage caused by the high ground in the SE segment of the cone. This is a very useful storage volume. Other areas at the south end of the Dunka River Basin might be considered also.

d) <u>South Kawishiwi River Basin</u>

This area is characterized by flat peat bog topography with ridges of exposed bedrock, and a thin covering of till. Provisional information published by the U.S. Geological Survey, Water Resources Division indicates that well yields in the area are generally low. For till and fractured bedrock over a large area it is reasonable to assume that a hydraulic conductivity of 10^{-2} ft./day would be applicable; this is equivalent to a coefficient of permeability of 3.5 x 10^{-6} cm/sec.

This information will be used to design a modular layout of ponds for a limited amount of uncontrolled foundation underseepage. We will assume that the depth of the ponds is 70 ft. and that the thickness of fractured bedrock is 200 ft., i.e. about 3 times the thickness of the tails in the ponds; below this depth the bedrock can be assumed to be tight. Embankments will be constructed of cycloned tailing sand and the coefficient of permeability is estimated to be 2 orders of magnitude greater than the till and bedrock, i.e. 1 ft./day (3.5×10^{-4} cm/sec.).

If the permeability of the foundations of the pond is about the same, or less than that of the tails in the pond, a simple flow net can be constructed for seepage through the foundations and sand embankments. Assuming the outside slope of the embankment is 3.0 horizontal on 1.0 vertical, the seepages through the embankment, Q_e, and foundations, Q_f, per foot length of perimeter embankment is then estimated from the following expressions:-

$$Q_e = \frac{hk_e}{7} \quad \text{and} \quad Q_f = \frac{0.03\ hk_e}{7}$$

where:- h = head of water in the pond

k_e = coefficient permeability of the embankment sand.

Using the parameters above, the seepage through the embankment is estimated to be 5×10^{-2} U.S. gal/min per foot length of embankment. This can be collected in ditches at the foot of the embankments and it can be regarded therefore as controlled seepage. The seepage through the foundations is estimated from the above equation to be 1.6×10^{-3} U.S.

gal/min per foot length of embankment. Now this water can emerge large distances away from the pond, and for this reason it is regarded as uncontrolled seepage. We will assume for the purpose of the example that to meet water quality standards uncontrolled seepage should not exceed 50 U.S. gals/min. in the vicinity of the ponds. The perimeter length of ponds to stay within this limitation can therefore be calculated as:-

$$50/0.0016 = 31,250 \text{ ft.},$$

and for quadrilateral shaped ponds this means that the average length of the arms of the embankment should not exceed about 8,000 ft.

The concept in this example for limiting uncontrolled seepage is that a series of smaller ponds would be operated in sequence, rather than building a single pond of large areal extent. On abandonment of each small cell the seepage would still continue, although it would tend to decay over a number of years. Therefore the size of each operating cell should be smaller than the maximum size calculated above. For this reason a mean pond size of 5,000 x 5,000 ft. has been chosen. Assuming flat lying topography and an average thickness of tails of 70 ft. the storage volume of each cell would be 65 x 10^6 cu. yds. Now 1 metric ton of crude ore needs 0.905 cu. yds. of storage, hence each cell could store 72 x 10^6 metric tons of tails. Considering an underground mine with a rated capacity of 12.35 x 10^6 metric tons per year of crude ore, producing 94.86 per cent tails over an effective operating life of 23 years, then 269 x 10^6 metric tons of tails would be produced. Hence, allowing for a small contingency, 4 cells would cater for the total production of tails. The modular layout of cells is shown on Figure 18. A cell would be constructed roughly every 6 years, and each cell could be rehabilitated on abandonment. Because of the low rate of underseepage the majority of the pond area, apart from the beach, could be kept submerged to abate the dust problem. Shortly after abandonment the ponds would have drained and consolidated sufficiently for waste rock from the mine to be placed over the tails to keep the dust down.

It will be assumed that in general foundation conditions of the embankments are good and an outside face slope at 3.0 horizontal to 1.0 vertical can be adopted. To allow for freeboard the embankments will be 75 ft. high on average. The total length of embankment from Figure 18 will be 65,200 ft., and hence the total embankment volume will be about 32 x 10^6 cu. yds. The total volume of tails produced would be 269 x 0.905 = 244 x 10^6 cu. yds., hence about 13 per cent of tails would need to be processed; this could probably be achieved using single stage cycloning and the cost of the tailing sand will be about 15 cents per cu. yd. The total cost of the embankments therefore would be \$4.8 million, or about \$1.2 million for each cell. The saving in cost by delaying expenditure on future cells would also be worthwhile. The cost of waste rock 3 ft. thick placed on top of the tails on abandonment, assuming an average haul distance of 3 miles would be about \$1.5 per sq. yd., or \$4.1 million for each cell. This is considerable compared to the cost of the retaining embankment, but the alternative of providing impervious liners to keep the tails permanently submerged would be greater still. The cost of providing a durable Hypalon liner 20 mils thick beneath the entire area of each pond would be about \$3.02 per sq. yd., or \$8.4 million. If this alternative were chosen, the use of an impermeable liner, curiously, would be for reasons of air quality rather than water quality. Because the BWCA is considered a Class I PSD region, see Appendix B, great care would need to be taken to control tailing dust emissions. The reliability of waste rock for permanent rehabilitation of the area would seem therefore to justify the cost. The other alternatives for dust control, see Appendix B, indicate that it would be necessary to stay at least 5 miles away from the BWCA, which effectively rules out tailing disposal in the northeastern half of area I-C considered in this example.

The siting of the cells in Figure 18 has been chosen to suit existing topography, and the final elevation of each cell would vary between about 1,510 for cell HJKL to 1,550 for cell FCJG. The perimeter of the cells have arbitarily been kept about 1 mile away from the boundary of the BWCA and about 1,000 ft. away from the main stem of the South Kawishiwi River. Crossing of the river by the tailing pipeline would be necessary, however, and special precautions would need to be taken to prevent spillage. The pipeline could be placed in a flume that would direct spillage into a catchment basin constructed alongside the river. Spilled tails could then be collected and pumped to the pond.

e) Partridge River Watershed

A large area of land suitable for tailing disposal is situated in the SE half of the Partridge River watershed, area I-B shown on Figure 3. One of the alternatives for disposal of the Reserve Mining Company's taconite ore tails, the Colvin site, is located in this area, and it was discussed in the EIS (1975). This alternative, however, was rejected and the land would therefore be available for Cu-Ni disposal schemes. The areas discussed in this example are more extensive than the Colvin alternative. Detailed layouts are not proposed, but rather the character and tailing disposal capacity of individual sections of the area will be indicated.

The boundaries of the area under consideration are shown on Figure 19, and are as follows:-

a) To the W and NW, the main stem of the Partridge River, which roughly overlaps with the eastern extent of the zone of potential mineralization.

b) To the SW, the Duluth Missabi and Iron Range railroad.

c) To the S, the arc of till moraine ridges just north of Skibo and Stone Lake extending to Big Lake.

d) To the E, the Reserve Mining Company Railroad.

e) To the N, a small separate area encircled by the Erie Mining Company Railroad main and branch lines, and the Reserve Mining Company Railroad.

The majority of the eastern part of the area seems to be well suited to the central discharge method of tailing disposal. Two cones, B the larger overlapping A the smaller, could be centered over the 2 hillocks as shown on Figure 19. Cone A would have an average radius of about 9,000 ft. Cone B extending from Colin Creek on the W to the Reserve Mining Company Railroad on the E would have an average radius of about 15,000 ft. With the boundaries and top elevations of the cones as shown, it should be possible to store about 123 x 10^9 cu. ft. of tails, assuming that an average slope of the face of the cones of 4 per cent could be achieved. Much of the boundary of these cones would be formed by the till morainal features, and no dyking at all would be necessary along those stretches.

Catchment ponds for water effluent could be formed on Colvin Creek and the other minor tributaries of the Partridge River. The general direction of seepage flow is probably to the west, and no special measures would be needed to control seepage on the N, S and E boundaries of the area.

A further small central discharge tailing disposal scheme, cone C, could be located in the area to the N encircled by Erie Mining Company Railroad; this would store about 6 x 10^9 cu. ft. of tails.

The remainder of the area to the W from Cranberry Lake, ABCDEF, Figure 19, is generally flat lying and it could be used for conventional tailing storage within embankments. The usable area is about 8.25 sq. miles. If tails were deposited to an average thickness of 100 ft., a volume of about 23 x 10^9 cu. ft. could be stored, and the elevation of the retaining embankment would need to be up to about elevation 1650. The total storage potential of this basin is therefore about 152 x 10^9 cu. ft. which is equivalent to 6220 x 10^6 metric tons of tails or about 6500 x 10^6 metric tons of crude ore. This is more than sufficient to handle the output of all of the potential model mines within 10 miles of this disposal area.

An alternative location for a central discharge tailing system would be a cone centered at D, as shown on Figure 20. This is located closer to the zone of mineralization, and it fills the basin formed by Colvin and Cranberry Creeks. The S and E boundaries are formed by the till morainal features, and the Partridge River to the W is protected by similar smaller till ridges. Seepage control could therefore be restricted to the N boundary around Colvin Creek. This cone would have an average radius of about 11,000 ft., and it should be possible to store about 52 x 10^9 cu. ft. of tails; this is equivalent to about 2200 x 10^6 metric tons of crude ore.

6.0 SUMMARY AND CONCLUSIONS

a) Assuming that tailing disposal areas are on average 100 ft. thick, 100 x 10^6 metric tons of crude Cu-Ni ore (96 x 10^6 metric tons of tails) would require 538.5 acres of land area. This is based on a tailing dry density of 90 lb./cu. ft., so that 1 metric ton of tails requires 0.905 cu. yds. (24.43 cu. ft.) of storage volume.

b) Storage by the central discharge system with a cone of average face slope of 4 per cent requires land area as shown on the log-log plot on Figure 21. The height of the cone can be deduced from the simple relationship:-

$$h = 4.71 A^{1/2}$$

where h is the height of the cone in feet, and A is the base area in acres.

c) In general, it is less costly and more environmentally favourable to keep the areas occupied by the mine, mill and tailing disposal system as compact as possible. As much integration as possible of waste rock and tailing disposal should be made. It is also preferable for control purposes to site all the facilities within the same watershed; breakage of tailing lines is commonplace and precautions to contain spillage need to be taken.

d) The area under consideration has been characterized by the following hydrological and geotechnical features:-

i) The Laurentian Divide

ii) The Embarrass and Dunka River Basins where deep deposits of relatively pervious sands and gravels are found.

iii) The shallow bedrock-moraine topography extending from N of the St. Louis River into the Partridge, Kawishiwi, Stoney and Bear Island River watersheds. In this region native construction materials tend to be found scarce, but in general relatively impermeable foundations will be found at shallow depths.

iv) The Toimi drumlin-bog terrain S of the St. Louis River, where construction materials should be plentiful and drumlin ridge topography could be utilized to build retaining embankments. The intervening peat bogs would tend to minimize seepage losses.

v) The Aurora-Markham till plain SW of Hoyt Lakes underlain by intermittent peat overlying fine sands and silts and deep deposits of clay till. Foundation conditions would be variable.

vi) The Embarrass Mountains and the Seven Beaver-Sand Lake Wetland, both areas of which are unsuitable for tailing disposal.

e) Good materials for embankment construction are well graded sand and gravel, glacial till and waste rock. Tailing sands are not ideal construction materials, but they are generally by far the cheapest material available. If tailing sands are used in conservatively designed embankment, using the downstream method for example, they behave quite satisfactorily.

f) The investigation, design and supervision of construction of tailing embankments should be entrusted to competent engineers trained in geotechnics.

g) The quantity of seepage excaping from a tailing pond depends on details of the geology of the foundation soils. A generalized approach cannot be made, and estimates of underseepage should only be made for site specific studies.

h) Methods of seepage control are applied to ensure embankment stability, and for control of the amount of seepage flow to maintain water quality. The most common methods of seepage control are gravel underdrains, collection ditches, pressure relief wells, impervious liners of either clay or plastic, and vertical slurry trench cut-off walls. Impervious PVC plastic liners carefully installed would cost about $11,100 per acre of pond lined, assuming material 20 mils thick; durable Hypalon liner of the same thickness would cost about $14,600 per acre. Vertical slurry trench cut-off walls installed to the minimum wall thickness of 2 to 3 ft., and to the maximum dehth of 90 ft. would cost about $2.37 million per mile length of wall.

i) Costs of embankment construction per mile length for varying mean heights of embankment are shown on Figures 22, 23, and 24 for tailing sand, glacial till and waste rock, respectively. These graphs summarize total costs including all labour, equipment and any contractor expense costs. Figure 22 for the tailing sand assumes conservatively that double cycloning would be needed, and a cost of 20 cents per cu. yd. is used; if single stage cycloning is envisaged these costs could be scaled down to about 15 cents per cu. yd. Graphs for 4 different outside embankment face slopes are shown, varying from 3 horizontal to 1 vertical for embankments built on sound foundations such as shallow stiff till or bedrock, to 6 horizontal to 1 vertical for very weak peat foundations.

Figures 23 and 24 show costs for fixed embankment geometries, but for a range of haul distances, which reflect different unit costs as shown below:-

Glacial till	-	\$1.30/cu. yd. for excavation and loading
	-	\$0.50/cu. yd. per mile for haulage
	-	\$0.80/cu. yd. for placement and compaction
Waste Rock	-	\$0.20/cu. yd. for processing, i.e. sorting of oversize
	-	\$0.50/cu. yd. per mile of overhaul
	-	\$0.50/cu. yd. for placement, dozing and spreading.

It seems unlikely that a glacial till borrow pit could not be found within a 5 mile haul distance, and hence this has been chosen as the limit. However, it is conceivable that waste rock could be overhauled from normal waste dump sites to the boundaries of the study area; for this reason an extra curve for a 10 mile haul has been included.

j) It is emphasized that the purpose of this study has been to characterize the Cu-Ni region for tailing disposal, to provide geotechnical guidelines for planning of tailing systems and for establishment of regulatory procedures, and to provide approximate construction costs. The report should be used with these objectives in mind; site-specific designs would require more detailed consideration.

Yours very truly

GOLDER BRAWNER & ASSOCIATES LTD.

David B. Campbell

Per: David B. Campbell, P. Eng.

N.A. Skermer, P. Eng.

NAS/DBC:rme

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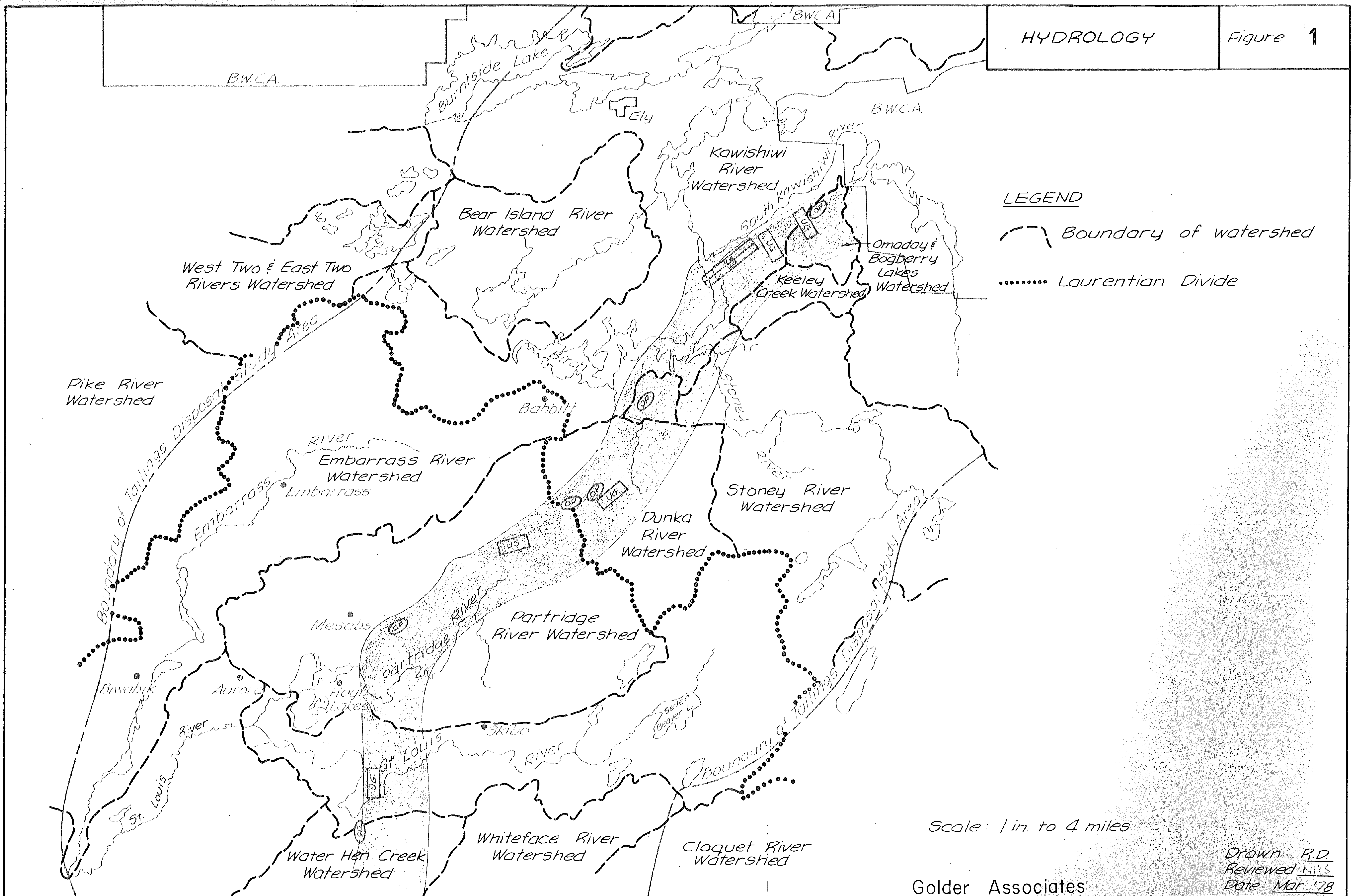
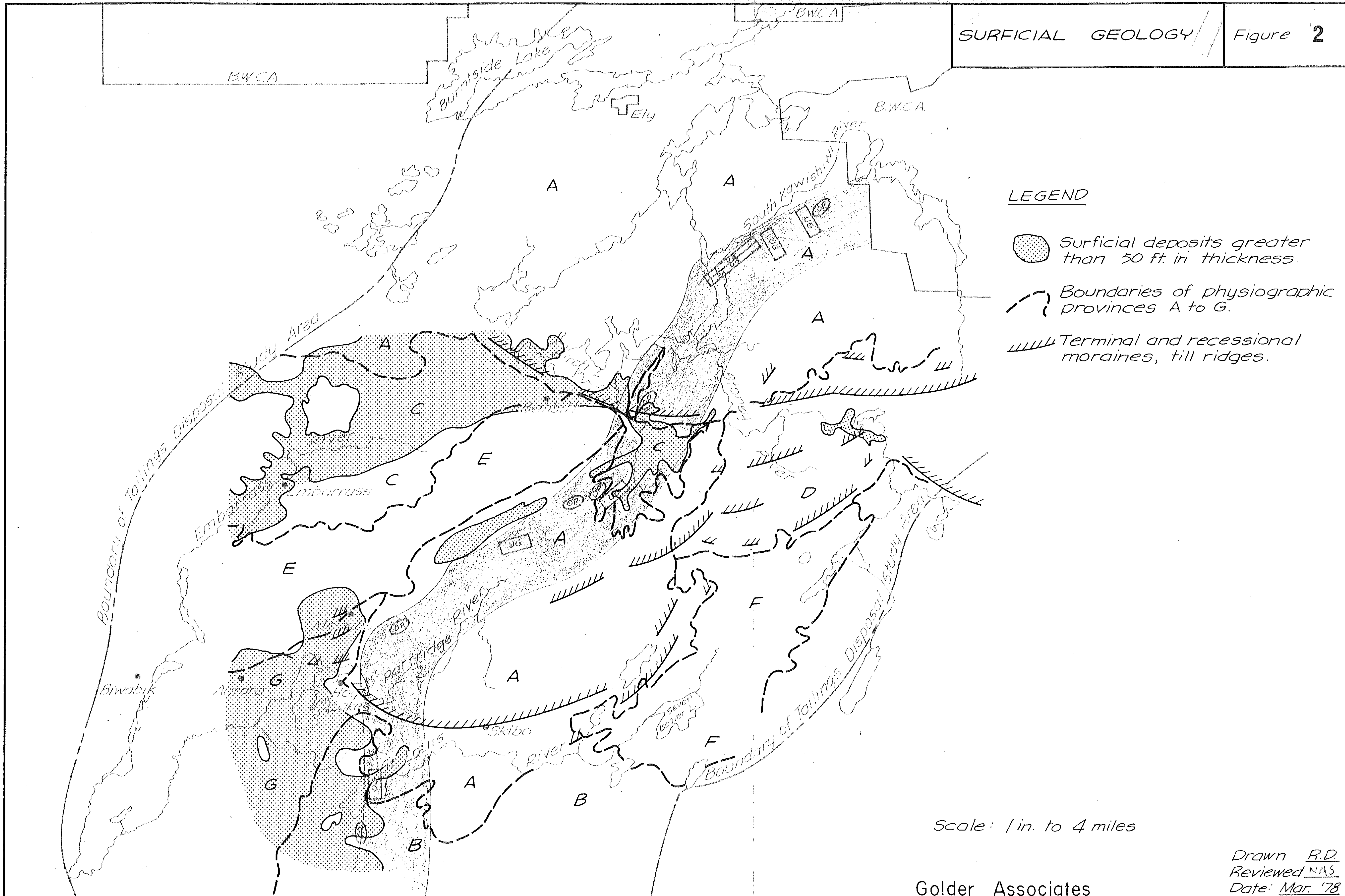


HYDROLOGY
Figure 1
LEGEND
Boundary of watershed
Laurentian Divide
B.W.C.A.
BWCA
Burntside Lake
Ely
Kawishiwi River Watershed
South Kawishiwi River
Bear Island River Watershed
West Two & East Two Rivers Watershed
Omaday & Bogberry Lakes Watershed
Keeley Creek Watershed
Birch
Babbitt
Pike River Watershed
Boundary of Tailings Disposal Study Area
River
Embarrass River Watershed
Embarrass
Stoney River Watershed
Dunka River Watershed
Partridge River Watershed
Partridge River
Mesabi
Biwabik
Aurora
Hoyt Lakes
Skibo
St. Louis River
Seven Beaver L.
Water Hen Creek Watershed
Whiteface River Watershed
Cloquet River Watershed
OP
UG
Scale: 1 in. to 4 miles
Drawn R.D.
Reviewed NIKS
Date: Mar. '78
V78034
Golder Associates



SURFICIAL GEOLOGY
Figure 2
LEGEND
Surficial deposits greater than 50 ft. in thickness.
Boundaries of physiographic provinces A to G.
Terminal and recessional moraines, till ridges.
B.W.C.A.
Burntside Lake
Ely
South Kawishiwi River
Boundary of Tailings Disposal Study Area
Embarrass
Partridge River
St. Louis River
Biwabik
Aurora
Skibo
Seven Beaver L.
Scale: 1 in. to 4 miles
Golder Associates
Drawn R.D.
Reviewed NAS
Date: Mar. '78
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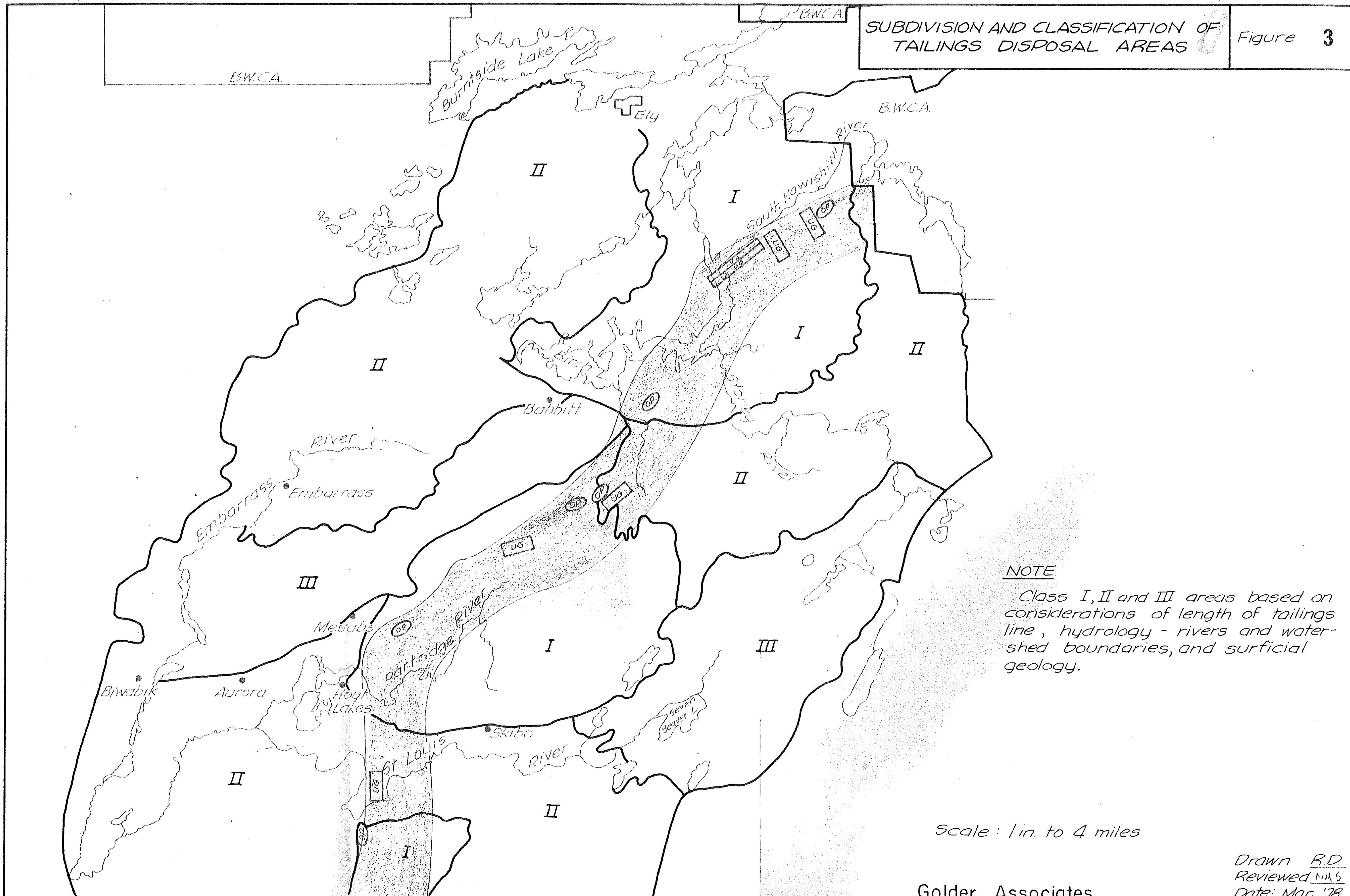

SUBDIVISION AND CLASSIFICATION OF TAILINGS DISPOSAL AREAS

Figure 3

NOTE

Class I, II and III areas based on considerations of length of tailings line, hydrology - rivers and watershed boundaries, and surficial geology.

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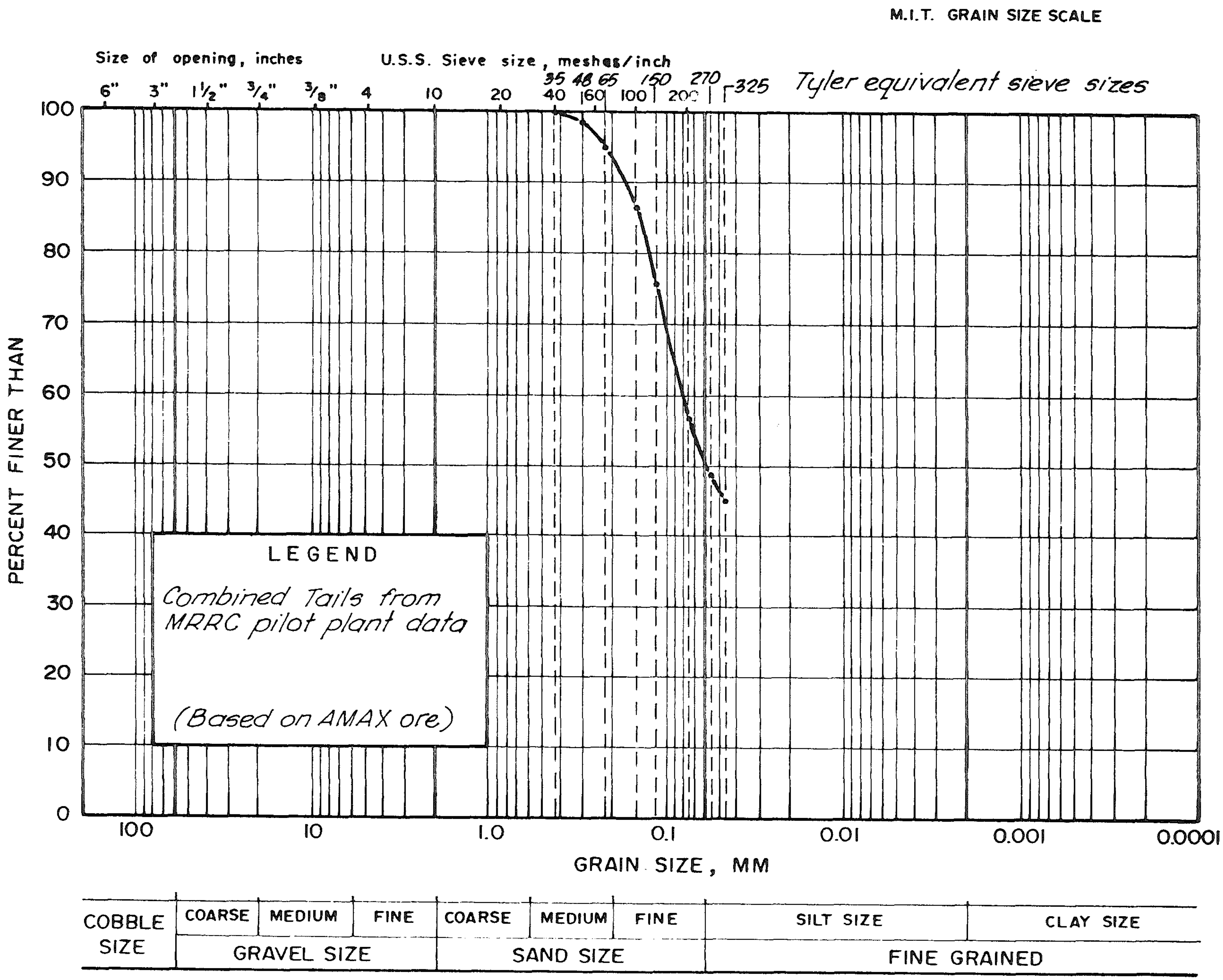

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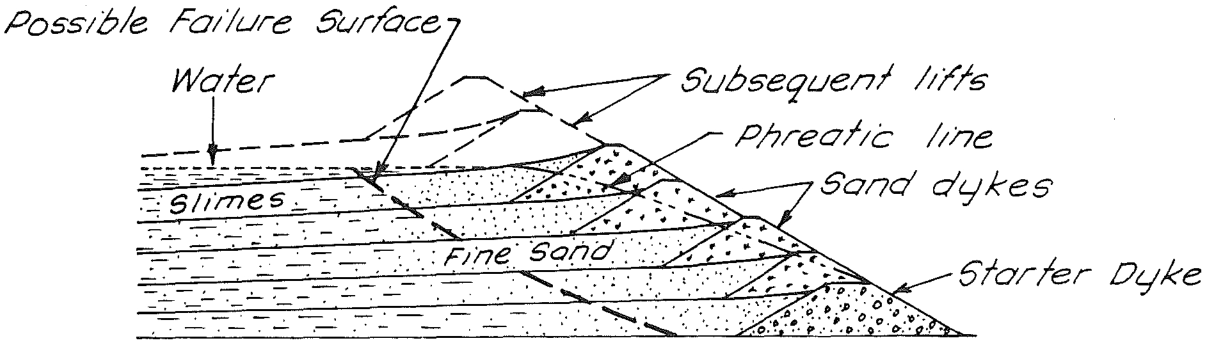
GRAIN SIZE DISTRIBUTION

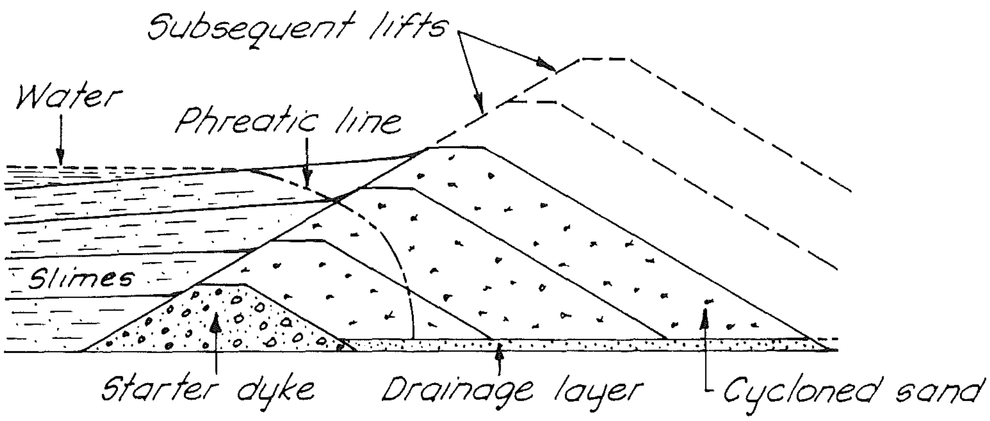
Figure 4

TAILING EMBANKMENT DESIGN

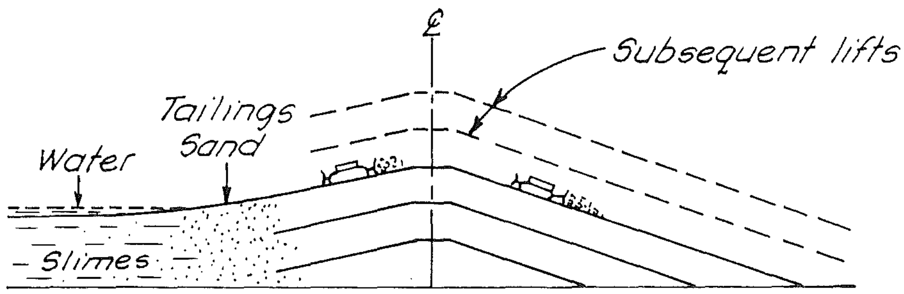
Figure 5



a) UPSTREAM METHOD



b) DOWNSTREAM METHOD



c) CENTRELINE METHOD

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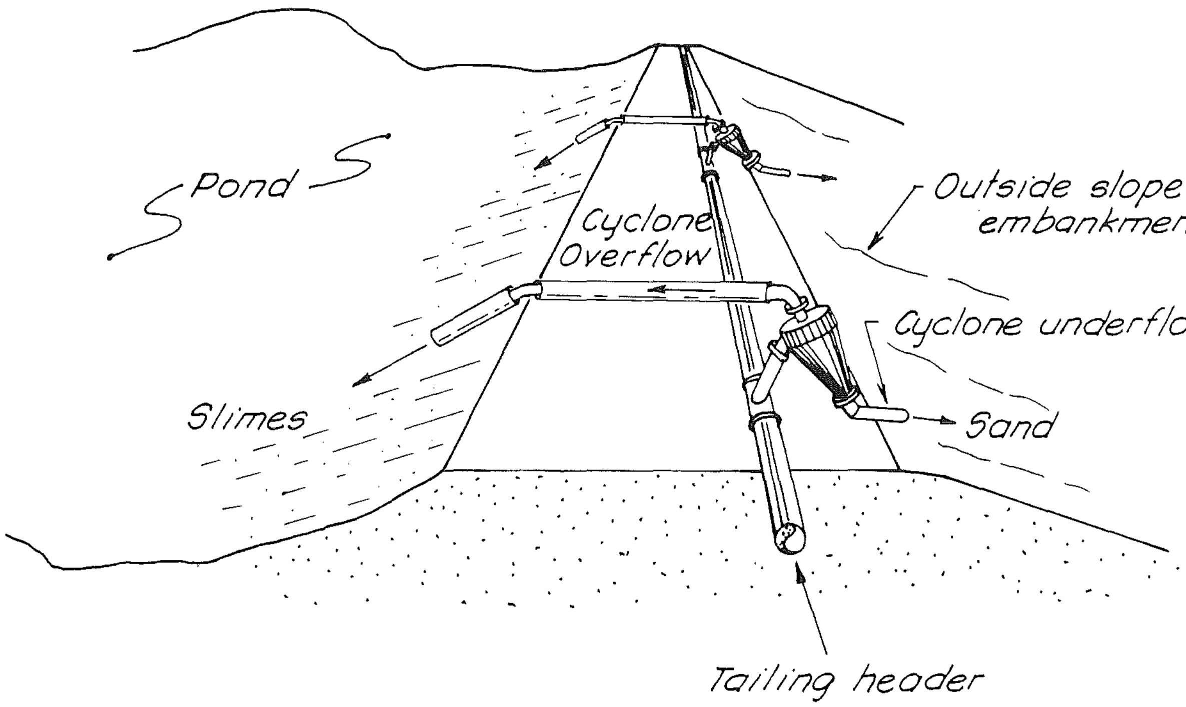
TAILING CYCLONING ARRANGEMENT **Figure 6**



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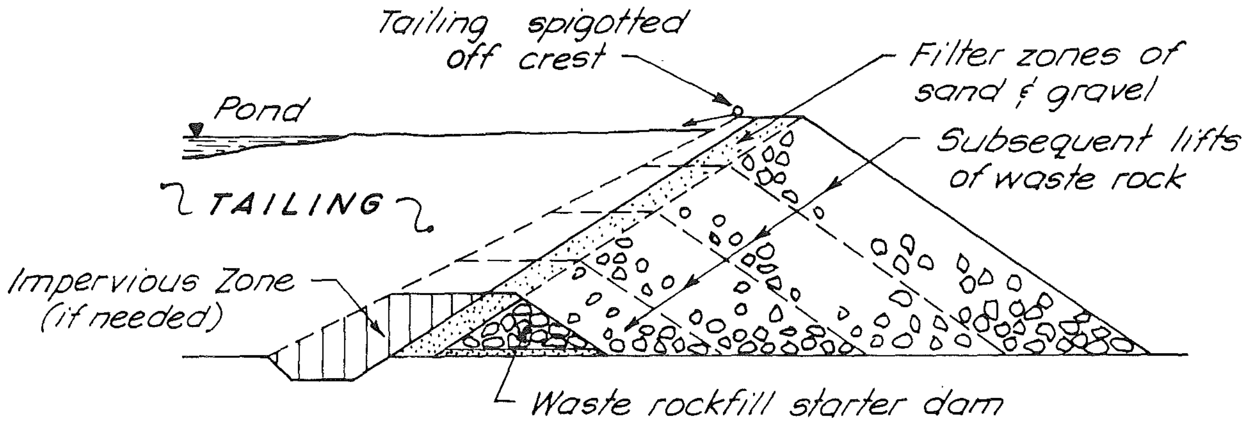
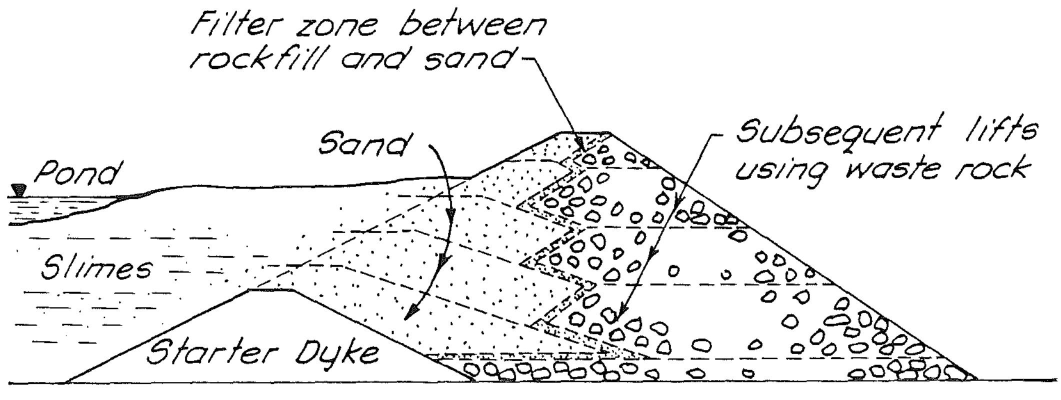
TAILING - WASTE ROCK EMBANKMENT Figure 7



a) WASTE ROCK



b) COMBINED CYCLONED SAND & WASTE ROCK

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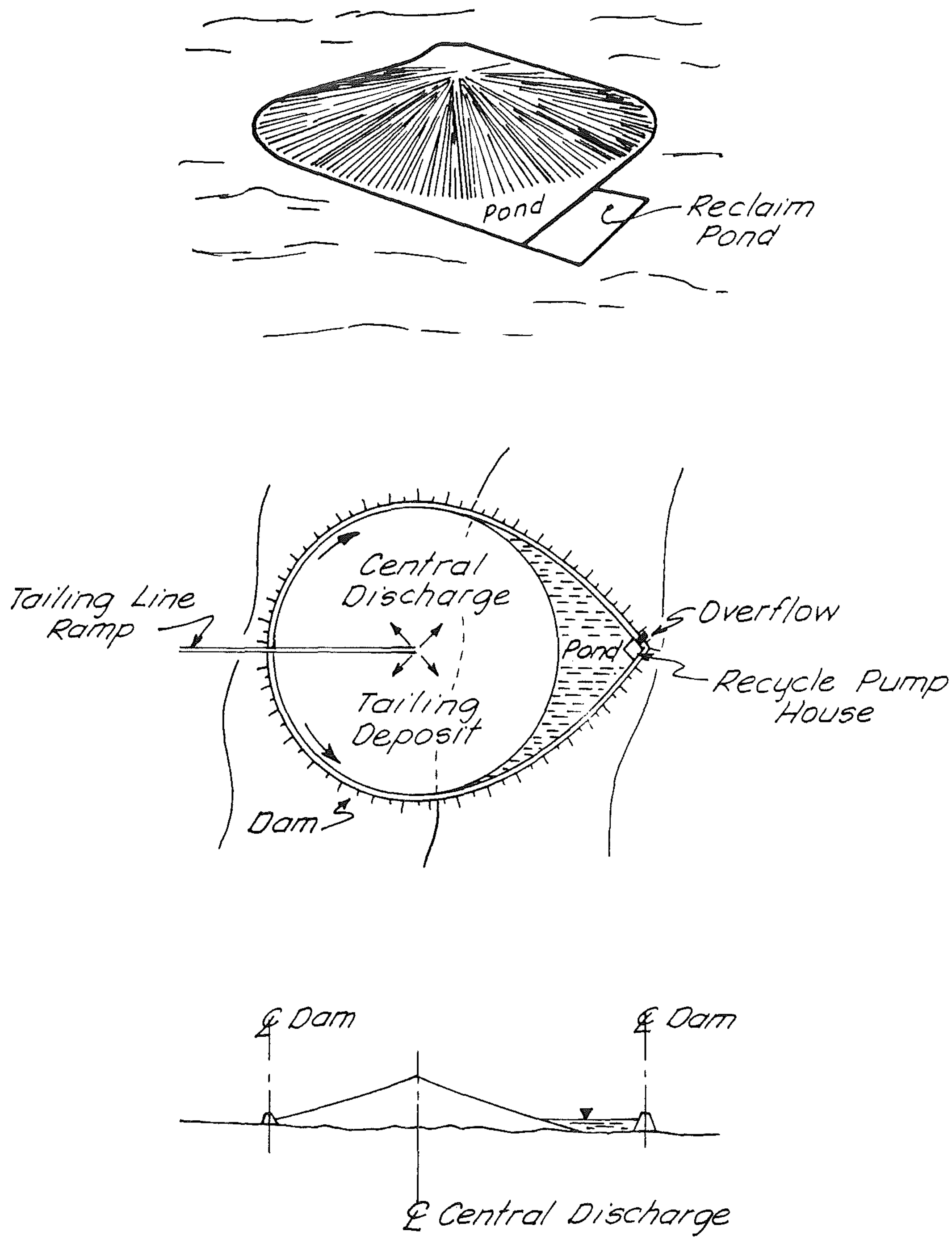
CENTRAL DISCHARGE Figure 8



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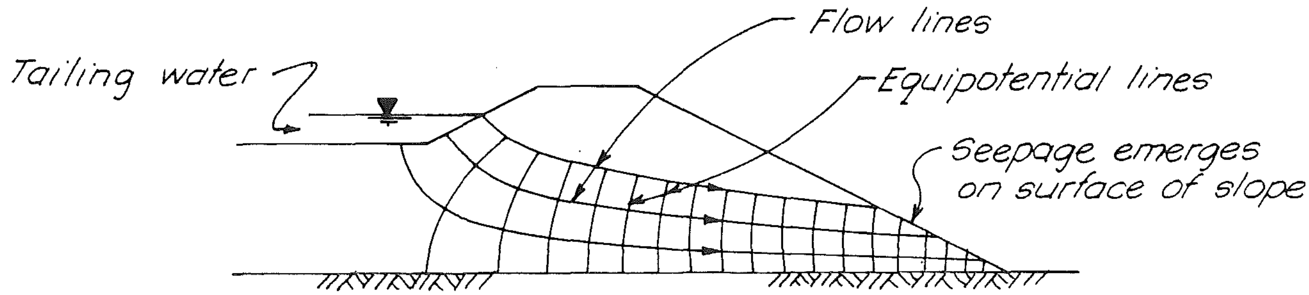
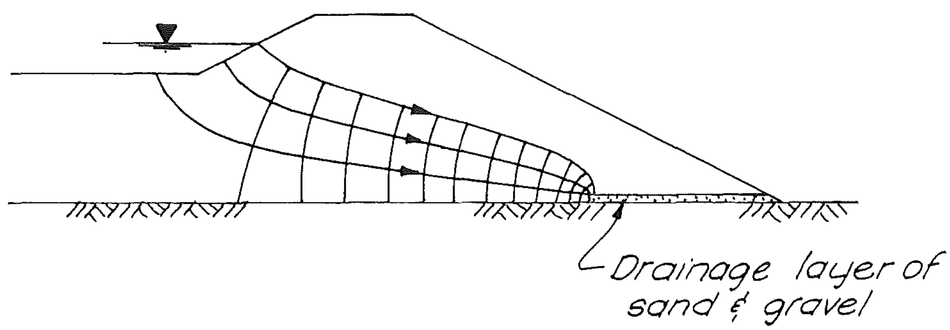
TYPICAL FLOW NETS

Figure 9



a) No drainage blanket. Homogeneous embankment on impervious foundation.



b) Blanket drain beneath downstream shoulder. Homogeneous embankment on impervious foundation.

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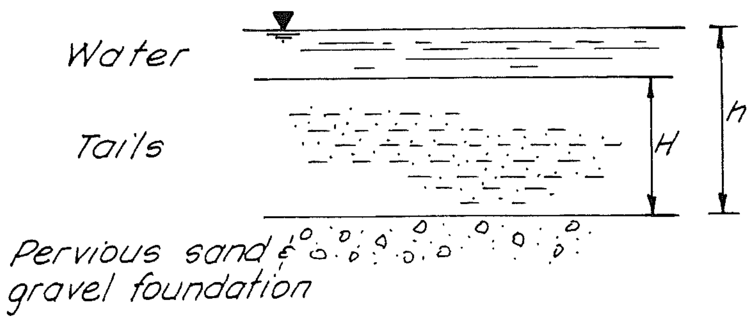
SEEPAGE OUT OF TAILING POND

Figure 10

a) Downward seepage under gravity.



By DARCY'S LAW:

Velocity of flow through tails, $v = ki$

where k = coefficient of permeability of tails

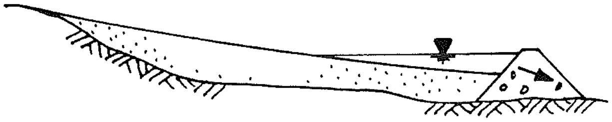
i = hydraulic gradient $\frac{h}{H}$

Quantity of seepage, $Q = vA$

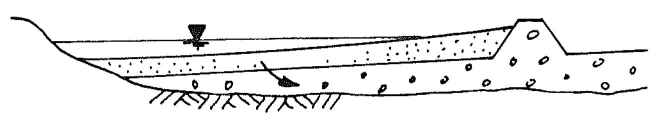
where A = area of tailing pond

Time for flow through tails $T = H/v$

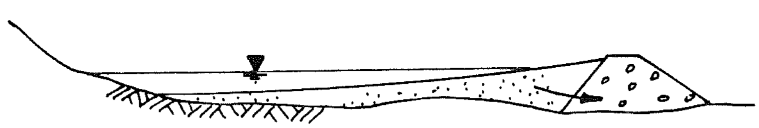
b) Flow through pervious embankment.



c) Flow through tails and pervious foundation.



d) Flow through tails and pervious embankment.



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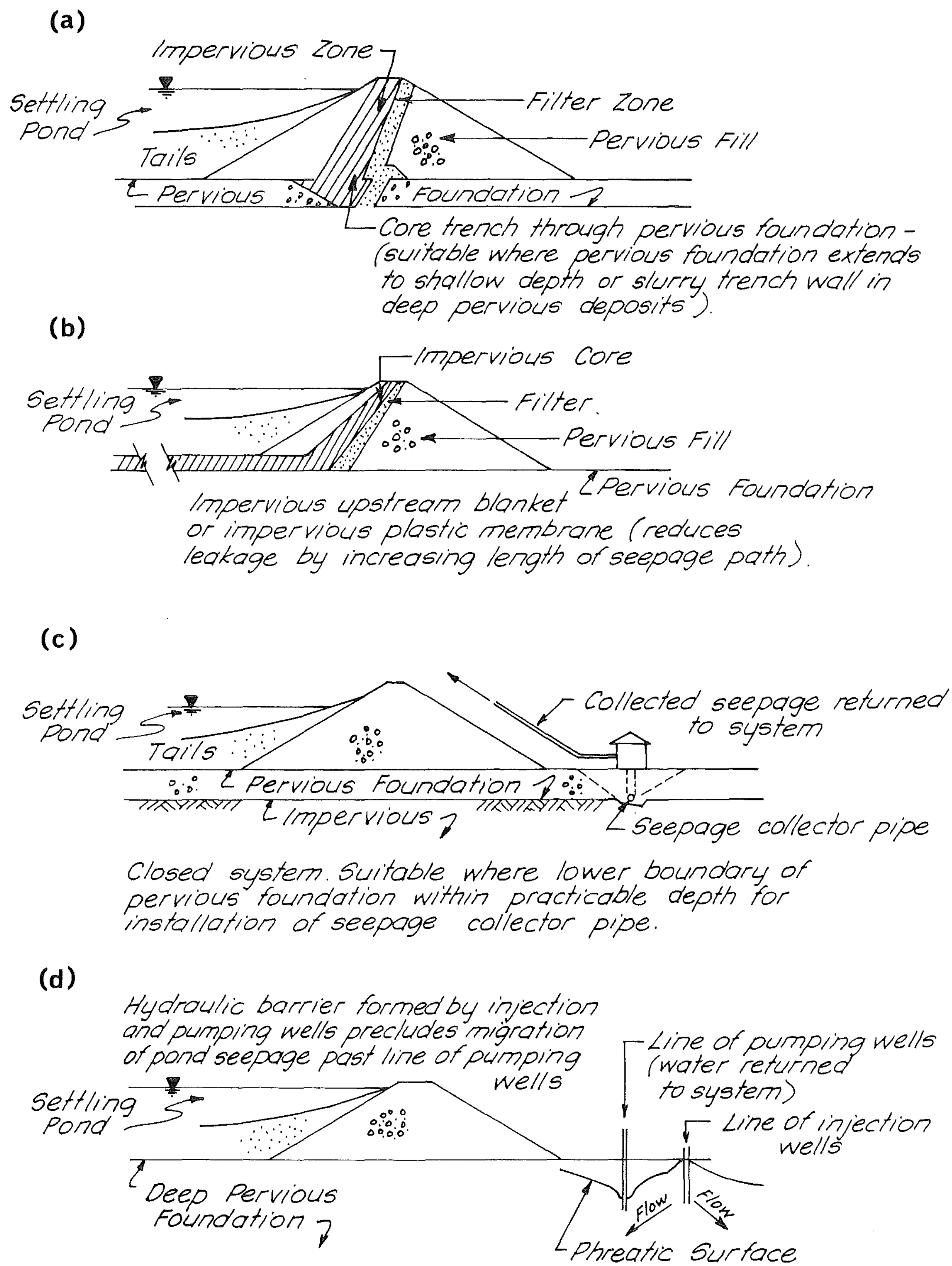
SEEPAGE CONTROLS

Figure 11



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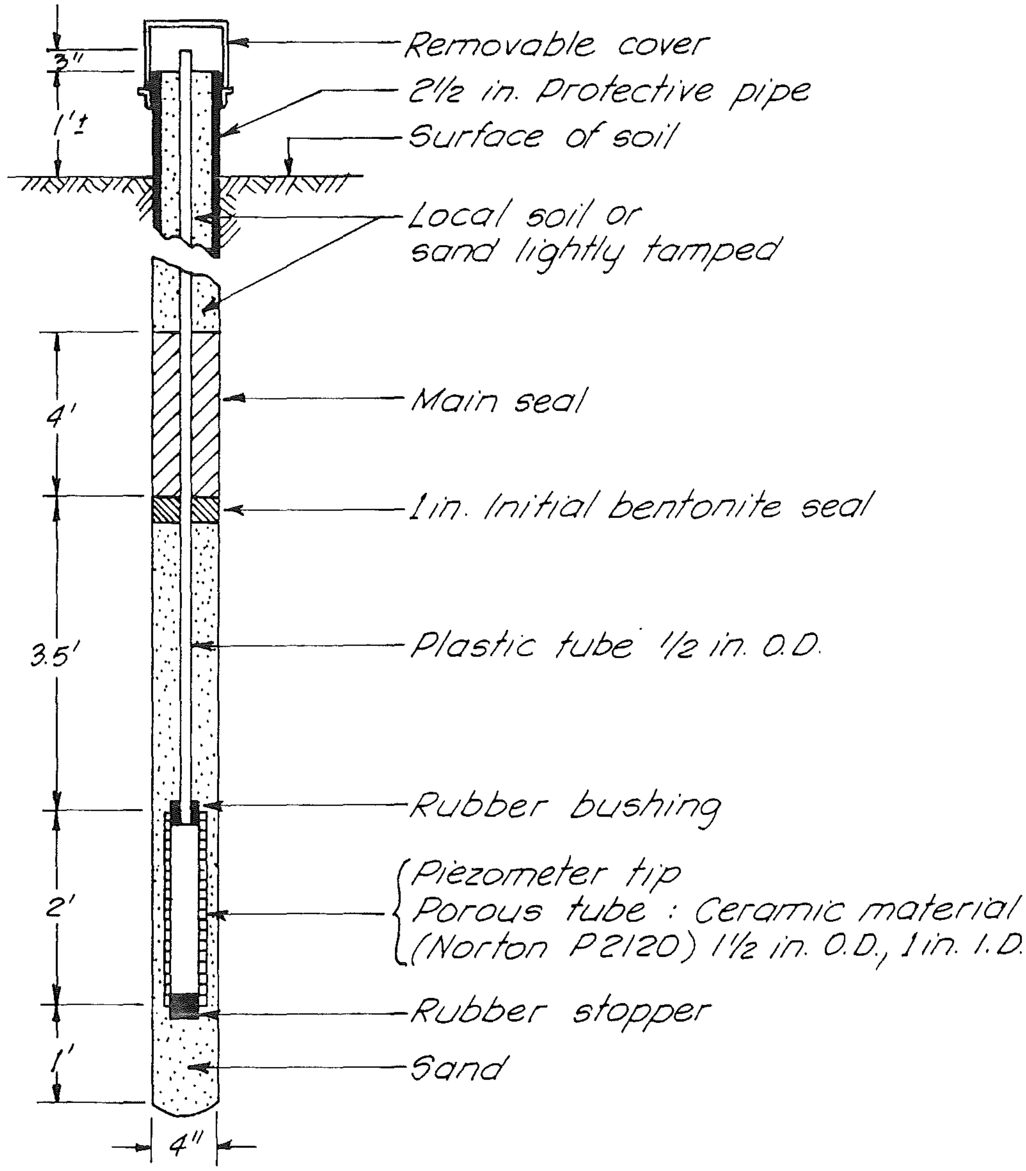
TYPICAL PIEZOMETER INSTALLATION

Figure **12**



Not to Scale

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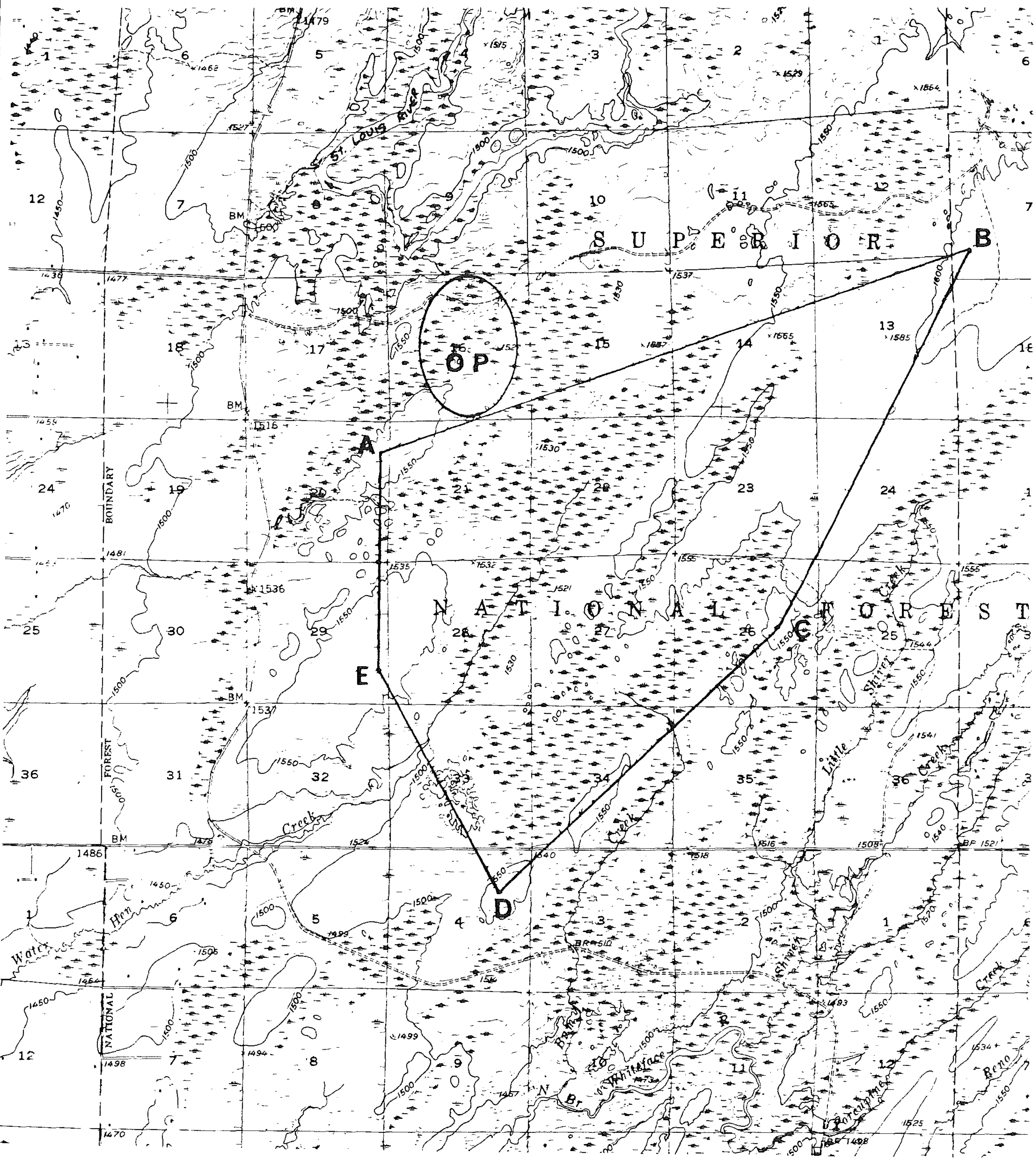
TAILING POND LAYOUT
SOUTH BRANCH WATER HEN CREEK

Figure 13



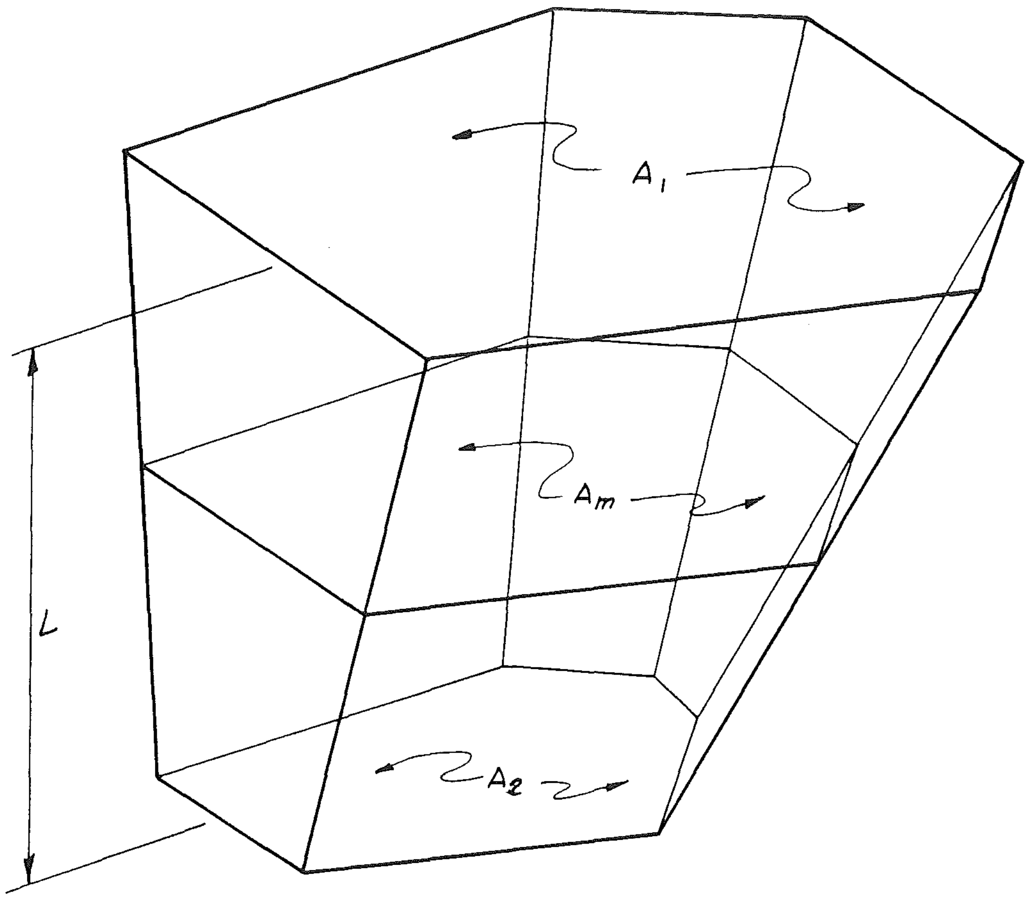
Scale 1 : 62,500
Contour Interval - 10 ft.

Topography - Markham Minn.
N 4715 - W 9200/15

V78034 - NAS Apr. '78

PRISMOIDAL FORMULA

Figure 14



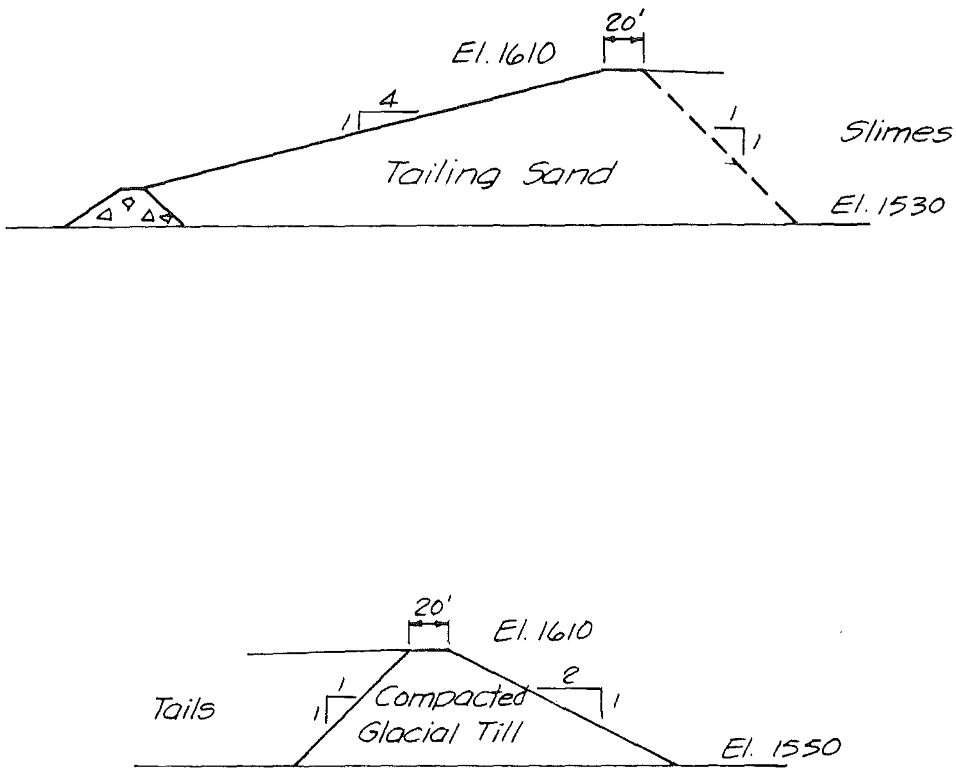
A_1 . Am and A_2 are parallel planes, and Am is midway between A_1 and A_2.

Volume of prism $V = \frac{L}{6}\left(A_1 + 4\,Am + A_2\right)$

V78034 . R.D. . NAS . Apr.'78

EMBANKMENT SECTIONS

Figure **15**



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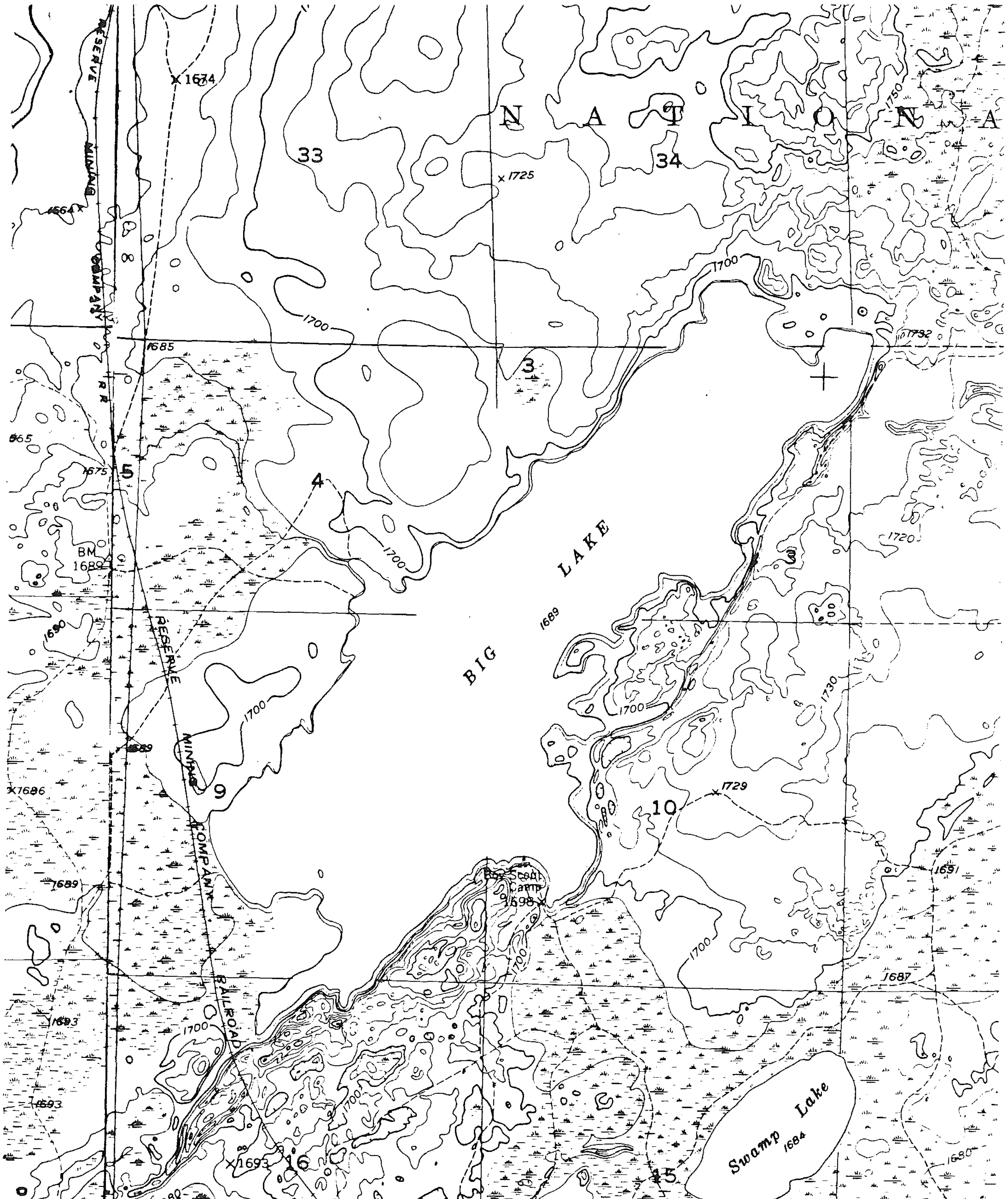
BIG LAKE

Figure 16



Scale 1 : 24,000
Contour Interval - 10 ft.

Topography SE Minn.
N 4730 - W 9145/7.5

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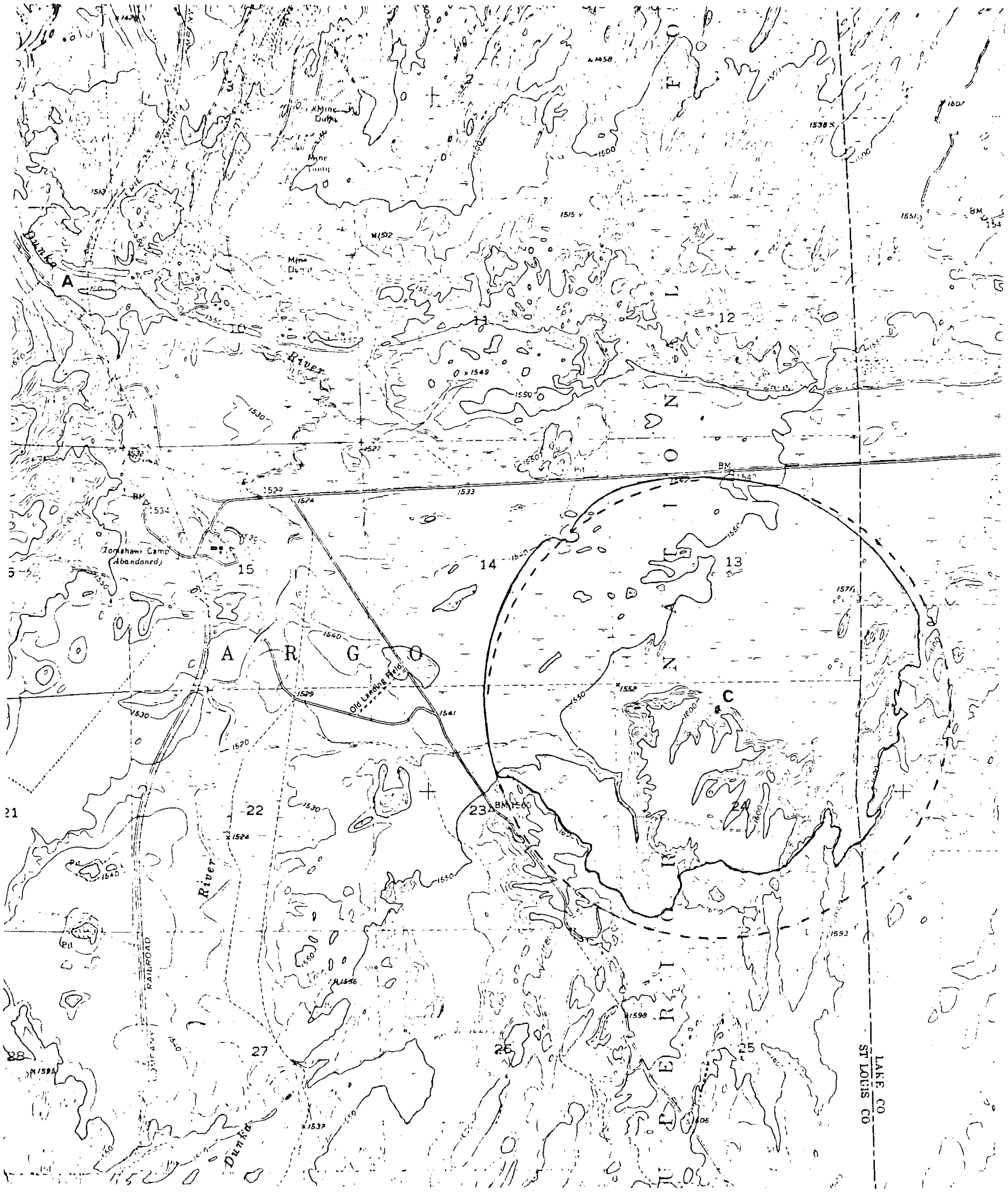
DUNKA RIVER BASIN

Figure **17**



Scale 1:36,000 (approx.)

Topography - Babbitt NE, Minn.
N 4737.5 - W91145/7.5

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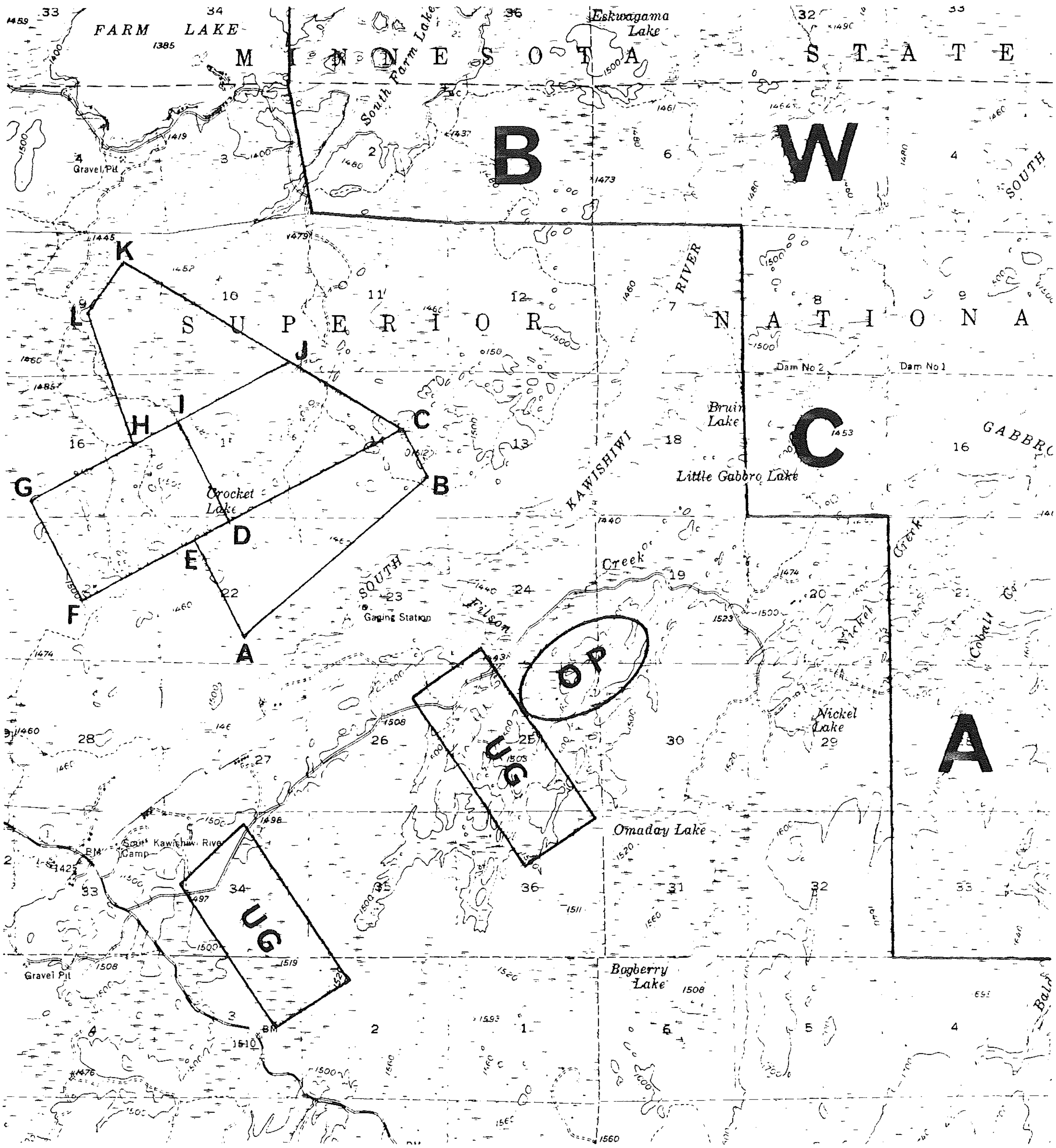
TAILING POND LAYOUT
SOUTH KAWISHIWI RIVER

Figure 18



Scale - 1:62,500
Contour interval - 20 ft.

Topography - Gabbro Lake, Minn.
N4745 - W9130/15

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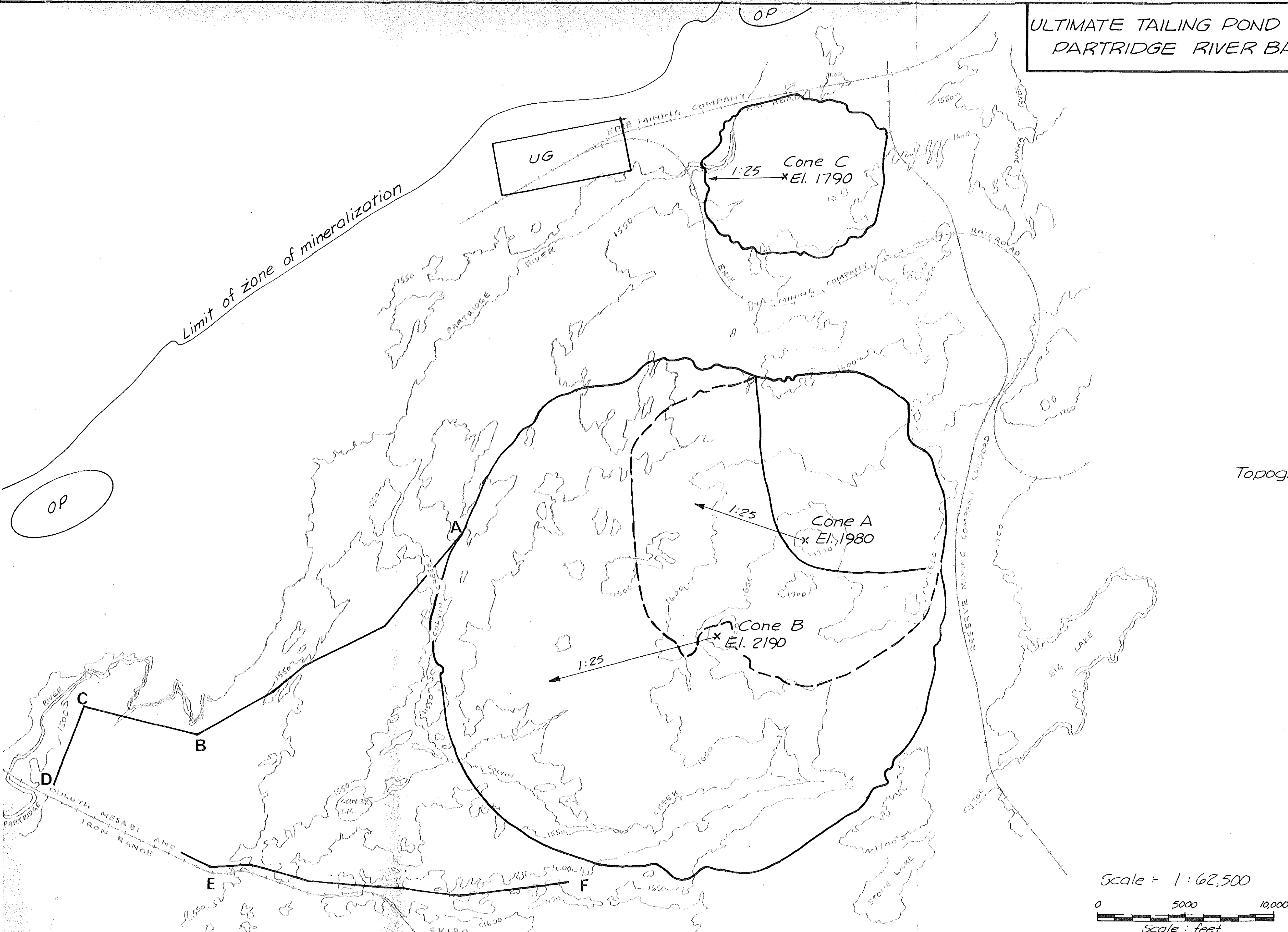
ULTIMATE TAILING POND STORAGE
PARTRIDGE RIVER BASIN

Figure 19



Topography - Allen, Minn.
N 4730 - W9200/7.5
- Babbitt, S.W. Minn.
N 4730 - W9152.5/7.5

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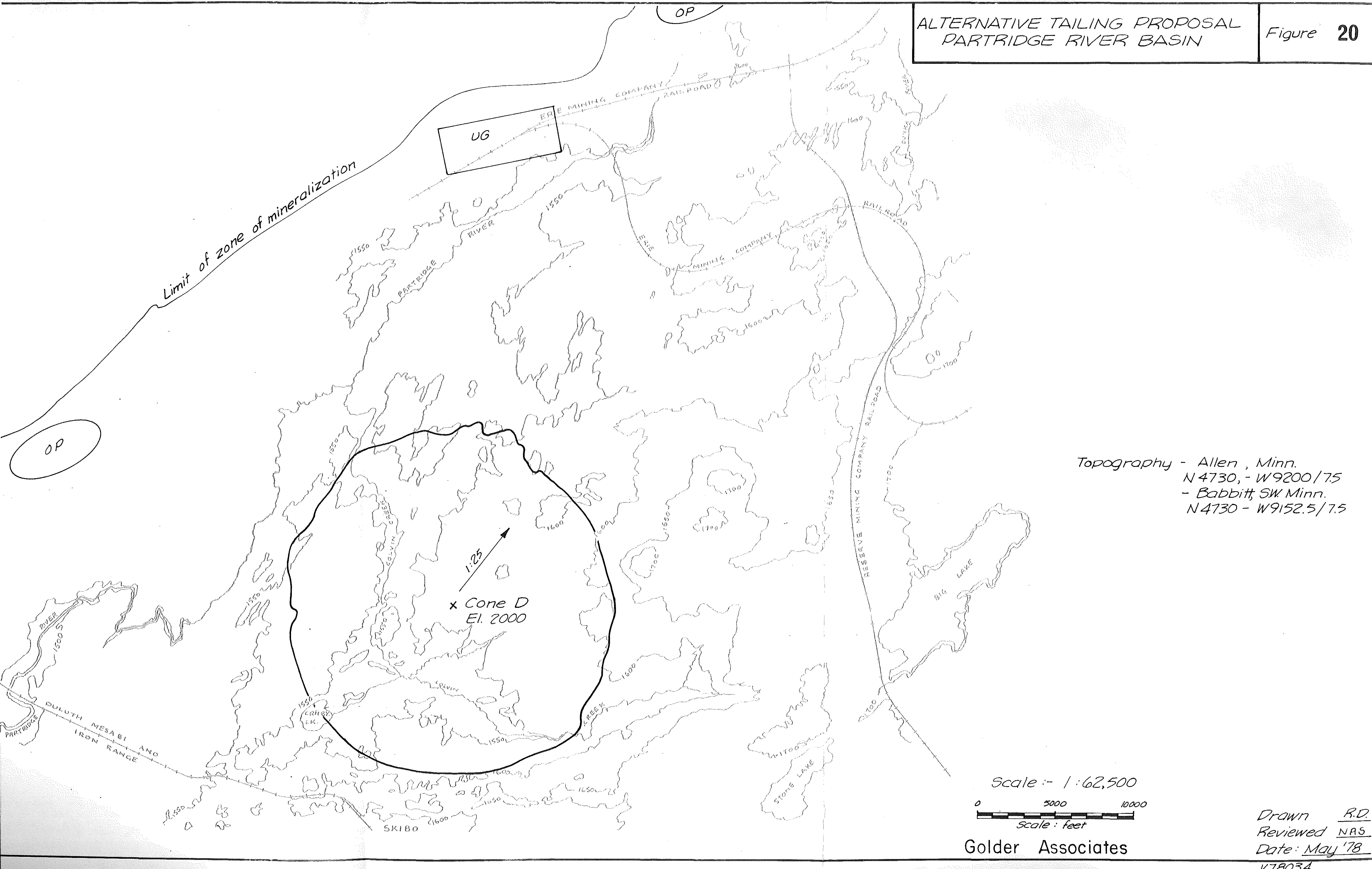
V78034

ALTERNATIVE TAILING PROPOSAL
PARTRIDGE RIVER BASIN

Figure **20**



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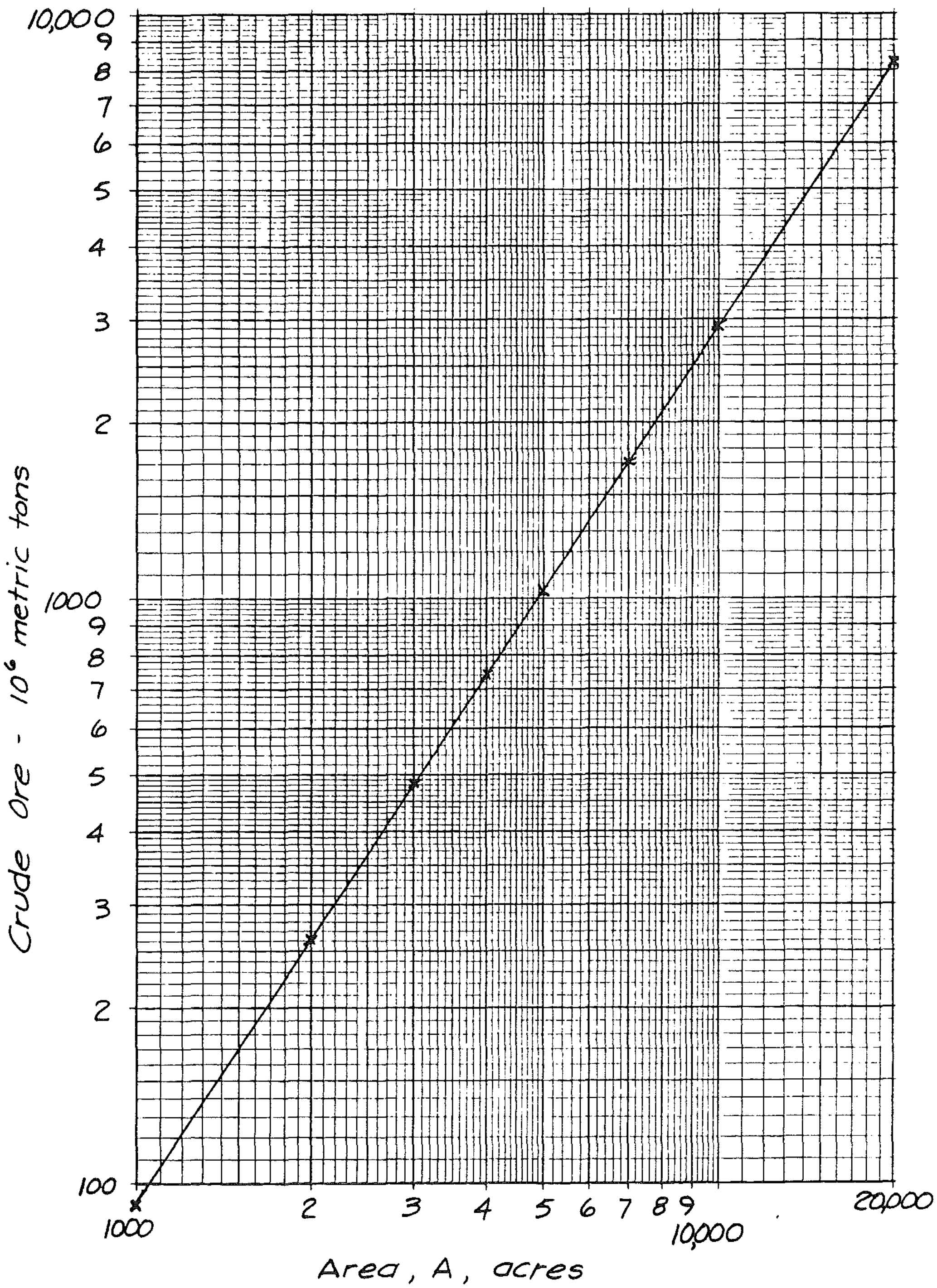
Reviewed NAS
Date: May '78
V78034

ORE TONNAGE vs. CONE AREA

Figure 21



Tonnage = 2915 $A^{3/2}$, where
A is area in acres, and
level ground is assumed.
Face slope of 4 percent.

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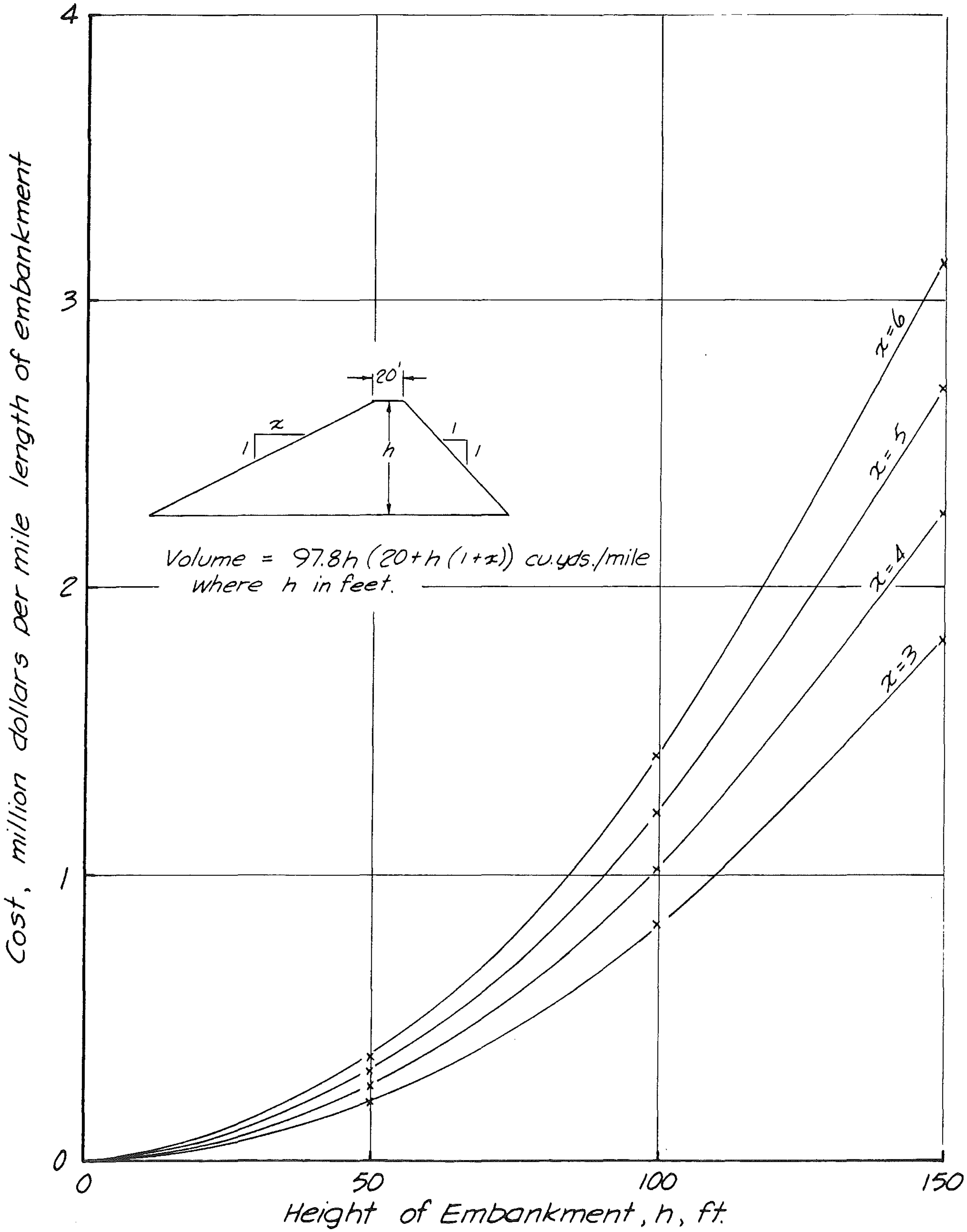
COST OF EMBANKMENTS BUILT FROM TAILING SAND

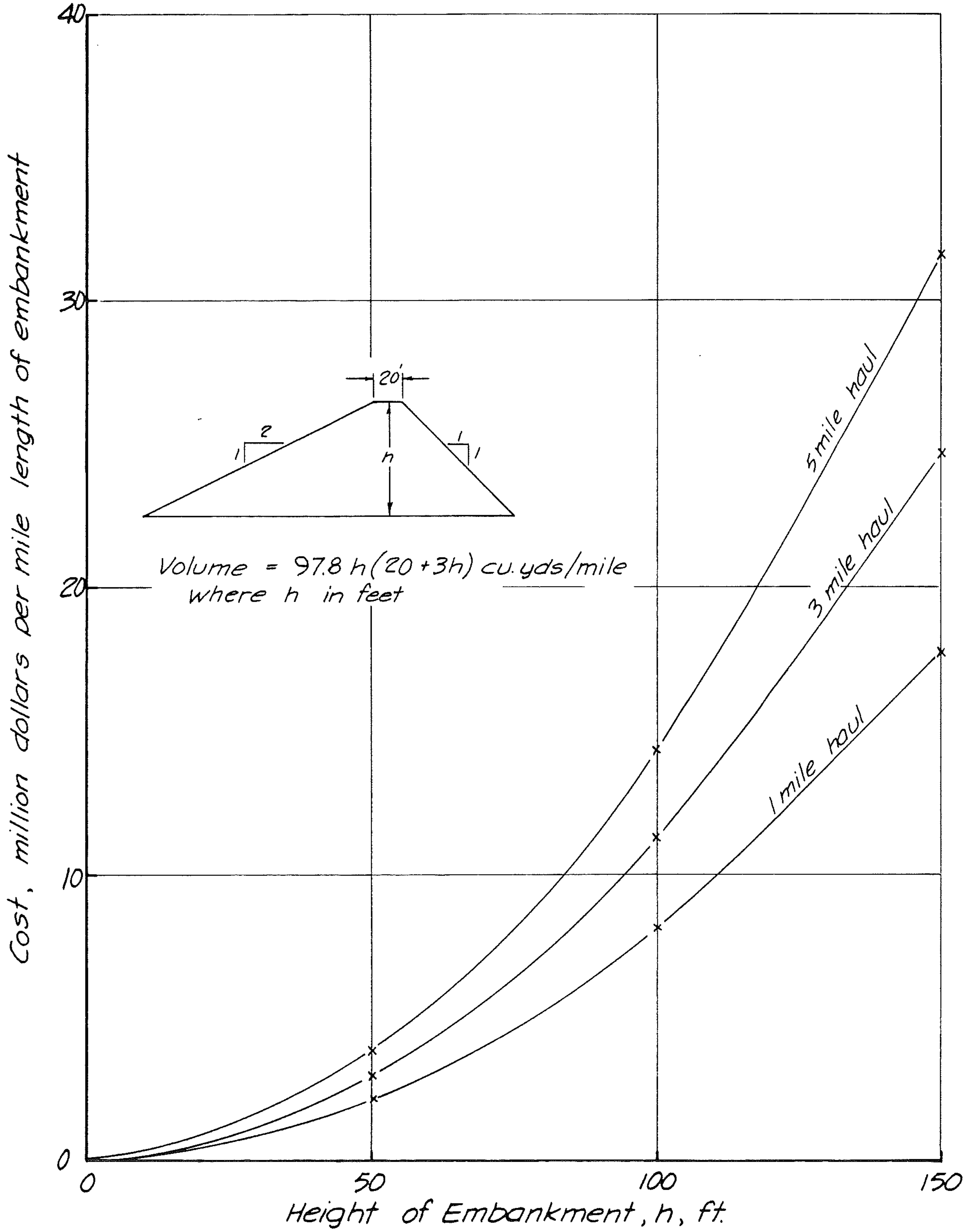
Figure 22



Project No. V78034. Drawn R.D. Reviewed NAS Date May '78

COST OF EMBANKMENTS BUILT FROM GLACIAL TILL

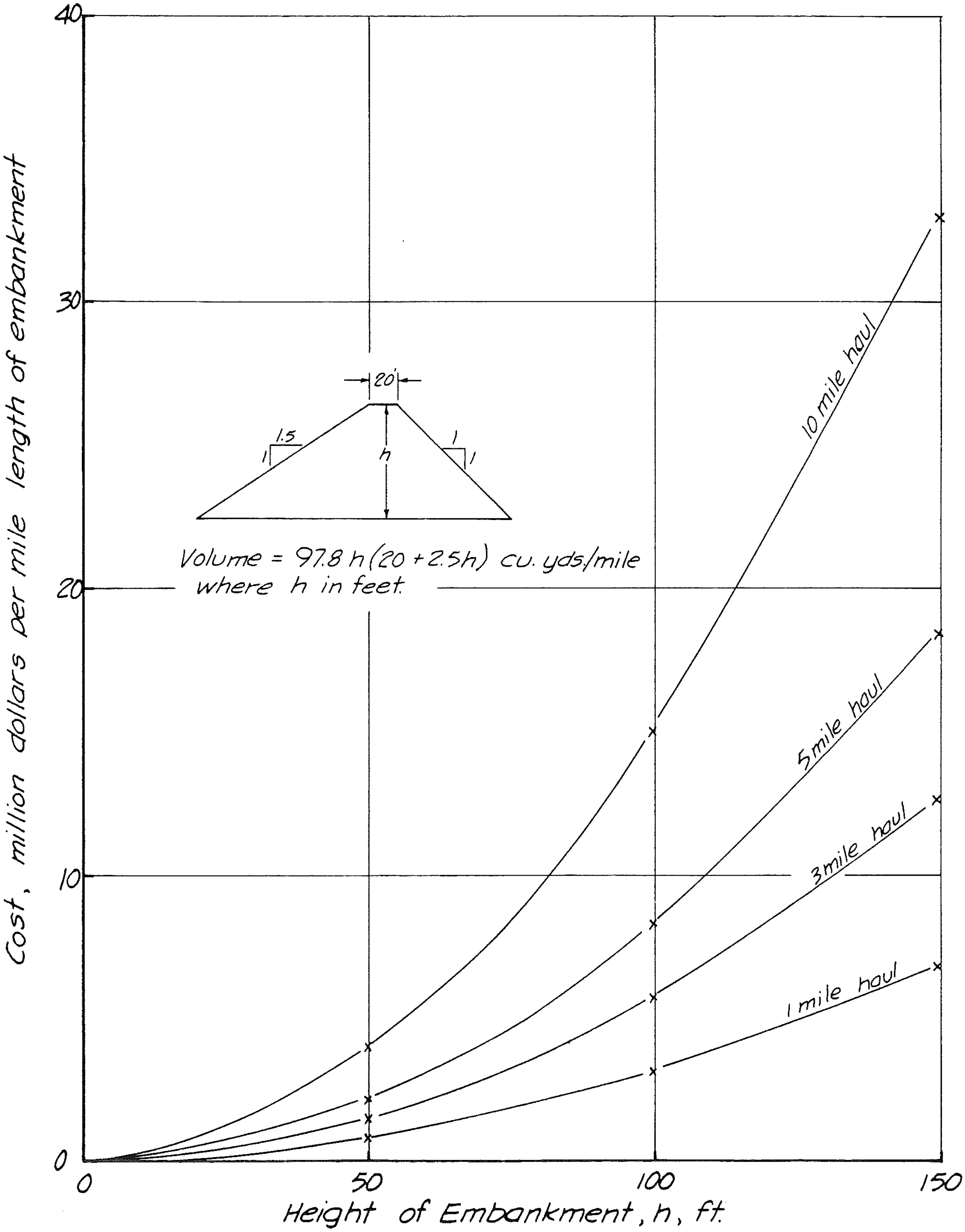
Figure 23



Project No. V78034 Drawn R.D. Reviewed NAS Date May '78

COST OF EMBANKMENTS BUILT FROM WASTE ROCK

Figure 24



Project No. V78034 Drawn R.D. Review NAS Date May '78

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APPENDIX A

AN EVALUATION OF TAILINGS DISPOSAL IMPACTS EXPECTED IN THE DEVELOPMENT OF COPPER-NICKEL SULPHIDE RESOURCES: RECLAMATION-RELATED SITING AND DESIGN CRITERIA

PREPARED FOR

GOLDER ASSOCIATES
AND
THE MINNESOTA ENVIRONMENTAL QUALITY BOARD

PREPARED BY

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DENVER, COLORADO

MARCH, 1978

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INTRODUCTION

Mining development in the MINESITE area of northeastern Minnesota will require the development of mining plans and assessment of environmental impacts. The natural resources in the study area and the maintenance of these resources will be of major importance. In accordance with the Minnesota Mined-Land Reclamation Act of 1976 and the proposed Minnesota Mineland Reclamation Rules (Department of Natural Resources Proposed Rules, NR 401-411), a reclamation plan will be required as part of the application for a permit to mine.

The reclamation plan will address mineland reclamation including the reclamation of tailings basins. The reclamation standards as proposed in NR 407 list specific requirements with regard to the reclamation of tailings basins. The purpose of the reclamation program is to restore tailings areas to productive use and to control possible adverse environmental effects.

The purpose of this part of the "Evaluation of Tailings Disposal Impacts Expected in the Development of Copper-Nickel Sulphide Resources" was to develop siting and design criteria relative to reclamation of tailings in the MINESITE study area. The specific objectives of the study were to: (1) define reclamation-related criteria to be applied in the evaluation of alternative locations for siting of tailings disposal areas, (2) compile reclamation and restoration design criteria to be applied in the evaluation of plans for the reclamation of tailings basins, and (3) to identify generic reclamation-related constraints or controls applicable to the tailings disposal areas in the MINESITE study area. The early consideration of reclamation in the siting and design of tailings basins permits the development of a program to control and mitigate possible adverse environmental effects.

LOCATION CRITERIA

Suitable locations for the siting of tailings basins can be identified on the basis of engineering and environmental considerations, including reclamation. Reclamation will be an essential part of conservation and land management in the MINESITE study area. Evaluation of alternative tailings basin sites will include the assessment of planned future land uses and reclamation potentials. Location criteria for evaluating alternative locations have been developed and address: existing land uses, alternative land uses, vegetation and wildlife, topography, soils and aesthetics.

Existing Land Uses

Land uses in the MINESITE study area include agriculture; recreation; urban and non-urban residential, commercial industrial uses; and mining. Much of the land in the MINESITE study area is national forest or state forest land. The following criteria provide guidance in evaluating alternative locations for tailings basins.

- No tailings basin shall be located on or within 1/4 mile of the boundary of:
  - (1) any National or State Wilderness Area
  - (2) any National or State Wild, Scenic or Recreational River or River District

(3) any non-mining site designated in the National or State register of historic places

(4) any National or State park

- Tailings basins shall be located to be compatible with existing land uses.

Alternative Land Uses

Possible future land uses in tailings basin areas should be considered in the site selection phase. The selection of alternative land uses must involve consideration of natural resource uses, potential land use conflicts and planned land uses in the surrounding area and region. The following list of criteria provides a basis for evaluating alternative locations:

- No tailings basin shall be located on areas being studied or proposed for inclusion in the State Outdoor Recreation System unless no reasonable alternative exists.
- Tailings basin shall be located to be compatible with planned land use.
- Alternative land use of tailings basin shall be compatible with surrounding planned future land use.
- Soil capabilities shall be compatible with planned land use and capable of maintaining productive land use.

- Consideration shall be given to the potential recreation enhancement opportunities.

Vegetation and Wildlife

Vegetation and wildlife are important considerations in selecting a site for a tailings basin. Consideration of vegetation and wildlife in the siting phase provides an opportunity to plan for continued productivity and wise use of these resources in the region. Location criteria relative to vegetation and wildlife are:

- No tailings basin shall be located in any designated State Scientific or Natural Area.
- No tailings basin shall be located within a National Wildlife Refuge, State Wildlife Management Area, or National Waterfowl Production Area except when no reasonable alternative exists.
- Consideration shall be given to vegetative diversification and opportunities for wildlife habitat enhancement.
- Consideration shall be given to unique vegetation and wildlife resources and important plant and animal communities which are limited in the region.
- No tailings basin shall be located in designated Critical Habitat of any Federally-listed endangered species.

Topography, Soils and Aesthetics

Other factors which must be considered in selecting suitable locations for tailings basins include topography, soils and aesthetics. These factors are interrelated with other aspects of the site selection process.

- o Tailings basins shall be located to be consistent with land form aspects defined in NR 407 C.1 and D.1.
- o Topography on and in the vicinity of tailings basin shall be compatible with planned future land uses.
- o Soil types on and in the vicinity of tailings basin shall be compatible with planned future land uses.
- o Soils suitable for reclamation either as backfill or top soil should be available and reasonably accessible.
- o Tailings basins shall be located to be visually compatible and consistent with NR 407 C.
- o Tailings basins should be located in an area where it is aesthetically acceptable.
- o The location of tailings basins should make use of natural screening features (e.g., vegetation and land forms).

RECLAMATION/RESTORATION CRITERIA

The reclamation of tailings basins is an essential part of the conservation of natural resources and resource use management. Reclamation plans and programs need to be developed early in the feasibility phase of mine planning and development. Early planning provides the opportunity to design successful reclamation programs while maintaining a degree of flexibility for future resource use. The design criteria included below provide some guidance for the planning and development of a successful program of reclamation and restoration of tailings basins in the MINESITE study area. Consideration is given to tailings basin construction and operation, planned land use, soil properties, vegetation treatment and aesthetics.

Tailings Basin Construction and Operation

- Tailings basins shall be designed and constructed in accordance with the standards defined in NR 407 D.g.
- Tailings basin construction and operation shall expose the smallest practical area of bare soil surface for the shortest possible time (e.g., surface stabilization and reclamation shall be initiated at the earliest practical time).

- Chemical stabilizing amendments shall be applied to exposed surfaces as a temporary means of controlling soil erosion until more permanent means (i.e., physical and vegetative) can be applied.
- Scheduling priority shall be given to areas susceptible to erosion (e.g., berm embankments with slopes of greater than 5%).
- A suitable plant growth medium shall be developed on the tailings disposal area.

Planned Land Uses

- Consideration shall be given to all alternative land uses to identify uses with highest potential for development and reclamation of the tailings basin.
- Consideration shall be given to the potential diversification of planned land uses for development and reclamation of the tailings basin.
- Planned land use shall be designed to maximize potential multiple uses of the tailings basin.
- Planned land use shall be compatible with the best available reclamation technology.
- Planned land use shall provide continued productive use in the tailings disposal area.

Soil Properties

- Overburden coverings, backfill and/or tailings shall be developed to provide a suitable plant growth medium.
- Mulching, composting, sewage sludge or plowing under of initial planting or other appropriate materials shall be applied to increase the amount of organic matter in the soil and to provide a substrate for soil microbes.
- Fertilizers, sewage wastes or other sources of nutrients necessary for plant growth shall be applied as required to support vegetation.
- Appropriate techniques shall be applied to improve soil pH, salinity and toxic material levels as necessary to permit satisfactory growth of vegetation.
- Surface soil stabiliztion techniques shall be applied to the tailings surface as required to minimize loss of vegetation due to wind-blown soil erosion damage.
- An irrigation regime shall be designed to provide adequate soil moisture balance during the germination and seedling establishment.

Vegetation Treatment

- Vegetation treatment shall be designed in compliance with the goals and requirements defined in NR 407 E.

- Vegetation treatment shall be designed to provide (1) rapid stabilization of surface soils, (2) mulch for soil development, (3) long term stabilization of reclaimed areas, and (4) permanent, self-sustaining vegetation cover compatible with or made up of species indigenous to the surrounding area.
- Vegetation treatment shall involve a seed mixture and successive plantings suitable for (1) species and ecotypes which are compatible with planned land use and the variety of microclimate sites on the tailings disposal area, (2) a diversity of species capable of maintaining interspecific associations, and (3) a diversity of rooting depths and patterns suitable for long term soil stability.
- Seeding and transplanting shall be consistent with best available technology including seeding and transplanting rate and spacing, seeding and transplanting methods, mulching, protection from animal damage, and irrigation as required.

<u>Aesthetics</u>

- Visual buffers or barriers shall be developed consistent with NR 407 C.2 and E.3 to provide an aesthetically acceptable tailings disposal area.

RECLAMATION-RELATED CONSTRAINTS

The major reclamation-related contraints on reclaiming tailings basins in the MINESITE study area include certain soils properties, biological sterility, and toxic materials in the tailings. The degree to which each of these major constraints will influence reclamation will vary with the characteristics of the tailings. Control of some of these constraints (e.g. biological sterility) is feasible and can be accomplished through the application of appropriate techniques. Constraints related to such soil properties as salinity and pH are somewhat more difficult to control.

Soil Properties

Soil properties, including excessive acidity or basicity, excessive salinity, and soil texture or particle size, are among the most difficult reclamation-related constraints to control. Excessive soil pH of the tailings material may restrict germination or reduce vegetation growth. Excessive salinity may restrict or limit revegetation success. These problems of pH and salinity are difficult to control. A combination of problems (e.g., pH and salinity problems in combination) may be even more difficult to control and may place additional constraints on the reclamation potential on the tailings. Neutralizing amendments and leaching may offer some control of these problems. Natural weathering will, in time, provide some improvement in some of these conditions.

The loose sandy nature of many tailings affect revegetation because these sandy soils are easily blown about by wind. As these wind-blown materials move across the surface they create a "sand-blasting" effect which may result in physical damage to or destruction of vegetation. This type of wind erosion can be temporarily controlled by chemical soil binders.

Biological Sterility

Milling processes such as those used in copper production result in tailings which are biologically sterile. These tailings are essentially devoid of organic matter and lack the nutrients and microbial populations necessary to support and sustain plant growth. For many tailings, biological sterility can be controlled through the application of appropriate fertilizers, organic materials or other suitable soil amendments and the gradual buildup of microbial populations in the developing soils.

Toxic Materials

Toxic materials at levels high enough to effect plant growth are a potential constraint to successful revegetation. The availability of toxic elements such as iron, manganese, and copper, are important since toxic levels of these elements may hinder revegetation efforts.

APPENDIX B

PRELIMINARY EVALUATION OF COPPER-NICKEL PROJECT

TAILINGS EMISSIONS AND ASSOCIATED AIR QUALITY IMPACT

Performed
for

State of Minnesota

Environmental Quality Council

by

NUS Corporation
4 Research Place
Rockville, Maryland 20850

March 1978

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Environmental Safeguards Division

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I. Introduction

A preliminary air quality analysis has been conducted to evaluate the impact of fugitive particulate emissions from tailings storage which would be associated with development of copper-nickel sulphide resources in northeastern Minnesota. The analysis has been based on a scenario postulated for one hypothetical mine site and, therefore, results from this evaluation are generic. However, these results are also applicable, subject to judicious interpretation, to specific potential mine sites in the region.

The specific objective of this analysis has been to obtain preliminary estimates of the incremental increase in suspended particulate concentrations in the region due to emissions from tailing storage at one hypothetical site. Therefore, neither the impact of other sources (including other fugitive emission sources which would be expected at a mining site, in addition to tailing emissions), nor a detailed evaluation of present background air quality levels in the region have been addressed.

The air quality impact analysis can be characterized as conservative in nature (i.e., based on assuming pessimistic conditions with a reasonable probability of occurrence). Results should be considered as engineering estimates since there are presently large uncertainties associated with the estimation of emission rates for fugitive dust sources such as tailing piles.

II. Background Technical Information

A. Regional Meteorology

The copper-nickel resource region of interest is located near Ely in the Mesabi Iron Range of northeastern Minnesota. The resource area is approximately 1 mile wide and 40 miles long. This area is approximately 80 miles north of Duluth and 80 miles southeast of International Falls. The area being considered for tailing disposal extends for about 12 miles on each side of the resource area.

The terrain in the region can be considered as irregular plains with local relief of about 100 feet and, therefore, topography has not been considered a significant factor for this study. The resource area is located within the Superior National Forest. Therefore, much of the region is heavily wooded although there are swamplands in low lying areas. The northern portion of the resource area is in close proximity to the Boundary Waters Canoe Area.

The climate of the region can be considered continental, subject to frequent polar air outbreaks throughout most of the year. Precipitation is moderate with maximum monthly normals during the summer. During the winter the ground is usually snow covered with frost penetration in the ground ranging from a few inches to 60 inches.(1)

The climatic normals and dispersion conditions for the region can be characterized by data from International Falls. Temperature, precipitation and wind normals are summarized in Table 1.(1)

TABLE 1

Climatic Normals

International Falls, Minnesota[1]

	Temperature (°F)			Precipitation	Wind		Mean No. Days	
	Daily Maximum	Daily Minimum	Monthly	(inches)	Mean Speed (mph)	Prevailing Direction	Precipitation ≥0.1 inch	Snow ≥1 inch
January	12.8	-9.1	1.9	0.85	9.2	W	12	4
February	19.4	-5.5	7.0	0.71	9.1	W	9	3
March	32.3	8.9	20.6	1.10	9.5	W	10	3
April	49.1	27.3	38.2	1.67	10.5	NW	10	2
May	62.5	37.7	50.1	2.75	10.1	NW	12	<0.5
June	72.4	48.3	60.4	3.91	8.7	SE	13	0
July	78.2	53.4	65.8	3.98	8.0	W	11	0
August	75.5	50.9	63.2	3.39	7.7	SE	12	0
September	64.2	41.7	53.0	3.32	8.8	SE	11	<0.5
October	54.0	32.9	43.5	1.69	9.5	SE	9	1
November	32.5	17.3	24.9	1.30	9.9	W	11	3
December	18.1	-0.8	8.7	0.98	9.1	W	12	3
Annual	47.6	25.3	36.5	25.65	9.2	W	133	18

The seasonal and annual frequency of occurrence of stability conditions are summarized in Table 2[2]. Neutral conditions predominate throughout the year. The occurrence of high wind speed episodes are also generally associated with neutral stability.

B. Hypothetical Meteorology

Hypothetical meteorological conditions have been postulated to be used as input for calculating particulate emission rates and atmospheric dispersion factors. The two averaging time frames of interest are for 24-hours and annually in order to assess impacts relevant to air quality standards for these periods.

Annual average meteorological conditions which have been postulated are summarized in Table 3. The predominant stability condition ("D" - neutral stability) and normal wind speed (9.2 mph) for International Falls, as presented in Tables 1 and 2, have been selected for the annual computations. The frequency of occurrence of winds in the hypothetical sector of interest was assumed to be 10% (i.e., the approximate values for the prevailing wind direction, west, for the region).[3]

Neutral conditions were also assumed to occur during the 24-hour calculational period. As previously stated, high wind speed conditions are generally associated with neutral stability. An examination of 1970 - 1974 meteorological summaries for International Falls provided to NUS by MEQC, indicate that the wind speed class of 20-24 mph was the highest

TABLE 2

Frequency of Occurrence of Atmospheric Stability Categories (%)
International Falls, Minnesota[2]

	Unstable	Neutral	Stable
Spring	0-5	56-65	26-35
Summer	6-15	46-55	26-35
Fall	6-15	66-75	26-35
Winter	6-15	56-65	26-35
Annual	0-5	56-65	26-35

TABLE 3

Hypothetical Atmospheric Dispersion Conditions

Period	Stability	Wind Speed (mph)	Frequency of Winds in Sector of Interest (%)
Annual	D	9.2	10
24-Hours	D	22.0	100

category for which at least 24 hours of occurrence per year (though not necessarily sequential) have been reported for any one sector. For conservatism, winds were assumed to flow 100 percent of the time during the 24 hour period of interest.

C. Tailings Storage Scenario

Tailings will consist of a combination of waste material and low grade ore (which presently is not economical to recover). The tailings will consist mainly of fines as indicated in the particle size distribution presented in Table 4. These particles will have a density of 2.5 to 2.7 g/cm^3.

The tailings will be discharged in the form of a wet slurry onto disposal/storage piles 100 feet high. The total area required for these tailings at a hypothetical site has been postulated to range from 1,000 to 5,000 acres. The configuration of the tailings area will vary depending on site-specific factors, especially topography. For conservatism in estimating emission factors, a dry tailings area of 5,000 acres has been used, although portions may actually be covered with water. However, results can be scaled to different area sizes as necessary.

Specific mitigating measures to be used to control fugitive emissions from the tailings have not been determined at this time. Therefore, the efficiency and costs of alternative control methods will be explored as a part of this analysis.

TABLE 4

Particle Size Distribution (by weight) of Tailings

Particle Diameter (μm)	Percent $\leq$ Indicated
200	100
100	73
50	48
30	34
20	28
10	18
5	12
2	7
1	5

III. Emission Factors

A. Background Theory

Particulate emissions from the tailings are considered "fugitive" since they are not discharged to the atmosphere in a confined flow stream. Fugitive particulate emissions are generated by the following two basic physical phenomenon:[4]

i) Pulverization and abrasion of surface materials by application of mechanical force implements (wheels, blades, etc.)

ii) Entrainment of dust particles by the action of turbulent air currents.

Because of the slurry pipeline method of discharge to the tailing's pile, mechanical disturbance of the stored material by men or machinery will be minimal. Therefore, wind erosion is considered the predominant factor in the generation of fugitive emissions for this study.

As previously indicated, particles on the earth's surface are set in motion by aerodynamic forces (wind) and/or mechanical distrubances (the actions of man and/or animals or as a result of impinging particles). Once particle motion starts, movement continues in three modes of transport: (1) surface creep, (2) particle saltation and (3) airborne suspension.[5]

Surface creep is characterized as the rolling and sliding of particles (generally $>1,000\ \mu m$) where the aerodynamic or mechanical forces fail to exceed gravitational forces. Particle saltation is characterized by particles (generally 50-1,000 μm) which after injection into the air travel only about 10m due to their large mass. However, the impingement of these saltation particles at the surface results in dislodgement of other particles similar to a sandblasting effect. Airborne suspension is characterized by particles (generally < 50 μm) whose settling velocity is less than the average vertical fluctuations of the surface wind. These particles are frequently injected into the air by saltation effects and can travel large distances.

The primary natural mechanism for suspension and resuspension of small particles (i.e., < 50 μm) is due to mechanical disturbances caused by saltation. Dust-sized particles are generally not directly transported into the air by aerodynamic forces since the drag forces are small for any individual particle due to its size and occurrence within the laminar boundary layer at the ground surface. Also these small particles are frequently shielded from the wind by larger particles.[6] However, small aggregates of these particles can act as effective larger sized particles and, therefore, be subject to aerodynamic forces. These aggregates can be transported by the wind but also tend to be disintegrated back to dust. The net result is generally only minimal transport due to aerodynamic forces of the total amount of small particles available at the ground surface.[7]

Saltation is initiated when a threshold wind speed is achieved. The fluid threshold wind speed is defined as the wind speed which results in the moment about the sand grain pivot point due to drag forces equalling the moment due to the particle weight. The resulting fluid threshold wind speed can be calculated as follows:[6]

$$u_t = \frac{a}{k} \sqrt{\frac{\varrho_p - \varrho_a}{\varrho_a} G \quad d} \quad \ln\left(\frac{z}{z_o}\right) \qquad \text{(Equation 1)}$$

where:

u_t	=	Fluid threshold wind speed (m/sec)
a	=	A constant ($\approx$0.1 for air)[6]
k	=	von Karman's constant (0.4)
ρ_p	=	Density of particle (g/m^3)
ρ_a	=	Density of air (g/m^3)
G	=	Gravitational acceleration (m/sec^2)
d	=	Sand particle diameter (m)
z	=	Height of wind measurements (m)
z_o	=	Roughness height (approximately 0.01 m during saltation)[6]

The above equation is not applicable to suspensible size particles.[6]

It should be noted that wind erosion is generally not considered to be significant for wind speeds less than approximately 5 m/sec.[4] Computed values of the fluid threshold wind speed for this study range between 1 to 2 m/sec. However, as will be discussed later, the relative wind erosion rate of these low wind speeds are very small.

Particle diameters used in Equation 1 are based on the concept of "equivalent" diameters. The equivalent diameter accounts for the irregularities in the shape of particles and the resulting aerodynamic effects. It can be defined as the size of a sphere which would have similar aerodynamic characteristics as the actual particle of interest. For desert sand the mean particle diameter is multiplied by a factor of 0.75 to obtain the equivalent diameter.[8]

A range of particle sizes can be expected for mixtures such as natural sand. The lowest threshold wind would be associated with the smallest particle sizes. However, the resulting saltation will generally only be temporary as particles of the required size are transported away and become unavailable for future saltation. For areas with large fetches this may not be a restriction. The most common threshold velocity used for saltation estimates is the "initial" threshold wind velocity. This is the threshold associated with the predominant diameter.[6,7]

The fluid wind speed threshold, as discussed above, is associated with natural distrubance (i.e., due to the wind) of the particles at the ground surface. An "impact" threshold wind speed is defined for situations where the surface is artificially disturbed. Saltation will be maintained downwind of the artificial disturbance for winds at or greater than the impact threshold speed. For particles of greater than 250 μm the impact threshold is approximately 0.80 times the fluid threshold. For particles of smaller diameter the impact threshold approaches the fluid threshold.[7]

The rate of saltation can be estimated by equating the momentum lost by the air with that necessary to keep the particles in motion. The resulting expression for the saltation rate is as follows:[6]

$$S = \left(\frac{k}{\ln\left(\frac{z}{z_o}\right)} \right)^3 c \sqrt{\frac{d}{D}} \frac{\rho a}{G} (u-u_t)^3 \text{ for } u \geq u_t \qquad \text{(Equation 2)}$$

and:

$$S = o \text{ for } u < u_t \qquad \text{(Equation 3)}$$

where:

S	=	Saltation rate (g/m-sec)
u	=	Wind speed at reference height z (m/sec)
D	=	Standard particle diameter, 250μm, used for reference purposes (m)[6]
c	=	Experimental constant depending on particle size distribution (nearly uniform sand, c = 1.5; naturally graded sand, c = 1.8; wide range of grain sizes, c = 2.8)[6]
ρ_a	=	Density of air (g/m^3)

The primary mechanism for suspension, as previously discussed, is due to saltation. Therefore, it would be reasonable to assume that the suspension rate is proportional to the saltation rate. This assumption is standardly used for estimation of suspension although only limited field verification exists.[6]

B. Models

There are numerous models and formulations presented in the technical literature to compute particulate transport incorporating the suspension process. Much of the work in this field has been sponsored by EPA (relative to fugitive dust) as well as ERDA and the national laboratories (relative to the suspension of particulate radionuclides). The Wind Erosion Laboratory of the Department of Agriculture has also been a major technical contributor.

A literature survey indicates that, due to the complexity of the problem, present models are not entirely satisfactory in accounting for all of the physical processes involved or maintaining a complete mass balance. There are two basic types of models, physical and empirical, which have been used to describe suspension and resuspension processes. These models are briefly discussed as follows:

Physical models are characterized by an attempt to describe the physics of the suspension/resuspension and atmospheric transport processes in terms of theoretical equations. Description of the physics involved usually involves calculation of horizontal and vertical fluxes of particulates with an attempt to maintain a material balance. Because all of the physical processes involved are not completely understood, the resulting models usually incorporate some empirically derived expressions but are generic in nature. In general, these models are more complex than the completely empirical models making application to specific problems more difficult.

Most of the available models for suspension/resuspension are empirical in nature. Therefore, application of these models may be limited unless extensive experiments were conducted for numerous conditions and locations. However, given the above generic limitations, there have been a few specific types of suspension models which have been identified which have practical application potential. These include flux and wind erosion models.

The flux models generally relate the friction velocity (or equivalent terms) to the measured flux of suspended particles based on field experiments.[2,9,10] Also related to the flux models are resuspension models. The resuspension models predict the resuspension of deposited materials, but they can also be employed to estimate initial suspension rates.

Estimation of wind erosion has been a major concern of the agriculture community for some time. The Wind Erosion Laboratory of the Department of Agriculture has been involved in the development of wind erosion models for agricultural problems applicable to most areas in the United States. These models are also generally applicable to other suspension/resuspension applications. A standard term used to characterize erosion is the soil erosion rate (E) which is a function of several factors as indicated below:[11]

$$\text{Soil Erosion Rate} = E\,(I, R, C, L, V) \qquad \text{(Equation 4)}$$

where:

I	=	Soil erodibility index
R	=	Soil ridge roughness factor
C	=	Climatic factor
L	=	Field (exposure) length factor
V	=	Equivalent vegetative cover factor

The soil erosion rate equation is solved in a stepwise procedure involving graphical solutions.

The wind erosion equation is considered applicable to this study because the same processes and variables which affect the rate of topsoil losses also affect the generation of suspended particulates. A summary of the pros and cons of using this equation are summarized as follows:[12]

Pro

i) The wind erosion equation is based on extensive data and research;

ii) The procedure includes several major parameters which effect the emission rate;

iii) It requires data which are usually readily obtainable.

Con

i) The assumption that a relative constant percent of the total soil losses becomes suspended does not have any substantuating data;

ii) Only limited data are available to provide an estimate of the percent of total soil losses that become suspended;

iii) It is not directly applicable to estimating short-term emission rates.

The researchers who developed the wind erosion equation are not necessarily in agreement with the application of this equation to estimating the suspended emission rate. However, this equation has been used by EPA and by other organizations sponsored by EPA to develop emission factors for fugitive dust sources. Even with the limitations previously discussed, the wind erosion equation can be considered as the best practical method to obtain engineering estimates at this time.

C. Calculations

Fugitive particulate emissions for this study has been estimated using the following modified form of the standard wind erosion equation:[12]

$$E_s = A\,I\,K\,C\,L'\,V' \qquad \text{(Equation 5)}$$

where:

E_s	=	Suspended particulate emission rate (tons/acre/year)
A	=	Fraction of wind erosion losses which are suspended particulates (dimensionless)
I	=	Soil erodibility (tons/acre/year)
K	=	Surface roughness factor (dimensionless)
C	=	Climatic factor (dimensionless)
L'	=	Unsheltered field width factor (dimensionless)
V'	=	Vegetation cover factor (dimensionless)

Emission rates have been calculated for particulates less than 30 μm. The 30 μm value is the effective aerodynamic cutoff diameter for the capture of dust by a standard high-volume filtration sampler. Also, during typical wind speeds of 10 mph, particles larger than 100 μm are likely to settle out within 6-9m from the source and 30-100 μm particles within 100m. Particles less than 10-15 μm are more likely to be suspended for very large distances and particles between 10-30 μm will be deposited at intermediate distances.[4]

The fraction of erosion losses which are considered suspended particulates (A) has been assumed to be 0.34 for this application. This value corresponds to the percent of the tailings which are less than 30 μm in diameter based on Table 1.

The soil erodibility index is a function of the amount of erodible fines (i.e., particles less than 840 μm).[12] For this application all of the material is less than 840 μm. Because there have not been values presented in the reference for material with less than 99 percent fines, the available data have been extrapolated. For the present evaluation a value of 500 tons/acre/year has been estimated for I. The uncertainty for this estimate is probably at least ± 30 percent.

The surface roughness factor, K, is a function of the height and spacing of the ridges and is equal to 1.0 for the essentially smooth surfaces of the tailings area.[12]

The climatic factor, C, is based on the following equation:[12]

$$C = 0.345 \frac{u^3}{(PE)^2} \qquad \text{(Equation 6)}$$

where:

C	=	Climatic factor (dimensionless)
u	=	Mean wind speed (mph)
PE	=	Thornthwaite's precipitation-evaporation index (inches)

Based on an annual average wind speed of 9.2 mph and an annual PE value of 112 inches (Reference 4), an annual value of approximately 0.02 was computed for C. It should be noted this value is somewhat lower than the value of 0.05 presented in USDA references which provide isopleth values of C for the United States.[13] For this study C = 0.02 has been used to represent annual region specific conditions.

A wind speed of approximately 22 mph was used to calculate C during a hypothetical 24-hour period associated with maximum air quality impact This resulted in a value of C = 0.29. It should be noted that the annual PE value (112 for this application) is traditionally used for periods of less than one year even though monthly variations occur.[12]

Wind erosion is directly related to the unsheltered width of the area subject to wind forces. Considering the 100' height of the tailings/site and the large horizontal dimensions involved, no credit has been taken for sheltering effects (i.e., L' = 1.0).[12] In general, tree heights are not sufficient to shelter the large dimensions of the tailings area. However, use of low lying areas between terrain ridges for tailings fill is a potential in the region, and could result in very significant reduction in emissions.

The vegetation cover factor, V', has been assumed to be 1.0 for the study. This implies that the tailings area is barren of vegetation, and therefore no credit can be taken for reduced emissions due to vegetative effects. The use of vegetation covers as a mitigating measure will be discussed in Section V.

Based on the above factors the calculated emission rates are presented in Table 5. As previously discussed, the estimated short-term emission rate used for the study is uncertain since the wind erosion equation was developed for long-term estimates.

TABLE 5

Suspended Particulate Emission Rates
(Particles < 30 μm)

	Tons/acre/year
Annual	3.4
24-Hour Maximum	49.3

IV. Atmospheric Transport Factors

The atmospheric transport of particulate emissions involves dispersion, deposition and plume depletion as well as resuspension. These factors are discussed as follows:

A. Dispersion

Atmospheric dispersion can be characterized for this application by use of standard Gaussian dispersion models for ground level area sources and for ground level receptors. Hypothetical meteorological conditions used as input for these calculations in this study have been presented in Table 3.

As previously discussed, the tailings area will consist of 1,000 to 5,000 acres although a specific configuration will depend on site specific conditions. In order to account for the initial horizontal plume dimension associated with an area source, the width of the tailings pile perpendicular to the wind flow was assumed to be 1,200m.

Short-term (i.e., the 24-hour period) and annual concentrations were computed based on the following 22.5° sector-average dispersion equation (based on Reference 14):

$$\chi = \frac{2.032\ F\ Q}{\sigma_{z_i}(x)\ u\ (x + x')} \qquad \text{(Equation 7)}$$

where:

χ	=	Concentration ($\mu g/m^3$)
Q	=	Emission rate ($\mu g/sec$)
F	=	Frequency of occurrence of winds in sector of interest (expressed as a fraction)

σ_{z_i}	=	Vertical dispersion parameter for stability class i and downwind distance x (m)
u	=	Wind speed for period of interest (m/sec)
x	=	Downwind distance of interest
x'	=	Virtual source distance correction for an area source of 1,200 m width (1,400 m)

Results of the 24-hour and annual computations are presented in Table 6.

B. Deposition and Depletion

As discussed in Section III, particles greater than 30 μm will settle out close to the source while those less than 10 μm are subject to transport for very large distances. Therefore, for this study, deposition is only considered for particles from 10 μm to 30 μm. This amounts to about a factor of 0.5 of the suspended particle distribution from the tailing emissions (i.e., of those particles less than 30 μm) by weight.

The deposition of particles is a removal process which is a function of the particle settling velocity and other complex factors (e.g., impaction, electrical forces, collection efficiency, etc.). The settling velocity due to gravitational forces is as follows:[15]

$$\nu_g = \frac{2r^2 G \rho_p}{9 \mu} \qquad \text{(Equation 8)}$$

where:

ν_g	=	Settling velocity (cm/sec)
r	=	Particle radius (cm)
G	=	Gravitational acceleration (cm/sec^2)
ρ_p	=	Particle density (g/cm^3)
μ	=	Atmospheric dynamic viscosity (g/cm-sec)

3.0 SCOPE, DESIGN CRITERIA AND METALLURGICAL DATA

This study investigated the effect on capital and operating costs of the distance between the tailings pond and concentrator for concentrator capacities of 12.33, 16.68 and 20.00 x 10^6 MTPY. The following design criteria were used:

Number of Tailings Pumping Stations - all pumping from the concentrator in stages where necessary
- the use of booster stations to reduce heads on tailings lines was not considered

Tailings Pumps - all metal fixed speed

Number of Tailings Lines - one operating and one spare surface line were considered for this study
- the use of multiple operating lines was not considered

Tailings Line Material - all tailings lines are of steel construction
- pipelines one mile and greater are insulated and heat traced

Velocities in Tailings Lines - 4.09 - 6.43 feet per second

Reclaim Pumps - Vertical turbine

Number of Reclaim Pumping Stations - one pumphouse at tailings pond

Number of Reclaim Lines - one operating surface line only

Reclaim Line Material - all reclaim lines are of polyethylene construction
- pipelines one mile and greater are insulated and heat traced

4.0 THE THICKENED AND UNTHICKENED METHODS

Flowsheets were prepared for each concentrator capacity illustrating both the thickened and unthickened methods. The three flowsheets are included in Appendix D.

The Unthickened Method

This method involves the collection of scavenger and cleaner scavenger flotation tailings in a pump box located in the concentrator. All- metal tailings pumps then deliver the total tailings to the tailings pond.

The Thickened Method

This scheme involves classification of the scavenger flotation tailings to remove the plus 150 mesh fraction. The minus 150 mesh fraction (cyclone overflow) is combined with the cleaner scavenger tailings and pumped into two parallel caisson type thickeners. Underflow from the thickeners is combined with the cyclone underflow in a pump box for delivery to the tailings pond by all-metal tailings pumps.

Thickener overflow is combined with reclaim from the tailings pond for reuse in the concentrator. It has been assumed for purposes of this study that the thickener overflows are suitable for direct recycle without further treatment.

A study was undertaken to compare the capital costs of the unthickened and thickened methods. Thickened/Unthickened capital cost ratios are summarized in Table 6 and illustrated graphically in Figure 7.

Velocities in Reclaim Lines - 4.4 - 7.76 feet per second

Pipeline Sizes - only standard pipe sizes have been considered

Topography
- assumed to be flat between concentrator and tailings pond
- dam height assumed to be 150 feet

Table 4 - Metallurgical Data

Capacity of Concentrator - MTPY	12.33×10^6		16.68×10^6		20.00×10^6	
Tailings - Wt%	94.86		96.19		96.82	
Concentrator Operating Time - %	95		95		95	
- hours per year	8322		8322		8322	
Tailings Tonnage - MTPH	1405.9		1927.4		2326.4	
- STPH	1549		2124		2564	
S.G. of Solids	3.0		3.0		3.0	
Density of Tailings - % Solids	30.0	50.0	30.0	50.0	30.0	50.0
Pulp Density	1,250	1,500	1,250	1,500	1,250	1,500
Volume of Tailings Slurry - USGPM	16,605	8,303	22,769	11,385	27,486	13,743
Volume of Water in Tailings - USGPM	14,446	6,191	19,808	8,489	23,912	10,248
% of Water Reclaimed	77.6	47.8	77.6	47.8	77.6	47.8
Reclaim Water - USGPM	11,210	2,955	15,371	4,059	18,555	4,899

Table 5 - Typical Screen Analysis Data

	Scavenger Tails	Cleaner Scavenger Tails	Combined Tailings
Wt%	80.06	14.67	94.73
+35 mesh	.4		.3
+48 mesh	1.6		1.4
+65 mesh	4.3		3.6
+100 mesh	9.7		8.2
+150 mesh	13.0	.8	11.1
+200 mesh	21.5	2.3	18.5
+270 mesh	8.8	3.5	8.0
+325 mesh	4.4	2.6	4.1
-325 mesh	36.3	90.8	44.8

Table 6 - Thickened/Unthickened Capital Cost Ratios

Distance Between Concentrator and Tailings Pond	Thickened/Unthickened Capital Cost Ratios		
	12.33×10^6 MTPY	16.68×10^6 MTPY	20.00×10^6 MTPY
1000 feet	3.12	3.20	3.60
1 mile	1.21	1.23	1.80
5 miles	.80	.77	.80
10 miles	.71	.54	.73

For distances of 1000 feet and 1 mile, it was concluded that the unthickened method was preferable and for the 5 and 10 mile distances the thickened method was best from a capital cost point of view.

These conclusions served as the basis for all subsequent cost estimates i.e. the unthickened method for distances of 1000 feet and 1 mile and the thickened method for distances of 5 and 10 miles.

TABLE 6

Conservative Suspended Particulate
Concentration Estimates ($\mu g/m^3$)
[No Controls]

Downwind Distance (m)	Annual	24-Hours
100	3.33×10^{3}	1.98×10^{5}
200	1.70×10^{3}	1.01×10^{5}
500	6.80×10^{2}	4.04×10^{4}
1,000	3.18×10^{2}	1.89×10^{4}
2,000	1.39×10^{2}	2.45×10^{3}
5,000	4.11×10^{1}	2.45×10^{3}
10,000	1.54×10^{1}	9.16×10^{2}
20,000	5.67×10^{0}	3.38×10^{2}
50,000	1.51×10^{0}	8.91×10^{1}

Based on this approach, a settling velocity of 3.1 cm/sec was computed for this application (i.e., for particles from 10 to 30 μm). (As a reference point a settling velocity of 1.2 cm/sec would be computed for similar size particles but with a density of 1.0 g/cm^3 instead of 2.6 g/cm^3.) This value was also assumed to approximate the deposition velocity. The effects of factors such as impaction, etc. as well as the settling velocity are accounted for in the deposition velocity ν_d. Deposition velocity for particles of unit density are presented in Table 7.[16] Values over soil are primarily a factor of the settling velocity. Other factors become significant in vegetation canopies such as forests. Based on density differences and data presented in Table 7, a deposition velocity of 4.9 cm/sec has been postulated for emissions from the tailings for this analysis (i.e., for particles between 10 μm and 30 μm) as illustrated in Equation 9.

$$\nu_d = (\nu_{d'} - \nu_{d''}) + \nu_{d'''} \qquad \text{(Equation 9)}$$

where:

ν_d	=	Effective deposition velocity
$\nu_{d'}$	=	Deposition velocity over forests (20 m high) for particles of unit density (assumed to be 3.0 cm/sec)
$\nu_{d''}$	=	Theoretical deposition velocity for particles of unit density (assumed to be 1.2cm/sec)
$\nu_{d'''}$	=	Theoretical deposition velocity for particles of density = 2.6 g/cm^3 without accounting for surface retention characterisitcs (assumed to be 3.1 cm/sec)

The amount of material deposited on the ground can be estimated by the following relationship (based on Reference 15):

$$\chi_{\text{Ground Surface}} = \chi_{\text{Air}} \; \nu_d \; \Delta t \qquad \text{(Equation 10)}$$

TABLE 7[16]

Deposition Velocities For
Particles of 20μm Diameter With
A Density of 1.0 g/cm^3

	ν_d (cm/sec)
Tilled Soil	1.4 - 1.6
Thin Grass	1.4 - 1.8
Thick Grass	1.5 - 2.0
Tall Thin Grass	1.5 - 2.1
Shrubs	1.5 - 2.7
Forest (10m Height)	2.3 - 2.7
Forest (20m Height)	2.7 - 3.3

where:

$\chi_{\text{Ground Surface}}$	=	Ground concentration ($\mu g/m^2$)
χ_{Air}	=	Air concentration ($\mu g/m^3$)
ν_d	=	Deposition velocity (m/sec)
Δt	=	Time interval of exposure to plume (sec)

The deposition process results in plume depletion. The depletion process can be roughly approximated by assuming a depleted source term Q_x which is a function of downwind distance and can be related to the original source term Q_o. The solution of this relationship is complex and is generally accomplished using numerical techniques. Graphical solutions are available for standard conditions which can be used to obtain results for other situations as follows:(15)

$$\left(\frac{Q_x}{Q_o}\right)_2 = \left(\frac{Q_x}{Q_o}\right)_{Ref}^{\nu_{d_2}/u_2} \qquad \text{(Equation 11)}$$

where:

$\left(\frac{Q_x}{Q_o}\right)_{Ref}$	=	Standard values of depletion assuming wind speed of 1 m/sec and deposition velocity of 1 cm/sec. (Refer to Reference 15 for standard values.)
$\left(\frac{Q_x}{Q_o}\right)_2$	=	Desired values of depletion for specific wind speed and deposition velocities of interest.

Depletion factors assuming "D" stability and a 9.2 mph wind speed for annual conditions and 22 mph for 24 hour conditions have been computed. These values are presented in Table 8 for deposition over bare ground/ grassland and for forest areas.

TABLE 8

Plume Depletion Factors

Downwind Distance (cm)	Annual		24-Hour Episode	
	Grass	Forest	Grass	Forest
100	0.74	0.63	0.89	0.83
200	0.66	0.53	0.85	0.78
500	0.58	0.44	0.81	0.72
1,000	0.51	0.35	0.76	0.66
2,000	0.44	0.28	0.72	0.60
5,000	0.35	0.20	0.66	0.52
10,000	0.23	0.13	0.59	0.44
20,000	0.17	0.06	0.49	0.33
50,000	0.08	0.02	0.37	0.21

C. Resuspension

Potential resuspension of previously deposited particulates can also provide an additional source of fugitive emissions. Most of the resuspension research to date has been associated with application to contamination events involving radioactive materials. Various resuspension models have been developed, but estimates based on these models are highly uncertain. For this application a resuspension factor model has been selected because of its simplicity.

The resuspension factor models are based on the assumption that the air concentration (due to resuspension) is proportional to the concentration at the soil surface as follows:[17]

$$K(m^{-1}) = \frac{\chi_{Air} \quad (\mu g/m^3)}{\chi_{Ground\ Surface} (\mu g/m^2)} \qquad \text{(Equation 12)}$$

Values of K can also be considered functions of soil erodibility, surface roughness, climatic factors, field length and vegetation cover as well as the area and depth of contamination similar to the soil erosion rate term E previously discussed.[17] Reported values of K for Pu range over 11 orders of magnitude ($10^{-2}m^{-1}$ for active conditions to $10^{-13}m^{-1}$ for aged material).[18]

Very low values are generally based on laboratory conditions and not field experiments.[18] However, a typical value used for Pu is $10^{-5}m^{-1}$ during the initial period. This value decays to a steady value of $10^{-9}m^{-1}$ assuming a 30-80 day half life.[18,19] The change of K with time accounts for weathering effects (i.e., leaching and mixing of the contaminant to greater soil depths).

The adequacy of resuspension factors has been frequently criticized because of their empirical nature and because they do not account for all of the physical transport processes involved. A significant deficiency is the general lack of dependence of K values on wind and stability conditions. However, the advantages of the K models are the relative simplicity in using them and the availability of limited field verification data. Also, there are large inaccuracies associated with the use of alternative models.

For this application an initial value of $K = 10^{-5} m^{-1}$ has been assumed which decays with a 50 day weathering half life to a steady value of $10^{-9} m^{-1}$ (the decay refers to weathering effects such as leaching, etc. and not to radionuclide decay). However, it should be noted that this approach is conservative since it does not directly account for depletion of the source material on the ground due to the resuspension and atmospheric transport processes.

The significance of resuspension for this study was determined by computation of the resuspension ratio as follows:[20]

$$R_{\overline{X}} = \frac{\overline{X}_R}{\overline{X}_P} \qquad \text{(Equation 13)}$$

where:

$\overline{X}_P$ = Average concentration due to the original plume
$\overline{X}_R$ = Average concentration due to resuspension

Equation 13 can also be expressed as follows:

$$R_{\overline{X}} = \nu_d K_o \frac{T}{0.693} \left[1 - \exp\left(-\frac{0.693\, t_2}{T}\right)\right] \qquad \text{(Equation 14)}$$

where:

ν_d	=	Deposition velocity (m/sec)
K_o	=	Initial resuspension factor value (m^{-1})
T	=	Half life for weathering (sec)
t_2	=	Time duration since start of release (sec)

Resuspension ratios based on Equation 14 as a function of time are presented in Table 9. The resuspension methodology employed does not account for effects of a forest canopy in a sophisticated manner. The effect of the forest on initial deposition rates is accounted for. However, the effect on resuspension is even more complex and has not been accurately accounted for but the methodology used is probably conservative.

TABLE 9

Resuspension Ratios

Exposure Time	Grassland	Forest Canopy
1 day	2.5×10^{-2}	3.9×10^{-2}
10 days	2.5×10^{-1}	3.9×10^{-1}
100 days	1.5×10^{0}	2.2×10^{0}
1 year	1.9×10^{0}	3.0×10^{0}
20 years	1.9×10^{0}	3.0×10^{0}

V. Mitigating Measures

The emissions and resulting concentrations reported in previous sections have been based on no controls. Potential mitigating measures which can be used include routine watering, chemical stabilization, use of vegetation covers or reduction of surface wind speeds across the exposed surfaces using windbreaks and shelterbelts.

Watering is an effective dust suppressent for only a few hours to several days. The use of watering results in formation of a thin surface crust, but this crust is easily destroyed by movement over the surface or by abrasion from loose particles blown across the surface. Therefore, watering must be accomplished frequently to be an effective control method. However, limitations of the weight of equipment that can be used and the tailings piles generally negates the use of watering trucks. Therefore, elaborate methods such as automatic sprinkling systems or large-wheeled, light weight application vehicles must be used.[21]

Chemical stabilizers also react with tailings to form a protective crust but the time span of effectiveness is significantly longer compared to water treatment. The same limitations for the use of heavy equipment also applies to this control method, although use of aircraft is a potential method for application. There are presently about 65 chemicals which can potentially be used.[21]

Fugitive emissions can also be controlled by physical stabilization such as rock, slag, bark, straw, etc. The practicality of this approach depends mainly on the local availability of these materials and economic considerations.[21]

Vegetative stabilization is a very effective control of fugitive emissions. However, there are problems associated with this method. There is generally resistance to vegetative growth due to excessive salts and heavy metals in the tailings as well as high surface temperatures and lack of water. Also, windblown particles may destroy young plants. These problems can be overcome with a combination chemical-vegetation technique. The chemicals reduce the sandblasting effect and serve to hold water near the surface. Vegetation germination and growth can also be enhanced by use of buried organic layers.[21]

The large size (both area and height) of the tailings area eliminates erection or growth of wind barriers from practical consideration. Generally the sheltered distance downwind from a barrier is 5 to 10 times H (where, H is the height of the barrier).[12] Reduction in emissions for this application may be a factor of 0.60 to 0.90 or greater providing that these exposure criteria are met.[12] The use of natural terrain, topographic features as wind barriers is a possibility in the rolling terrain of the region. But, again, the large dimensions of the tailings area assumed for this analysis negates this approach as an effective control method, although development of the resource region may actually involve several smaller tailings areas. The feasibility and efficiency of using natural wind barriers is highly dependent on local topography and, therefore, must be evaluated on a site specific basis.

The most effective mitigating measure is the reclamation of the tailing by covering with soil and planting with vegetation. Fugitive emissions from such an approach are expected to be negligible.

A summary of the effectiveness and costs for the alternative control methods are presented in Table 10. Detailed evaluations of stabilization methods for tailings are available from the U.S. Bureau of Mines (e.g., References 23-28).

TABLE 10[21,22]

Summary of Alternative Mitigating Measures

Control Method	Control Efficiency	Maintenance	Total Control Cost (per acre*)
Watering	50%	Continual	Not Available
Vegetation	65%	Minimal	\$200 - \$750
Slag Cover	Good	Moderate	\$350 - \$1050
Chemical Stabilization	80%	Moderate	\$150 - \$750
Combined Chemical - Vegetation	90%	Minimal	\$100 - \$270
4-inch Soil Cover and Vegetation	≈100%	Negligible	\$300 - \$650
12-inch Soil Cover and Vegetation	≈100%	Negligible	\$750 - \$1750

* 1973-1974 Cost Data

VI. Discussion of Results

The computed estimates of suspended particulate concentrations presented in Table 6 have been modified to account for depletion, resuspension and alternative mitigating measures using the methodology discussed in Sections IV-B, IV-C and V. Revised concentration estimates were calculated as follows:

$$\chi' = \chi \left[f + (f' D) \quad (1 + R_{\bar{\chi}}) \right] \left[1 - C \right] \qquad \text{(Equation 15)}$$

where:

χ'	=	Concentration which accounts for depletion, resuspension and control measures ($\mu g/m^3$)
χ	=	Concentration as presented in Table 6 ($\mu g/m^3$)
D	=	Depletion factor which is equivalent to $(Q_x/Q_o)_2$ defined in Equation 11 (dimensionless)
$R_{\bar{\chi}}$	=	Resuspension ratio as defined in Equation 13 (dimensionless)
C	=	Control efficiency (percent)
f	=	Fraction of the plume subject to deposition (i.e., particles from 10 to 30 μm)
f'	=	Fraction of the plume not subject to deposition (i.e., particles less than 10 μm)

Results using the above approach are presented in Tables 11 (annual values) and 12 (24-hour values).

Applicable air quality regulations are summarized in Table 13. Although the copper-nickel resource area is located in a Class II Prevention of Significant Detereoration (PSD) region, the Boundary Waters Canoe Area

TABLE 11

Annual Conservative Suspended
Particulate Concentration Estimates ($\mu g/m^3$)
(Assuming Depletion and Resuspension)

	Downwind Distance (m)	Controls: None	Watering	Vegetation	Chemicals	Chemical-Vegetation	Soil Cover-Vegetation
Grasslands	100	5.23×10^{3}	2.63×10^{3}	1.83×10^{3}	1.03×10^{3}	5.33×10^{2}	Negligible
	200	2.48×10^{3}	1.24×10^{3}	8.67×10^{2}	4.93×10^{2}	2.55×10^{2}	Negligible
	500	9.11×10^{2}	4.56×10^{2}	3.20×10^{2}	1.84×10^{2}	8.84×10^{1}	Negligible
	1,000	3.94×10^{2}	1.97×10^{2}	1.37×10^{2}	7.95×10^{1}	3.82×10^{1}	Negligible
	2,000	1.58×10^{2}	7.92×10^{1}	5.56×10^{1}	3.20×10^{1}	1.53×10^{1}	Negligible
	5,000	4.15×10^{1}	2.06×10^{1}	1.44×10^{1}	8.22×10^{0}	4.11×10^{0}	Negligible
	10,000	1.28×10^{1}	6.47×10^{0}	4.47×10^{0}	2.62×10^{0}	1.23×10^{0}	Negligible
	20,000	4.25×10^{0}	2.10×10^{0}	1.47×10^{0}	8.51×10^{-1}	3.97×10^{-1}	Negligible
	50,000	9.36×10^{-1}	4.68×10^{-1}	3.32×10^{-1}	1.81×10^{-1}	9.06×10^{-2}	Negligible
Forests	100	5.86×10^{3}	2.93×10^{3}	2.06×10^{3}	1.17×10^{3}	5.99×10^{2}	Negligible
	200	2.65×10^{3}	1.33×10^{3}	9.35×10^{2}	5.27×10^{2}	2.72×10^{2}	Negligible
	500	9.38×10^{2}	4.69×10^{2}	3.26×10^{2}	1.90×10^{2}	9.52×10^{1}	Negligible
	1,000	3.82×10^{2}	1.91×10^{2}	1.34×10^{2}	7.63×10^{1}	3.82×10^{1}	Negligible
	2,000	1.47×10^{2}	7.37×10^{1}	5.14×10^{1}	2.92×10^{1}	1.53×10^{1}	Negligible
	5,000	3.69×10^{1}	1.85×10^{1}	1.32×10^{1}	7.40×10^{0}	3.70×10^{0}	Negligible
	10,000	1.17×10^{1}	5.85×10^{0}	4.16×10^{0}	2.31×10^{0}	1.23×10^{0}	Negligible
	20,000	3.52×10^{0}	1.76×10^{0}	1.25×10^{0}	6.80×10^{-1}	3.40×10^{-1}	Negligible
	50,000	8.15×10^{-1}	4.08×10^{-1}	2.87×10^{-1}	1.66×10^{-1}	7.55×10^{-2}	Negligible

TABLE 12

24-Hour Conservative Suspended Particulate Concentration Estimates ($\mu g/m^3$) (Assuming Depletion and Resuspension)

	Downwind Distance (m)	Controls: None	Watering	Vegetation	Chemicals	Chemical-Vegetation	Soil Cover-Vegetation
Gra[illegible]nd[illegible]	100	1.90×10^{5}	9.50×10^{4}	6.73×10^{4}	3.76×10^{4}	1.98×10^{4}	Negligible
	200	9.49×10^{4}	4.75×10^{4}	3.33×10^{4}	1.92×10^{4}	9.09×10^{3}	Negligible
	500	3.72×10^{4}	1.86×10^{4}	1.29×10^{4}	7.27×10^{3}	3.64×10^{3}	Negligible
	1,000	1.68×10^{4}	8.51×10^{3}	5.86×10^{3}	3.40×10^{3}	1.70×10^{3}	Negligible
	2,000	7.19×10^{3}	3.64×10^{3}	2.48×10^{3}	1.41×10^{3}	7.44×10^{2}	Negligible
	5,000	2.06×10^{3}	1.03×10^{3}	7.11×10^{2}	4.17×10^{2}	1.96×10^{2}	Negligible
	10,000	7.33×10^{2}	3.66×10^{2}	2.56×10^{2}	1.47×10^{2}	7.33×10^{1}	Negligible
	20,000	2.54×10^{2}	1.28×10^{2}	8.79×10^{1}	5.07×10^{1}	2.70×10^{1}	Negligible
	50,000	6.15×10^{1}	3.12×10^{1}	2.14×10^{1}	1.25×10^{1}	6.24×10^{0}	Negligible
[illegible]est	100	1.84×10^{5}	9.31×10^{4}	6.53×10^{4}	3.76×10^{4}	1.78×10^{4}	Negligible
	200	9.19×10^{4}	4.55×10^{4}	3.23×10^{4}	1.82×10^{4}	9.09×10^{3}	Negligible
	500	3.51×10^{4}	1.78×10^{4}	1.25×10^{4}	6.87×10^{3}	3.64×10^{3}	Negligible
	1,000	1.59×10^{4}	7.94×10^{3}	5.67×10^{3}	3.21×10^{3}	1.51×10^{3}	Negligible
	2,000	6.70×10^{3}	3.39×10^{3}	2.32×10^{3}	1.32×10^{3}	6.62×10^{2}	Negligible
	5,000	1.89×10^{3}	9.56×10^{2}	6.62×10^{2}	3.68×10^{2}	1.96×10^{2}	Negligible
	10,000	6.69×10^{2}	3.30×10^{2}	2.38×10^{2}	1.37×10^{2}	6.41×10^{1}	Negligible
	20,000	2.26×10^{2}	1.15×10^{2}	8.11×10^{1}	4.39×10^{1}	2.37×10^{1}	Negligible
	50,000	5.44×10^{1}	2.67×10^{1}	1.87×10^{1}	1.07×10^{1}	5.35×10^{0}	Negligible

TABLE 13

Applicable Air Quality Regulations for Suspended Particulates ($\mu g/m^3$)

Averaging Time	Air Quality Standards		PSD Increments	
	Primary	Secondary	Class I	Class II
Annual	75	60	5	19
24-Hour	260	150	10	37

just to the north of the resource area is considered Class I. Also, the Mesabi Iron Range region which borders the western edge of the resource area and includes potential tailings disposal sites for copper-nickel resource development, is considered a non-attaining area.

The limiting air quality regulations for tailings impact are the allowable Class I and II PSD increments and the nonattainment classification of the Mesabi Iron Range. The nonattainment classification would require a program for offset emission reductions in the region or the "complete" control of fugitive emissions from the tailings. Even with most mitigating measures (other than the soil cover and vegetative approach) the 24-hour Class I and II PSD increments are predicted to be exceeded at relatively large distances from the source. However, conformance can be expected to annual PSD increments within 5-10 kilometer from the source for most control methods.

In summary, the optimum control measure would be reclamation of the tailings area by use of a soil covering and planting vegetation. Soil availability and costs may be limitation for this approach, but unnatural fugitive emissions would be essentially eliminated and conformance to air quality regulations can be assured. Depending on site specific conditions, a combination of chemical-vegetation mitigating measure plus the use of natural wind barriers (such as terrain features) is an alternative possibility. However, this second approach would probably require emission offsets in the Mesabi Iron Range because of the nonattainment classification.

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APPENDIX C

THE MINNESOTA ENVIRONMENTAL QUALITY BOARD

COPPER - NICKEL PROJECT

THE EFFECT OF TAILINGS POND DISTANCE FROM THE CONCENTRATOR ON CAPITAL AND OPERATING COSTS

PREPARED BY:

Kilborn Limited
Consulting Engineers
36 Park Lawn Road
Toronto, Ontario
M8Y 3H8

April, 1978

TABLE OF CONTENTS

Not included
N.A.S
Golder
4 May 78

1.0 INTRODUCTION

In a letter dated January 27, 1978 from David L. Veith, Technical Assessment Team Leader for the Minnesota Environmental Quality Board's Copper-Nickel Project, Golder Associates were invited to submit a proposal for tailings disposal systems for potential mining/concentrating operations in the northeastern section of the state.

Shortly after receiving the invitation, Golder Associates asked Kilborn Limited and NUS Corporation to form a project team for the study. A joint Golder Associates/Kilborn Limited/NUS Corporation proposal "An Evaluation of Tailings Disposal Impacts Expected in the Development of Copper Nickel Sulphide Resources, Duluth, Minnesota" was submitted to the MEQB in mid-February and on the 20th, Golder Associates were notified that the proposal in modified form had been accepted.

Kilborn's portion of the study involved the determination of order of magnitude capital and operating costs (in $U.S.) of tailings disposal systems (excluding tailings ponds) for concentrator capacities of 12.33, 16.68 and 20.00 million metric tons of ore per year. Distances of 1000 feet, 1 mile, 5 miles and 10 miles were selected as distances between the concentrator and tailings pond. Two methods of handling flotation tailings were investigated - i.e. the unthickened method and the thickened method.

2.0 SUMMARY

Capital Costs

Order of magnitude capital costs for twelve alternative tailings disposal and reclaim pumping systems are summarized in Table 1 and illustrated graphically in Figures 1 and 2.

Table 1 - Capital Costs

Capacity Of Conc. MTPY	Distance Between Concentrator and Tailings Pond	Cost-$ Total	Cost-$ Per Foot	% of Total Cost of Tailings Line
12.33×10^6	1000 feet	1,593,000	1593	43.4
" "	1 mile	4,936,000	935	66.3
" "	5 miles	16,523,000	626	51.6
" "	10 miles	31,555,000	598	65.1
16.68×10^6	1000 feet	1,947,000	1947	42.6
" "	1 mile	5,885,000	1115	65.8
" "	5 miles	19,942,000	755	51.9
" "	10 miles	33,541,000	635	61.2
20.00×10^6	1000 feet	2,111,000	2111	46.2
" "	1 mile	6,534,000	1239	69.6
" "	5 miles	26,315,000	997	58.7
" "	10 miles	44,930,000	851	68.2

As indicated in Table 1, the tailings line represents 40-70% of the total capital costs.

Details of the capital costs are included in Appendix A.

The capital cost estimates exclude- all civil works for the concentrator
- federal and state taxes
- escalation beyond the first half of 1978
- contingencies
- spare parts
- working capital
- engineering, procurement and construction management
- owner's costs
- land acquisition and right-of-way costs
- licences
- legal costs
- winter working costs
- incoming power lines and roads to the site
- tailings ponds and decant structures
- construction camp

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Figure 1 - Total Capital Cost vs Distance

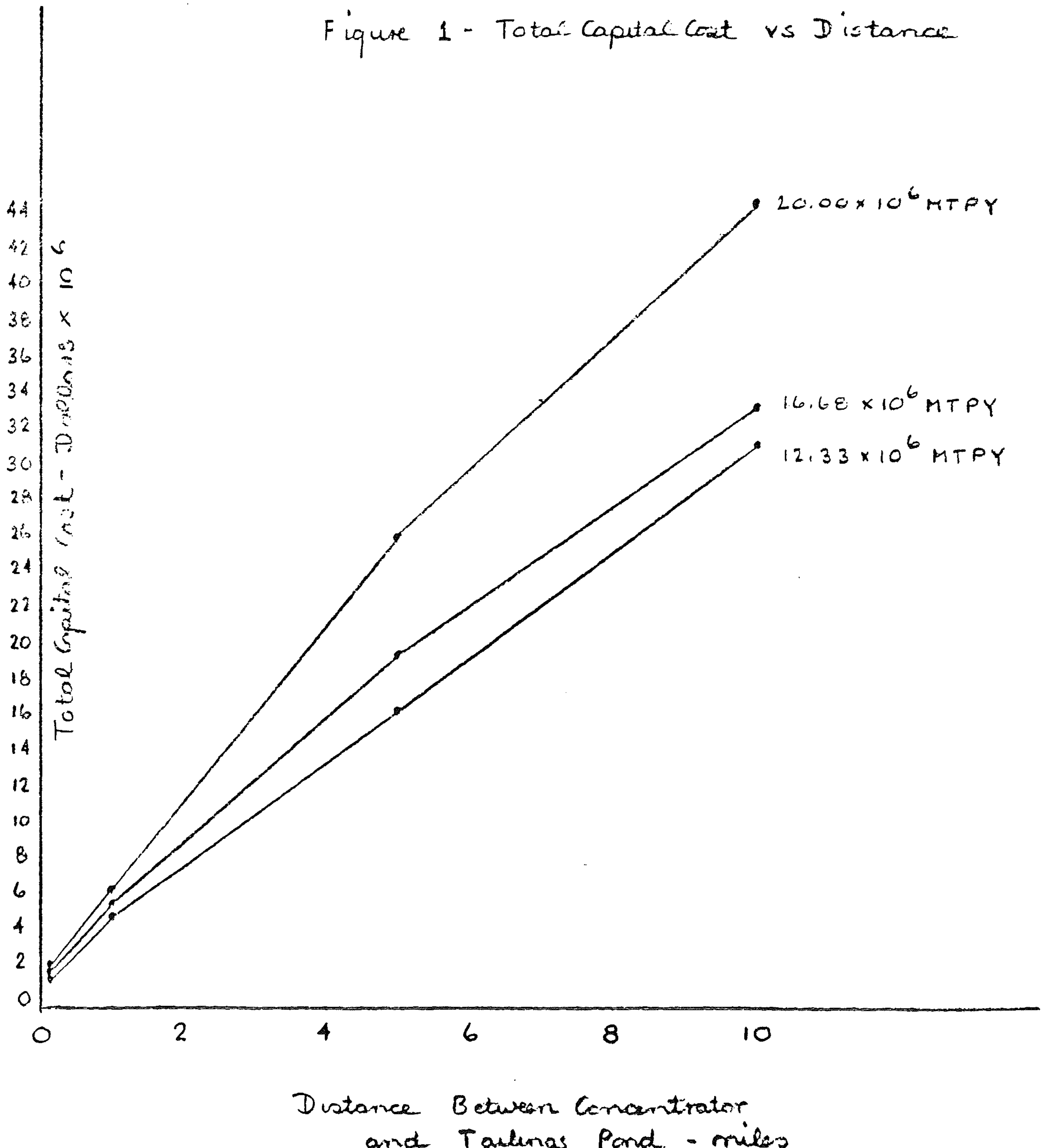

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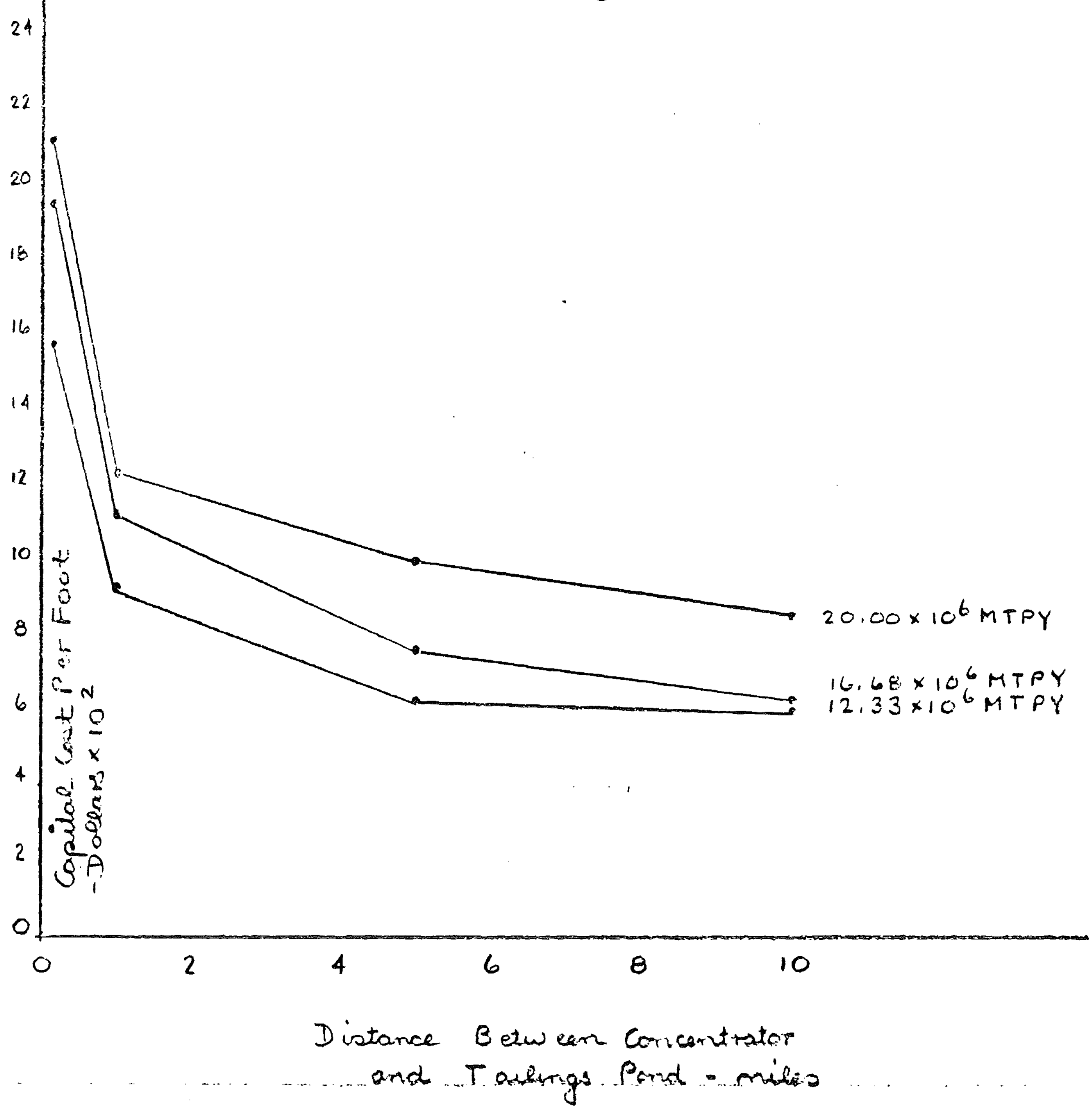
Figure 2 - Capital Cost Per Foot vs Distance



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Operating Costs

Order of magnitude operating costs for the twelve systems are summarized in Table 2 and illustrated graphically in Figures 3 and 4. Details are included in Appendix B.

Table 2 - Operating Costs

Capacity of Concentrator MTPY	Distance Between Concentrator and Tailings Pond	Cost - $ Annual	Cost - $ Per Foot
12.33×10^6	1000 feet	265,945	266
" "	1 mile	449,770	85
" "	5 miles	1,304,247	49
" "	10 miles	1,922,217	36
16.68×10^6	1000 feet	350,105	350
" "	1 mile	557,730	106
" "	5 miles	1,561,975	59
" "	10 miles	2,292,625	43
20.00×10^6	1000 feet	346,665	347
" "	1 mile	564,050	107
" "	5 miles	1,865,537	71
" "	10 miles	2,756,827	52

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Figure 3 - Annual Operating Costs vs Distance

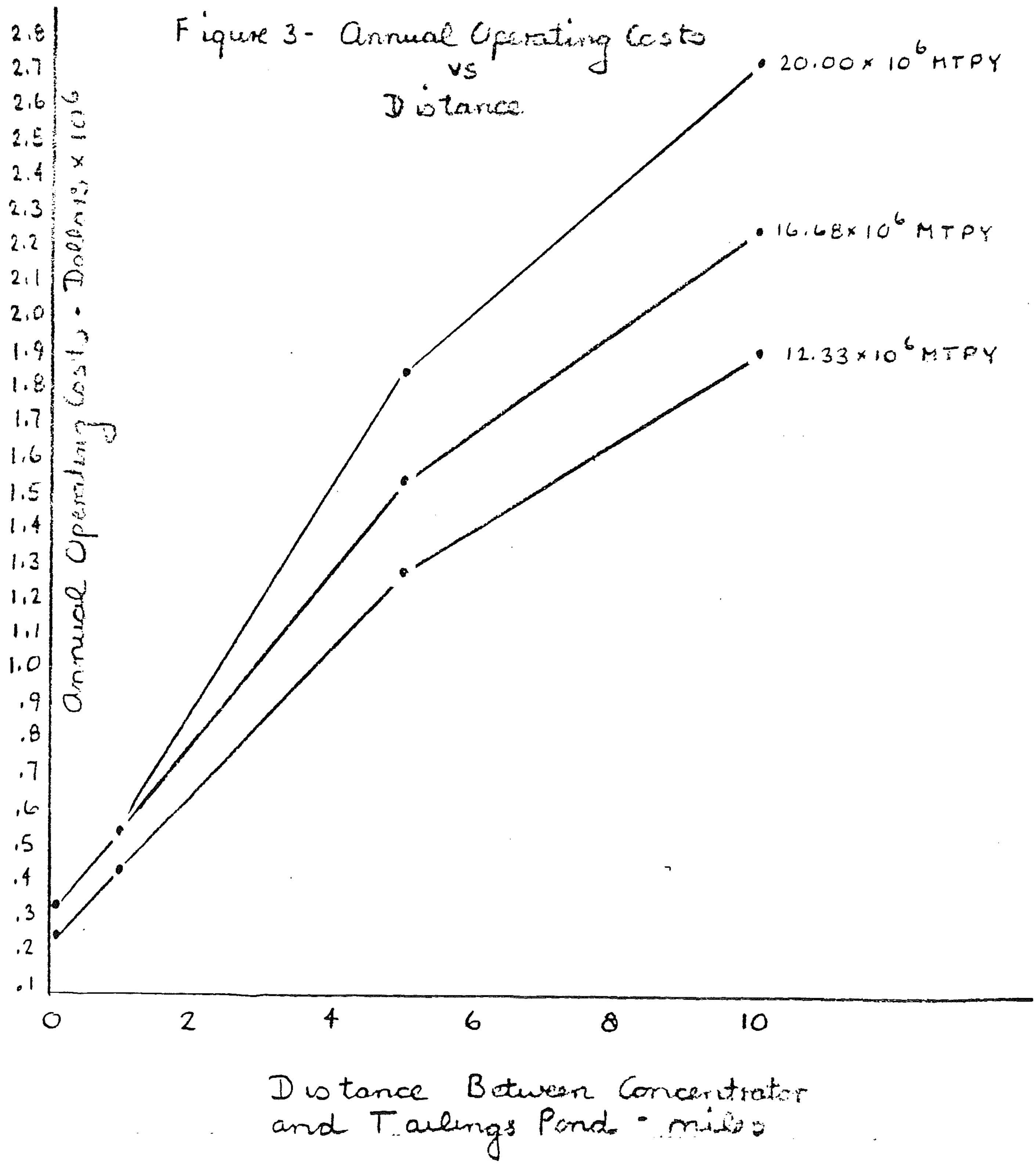

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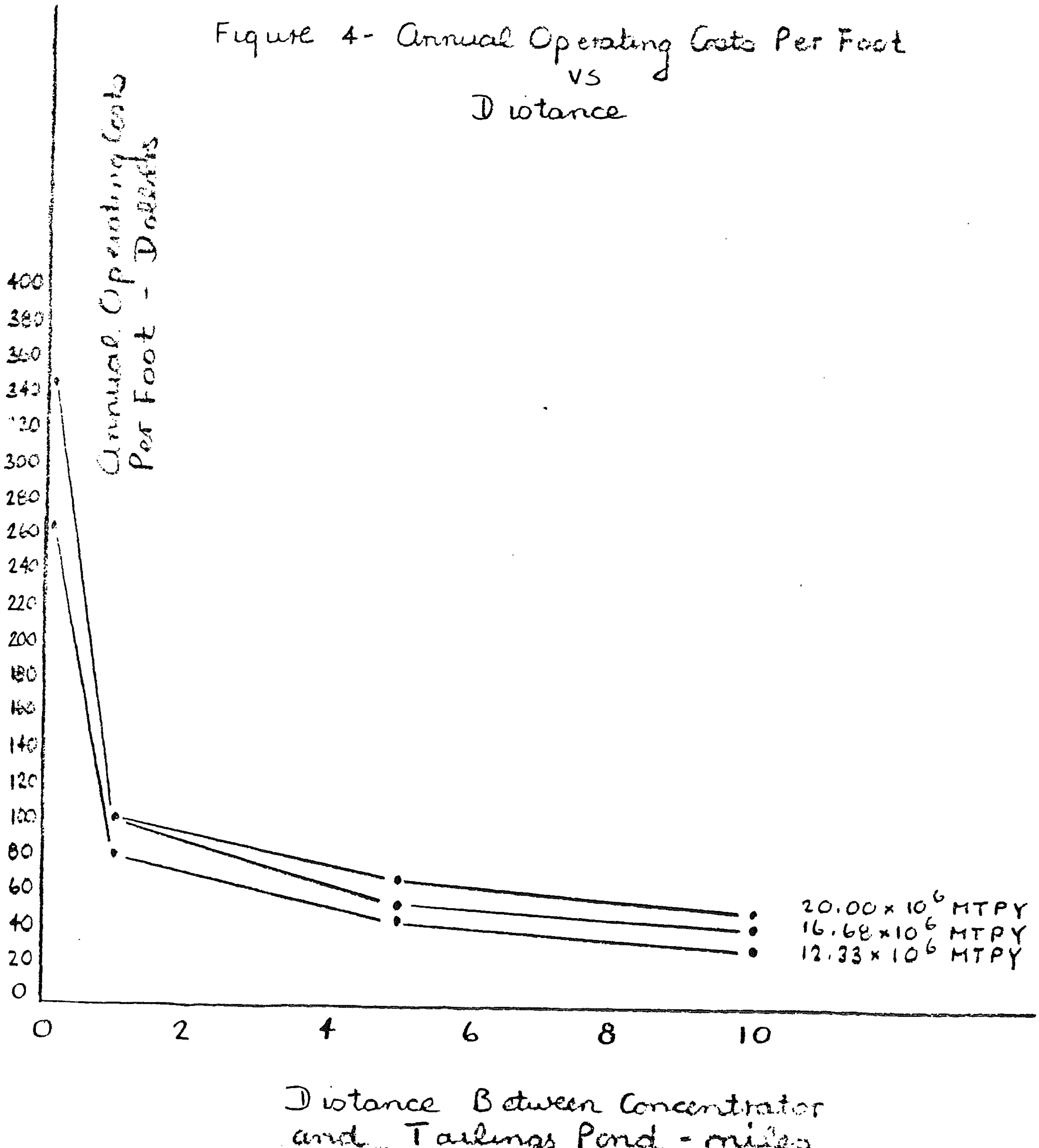
Figure 4 - Annual Operating Costs Per Foot vs Distance



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Power Requirements

Order of magnitude power requirements for the twelve alternative systems are summarized in Table 3 and illustrated graphically in Figures 5 and 6.

Table 3 - Power Requirements

Capacity of Concentrator MTPY	Distance Between Concentrator and Tailings Pond	Annual Power Requirements KWH	KWH Per Foot
12.33×10^6	1000 feet	9,810,000	9810
" "	1 mile	12,590,000	2384
" "	5 miles	26,330,000	997
" "	10 miles	28,620,000	542
16.68×10^6	1000 feet	13,340,000	13340
" "	1 mile	16,220,000	3072
" "	5 miles	30,430,000	1153
" "	10 miles	40,900,000	775
20.00×10^6	1000 feet	12,750,000	12750
" "	1 mile	15,040,000	2848
" "	5 miles	32,190,000	1219
" "	10 miles	40,390,000	765

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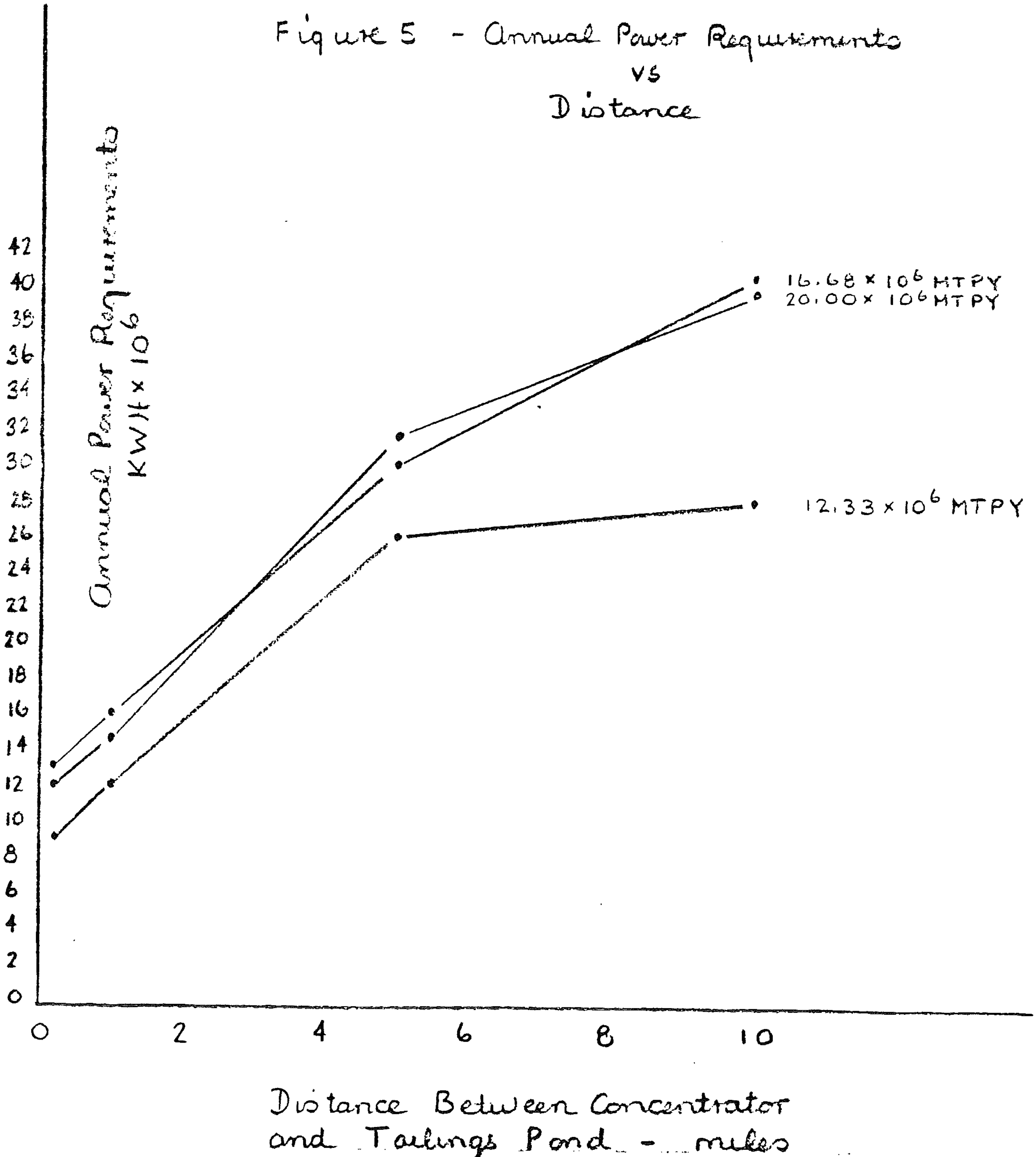

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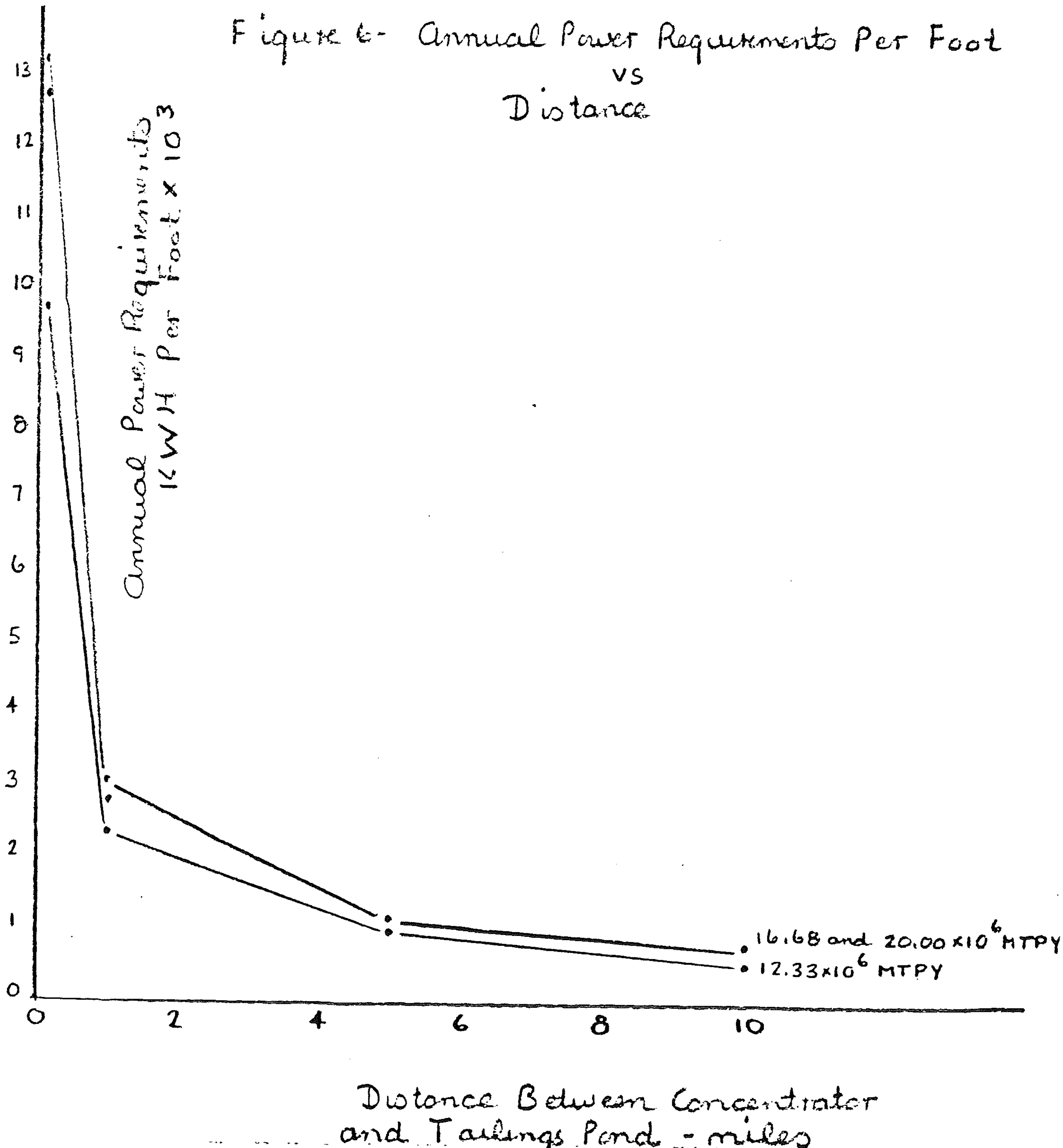

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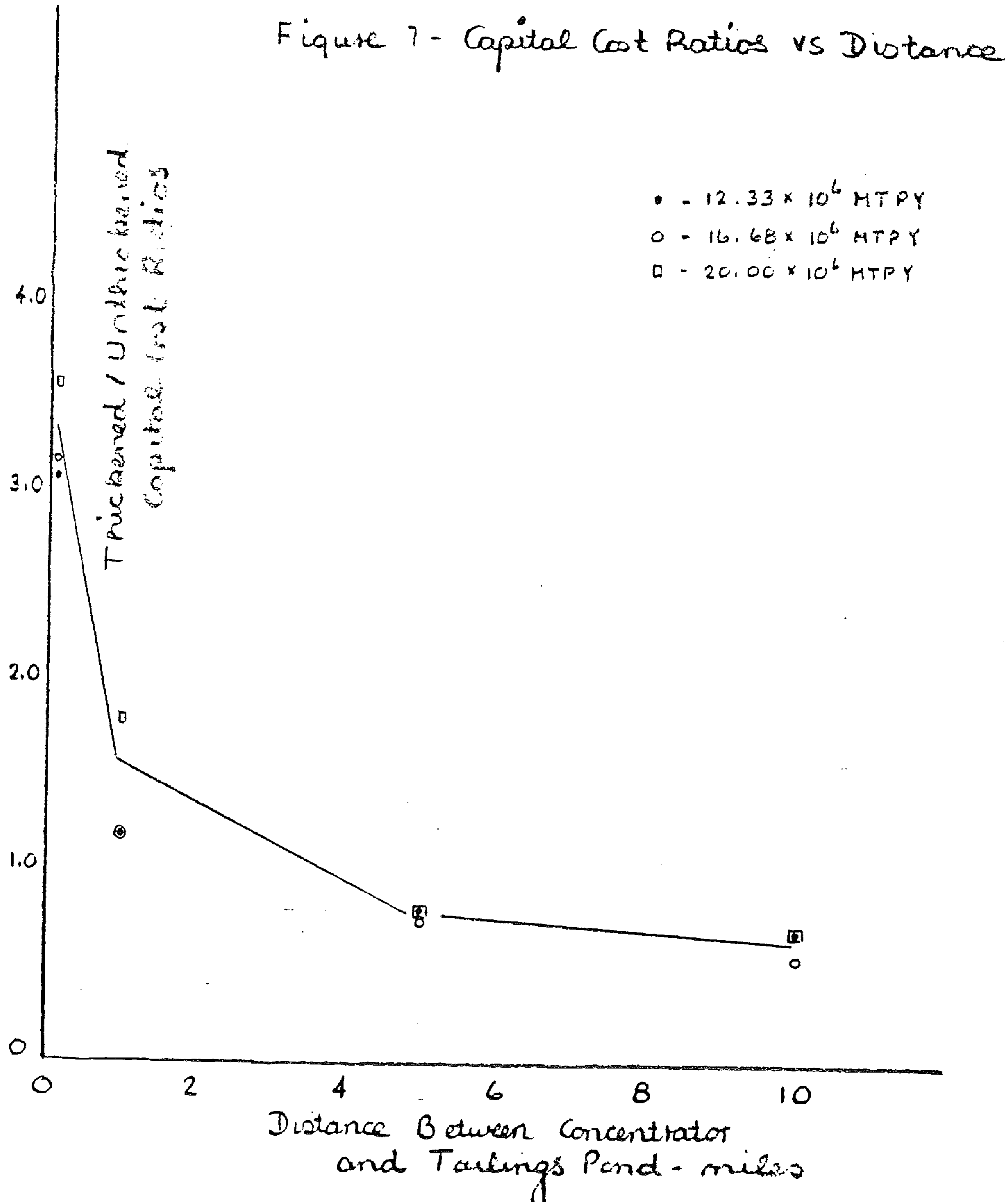
Figure 7 - Capital Cost Ratios vs Distance



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