OIL SAND RECLAMATION

A Study Integrating Mining, Tailings Disposal and Reclamation

Prepared for the

Alberta Department of the Environment

Volume 1 – Text

Prepared Jointly by

Techman Ltd.

and

Rheinbraun – Consulting GmbH

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September, 1979

Alberta Department of the Environment
Oxbridge Place
9820 - 106th Street
Edmonton, Alberta
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Dear Mr. Thiessen:

Techman Ltd. and Rheinbraun-Consulting GmbH are pleased to provide the Alberta Department of the Environment with two-hundred copies of the report entitled: Oil Sand Reclamation - A Study Integrating Mining, Tailings Disposal and Reclamation. The report consists of three volumes:

Volume I - Text
Volume II - Drawings
Volume III - Yearly Cost Summaries

We have appreciated the highly co-operative and supportive role of Mr. Lang and yourself in administering the project as well as the role of the Steering Committee in providing periodic technical review to the project. It was a pleasure to work with such a large number of professionals having a wide variety of experience in oil sands mining and reclamation.

The emphasis in the study is directed towards suggesting technically feasible solutions to improving the overall reclaimability of oil sands mines as well as estimating the cost of revegetating reclaimable land surfaces. Consequently the scope of the report is rather broad, encompassing many aspects of mining, tailings disposal and revegetation. During the course of the study, Techman Ltd. and Rheinbraun-Consulting GmbH prepared twelve mine plans for three actual ore bodies in the Athabasca

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oil sand deposit. Design and cost data are presented in considerable detail throughout the report in order to allow the reader to understand the conclusions of the report.

Techman Ltd. and Rheinbraun-Consulting GmbH jointly express their appreciation for the opportunity to prepare the report and hope that in the near future some of the recommendations for further investigations contained in the report will be followed-up.

Yours truly,
TECHMAN LTD.

D M Morgan

D.M. Morgan, P. Eng.
President,
TECHMAN LTD.

OIL SAND RECLAMATION

A STUDY INTEGRATING MINING, TAILINGS DISPOSAL AND RECLAMATION

VOLUME I - TEXT

Prepared for:

ALBERTA DEPARTMENT OF THE ENVIRONMENT

Prepared Jointly by:
TECHMAN LTD.
&
RHEINBRAUN - CONSULTING GmbH

JULY, 1979

COMMENTS FROM MEMBERS OF THE STEERING COMMITTEE RESPECTING THE FINAL REPORT

The Consultants are pleased to be able to publish the poins of view of the Project Steering Committee concerning the report entitled: Oil Sands Reclamation - A Study Integrating Mining, Tailings Disposal and Reclamation. It was deemed appropriate to include the opinions of the Project Steering Committee members since a diversity of view points exists as a result of the report's comprehensive nature. This report contains an interpretation of information obtained from a variety of sources including literature, experience of operators in oil sands and other mines, as well as the in-house experience of the Consultants. The comments of the Project Steering Committee follow:

ALBERTA ENVIRONMENT - MR. H.W. THIESSEN

The report entitled Oil Sands Reclamation - A Study Integrating Mining, Tailings Disposal and Reclamation was prepared for Alberta Environment by Techman Ltd. and Rheinbraun-Consulting GmbH, and has been reviewed by the Project Steering Committee chaired by Mr. H.W. Thiessen of Alberta Environment. The content of this report does not necessarily reflect the views of Alberta Environment or any of the members of the Steering Committee. Any mention of Trade names for commercial products does not constitute an endorsement or recommendation for use.

Alberta Environment wishes to express the view that the Consultants have presented a final report of superior quality. The Department hopes the information in this report will be disseminated among and be given thorough examination by all concerned sectors of industry and government, in order to provide an improved understanding of the problems related to the reclamation of mined oil sands lands.

SYNCRUDE CANADA LTD. - MR. G.L. LESKO

The authors are commended for their effort in conducting an extremely complex and comprehensive study in the development and reclamation of oil sand resources. However, Syncrude Canada Ltd. is not necessarily in full agreement with all the assumptions, interpretations, conclusions and recommendations contained in this report. The wealth of provided information should be used with caution, because some of the conclusions and recommendations are based on untested assumptions and opinions rather than on experience of scientific results.

GREAT CANADIAN OIL SANDS LIMITED - MR. W.L. CARY

The Consultants have done a good job of highlighting the logistical complexity of oil sands developments. Their conclusion that the particular complexity of oil sands developments. Their conclusion that the particular type of excavator selected does not affect the potential for reclamation is important for those outside the industry to comprehend. Care must be taken to use the cost figures presented by the study as rough comparison costs only. This is due to the extensive use of futuristic unproven techniques, omission of many minor costs and the necessarily incomplete treatment of costs required to implement the Improved and Enhanced concepts. The manner of presentation also has a tendency to mask the real values involved. For example, the direct reclamation costs without including planting of trees varies between $4,800 and $25,000 per acre over the twelve cases analyzed. Also at GCOS we do not agree with the Consultants conclusion that one metre of prepared soil is required for self-sustaining forest cover. Examination of actual soil characteristics in the area and the reclamation efforts to date reveal that much less depth is more than adequate. In conclusion, we believe at GCOS that the study will provide a basic understanding of the oil sands mining and reclamation complexities to those who take the time to absorb all details of the report.

SHELL CANADA RESOURCES LTD. - MR. J.E. DAGENAIS

This study is comprehensive, and through the introduction of innovative concepts it can be viewed as a benchmark in its particular field of interest. Some points are raised that are controversial, and Alsands position on them is as follows:

1. Topsoil Thicknes - the thickness selected is considered arbitrary and requires more testing before it can be quantified.

2. Dry Tailings Disposal - This is considered unacceptable on a practical, year-round basis in view of the lack of proven technology to achieve dry tailings in an oil sands plant.

3. Dragline/Bucket Wheel Excavator Cost Comparison - The premise that there exists a threshold to the size of each machine is accepted.

ENERGY RESOURCES CONSERVATION BOARD - MR. N.A. STROM

The ERCB recognizes that the results and conclusions contained in the Oil Sands Development and Reclamation Model study are basically those of the consultant and do not and can not reflect diverse views of individual members of the Steering Committee. Indeed it may be worthy to note that the Steering Committee function was basically that of defining the broad scope of the study and establishing limits to certain areas of investigation as the multi-facted study program proceeded.

In the foregoing context it is believed that the results and conclusions of the study will provide a useful framework for examining the adequacy of forward planning for land reclamation programs of active surface mining bitumen resource operations (i.e. GCOS, Syncrude and the anticipated Alsands operations). The additional lead time of a decade or so until implementation of land reclamation gets under way should provide valuable lead time needed for resolving many current uncertainties such as minimum soil reclamation depth and material bulk handling and transporting schedules. Factors which are determined by site-specific physical conditions and also other factors (i.e. technology and economics) which may vary with time remain to be considerations of specific mine development proposals.

ALBERTA OIL SANDS TECHNOLOGY AND RESEARCH AUTHORITY - MR. F.J. WERTH

In the conclusions to the above study, it is stated that at oil sand mining rates equivalent to a synthetic crude production rate of greater than 80,000 BPCD, the costs associated with a dragline mining operation are higher than those of a mining operation based on bucket whel excavators. This implies that the dragline operation is not the more economical approach to mining for operations of 80,000 BPCD or greater.

As you are undoubtedly aware, Syncrude Canada Ltd. is currently using a dragline mining operation in a project designed to produce in excess of 125,000 BPCD, and the proposed 140,000 BPCD Alsands project is based on dragline mining. Thus, current commercial trend in oil sand mining does not support the conclusion reached in the study. This variance should be recognized and stated within the conclusions from the study. There may well be site specific factors that would lead to the choice of draglines over bucket wheel excavators.

In all of the dragline case studies, the draglines are dumping mined oil sand directly into a hopper. As stated in the report, there are no existing dragline/hopper operations in existence, of the magnitude required for an oil sands project. There may be problems associated with dumping from large buckets directly into a hopper. Impact loadings on the hopper could result in a very specialized design and construction, resulting in high cost. The problem of accurately dumping from a large bucket into a hopper may slow down the cycle time and reduce its overall digging capacity. In addition, the use of a hopper would tie the digging of oil sand directly to the conveying system. Problems with the conveying system would directly affect oil sand mining.

As pointed out in the report, bitumen extraction processes resulting in the production of dry tailings have not passed the pilot plant stage. Whether or not these processes will become commercially viable operations will not be known for several years. Thus, any conclusions made at this time, based on an operation producing dry tailings, are premature.

<u>ALBERTA ENERGY AND NATURAL RESOURCES - MR. F.W. McDOUGALL</u>

The final draft of the final report has been reviewed by the appropriate Departmental personnel and it is felt that the report adequately fulfills the objectives outlined in Alberta Environment Contract No. 77-143.

In regard to the objectives and requirements of the Department of Energy and Natural Resources, this is a valuable report in that it is the first stage in a Regional Development and Reclamation sequence. This will be particularly valuable in planning energy development and land use options.

While the consultants made use of the best data available, it is understood that several major gaps still exist in our information on the tailings produced by the Clark hot water extraction process.

1. <u>Settling Rate of Tailings Sludge</u> -

   How long will the sludge require external support?

2. <u>Environmental Hazard of Tailings Sludge</u>

   What impact would sludge release have on the Athabasca River, the Peace-Athabasca Delta and adjacent waters?

3. <u>Pond Water Balance</u>

   Will the dormant tailings ponds tend to dry out or will they eventually overflow?

These questions reflect concerns of the Department regarding the long term maintenance of the Tailings Pond dykes, the responsibility for indefinite maintenance and the environmental consequences if dyke maintenance is neglected.

At a more specific level, it has been pointed out that the costs of commercial and non-commercial forest planting are about $4,000 and $3,000 per hectare respectively (p. 6-33). Our staff feels that these cost estimates are exaggerated by about 400%.

These additional comments, however, should not detract from what we consider an excellent report and framework for planning in the Fort McMurray Oil Sands.

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FOREWORD

The contract to conduct a reclamation study integrating mining, tailings disposal and reclamation was awarded jointly to Techman Ltd., Calgary, Canada and Rheinbraun - Consulting GmbH, Cologne, West Germany in July, 1977. Negotiations of contract details were completed by the end of 1977 and the contract signed with the Alberta Department of the Environment in January, 1978. The activities of both consultants were coordinated by Techman Ltd. from its Calgary office.

The document provides a comprehensive review of the technical problems facing future oil sands developments and suggests options which will be helpful in formulating oil sands mining, tailings disposal and reclamation plans. By providing in-depth cost analysis for many of the activities occurring in an oil sands mining operation, the sensitivity of these activities with respect to reclamation and overall project economics can be assessed. The report is very timely considering the renewed interest in expanding oil production from the Athabasca oil sand deposit.

The project was under the general administration of Mr. Henry W. Thiessen, P.Ag., Assistant Deputy Minister of the Alberta Department of the Environment and Chairman, Land Conservation and Reclamation Council. Mr. Dennis D. Lang, P.Eng., was responsible for the operational details on behalf of the Alberta Department of the Environment. Overall review was provided by a Project Steering Committee with representation from the provincial government, industry and research agencies. Mr. Thiessen acted as Chairman for the Project Steering Committee.

The following persons served as members of the Project Steering Committee:

M.A. Carrigy, P.Geol., Vice Chairman, Alberta Oil Sands Technology and Research Authority, Alberta Department of Energy and Natural Resources.

W.L. Cary, P.Eng., Manager. Environmental Affairs Great Canadian Oil Sands Limited

R. Goforth, Ph.D., Head Environmental Affairs, Syncrude Canada Limited.

R.A Hursey, Ph.D., R.P.F., Research Manager, Land Systems, Alberta Oil Sands Environmental Research Program.

F. McDougall, Deputy Minister, Renewable Resources, Alberta Department of Energy and Natural Resources.

N. Strom, P.Eng., Manager of Oil Sands, Energy Resources Conservation Board.

F. Wilkin, P.Eng., Manager, Environmental Affairs, Alsands Project Group.

The following persons served as replacements and additions to the original Project Steering Committee members:

Dennis Bratton, P.Eng , Acting Head, Regulated Surface Operations Branch, Land Conservation and Reclamation Division, Alberta Department of the Environment.

J.A. Brennan, Assistant Deputy Minister, Alberta Forest Service, Alberta Department of Energy and Natural Resources. (for F. McDougall)

R. Gorby, P.Eng., Coordinator of Energy Resources Conservation Board Applications, Alsands Project Group. (for F. Wilkin)

D.G. Harrington, P.Ag., Director, Land Conservation and Reclamation Division, Alberta Department of the Environment.

C.W. Leary, Head, Land Reclamation Section, Reforestation and Reclamation Branch. Alberta Forest Service, Alberta Department of Energy and Natural Resources.

G. Lesko, Ph.D., Director, Environmental Affairs, Syncrude Canada Limited. (for R. Goforth)

R. Marvin, P.Ag., Manager, Resource Management and Reclamation Group, Alberta Department of Energy and Natural Resources.

F.M. Ratushny, P.Eng., Manager of Mining and Extraction, Alsands Project Group.

L.R. Turner, P.Eng., Manager, Extraction and Upgrading Processes, Alberta Oil Sands Technology Research Authority, Alberta Department of Energy and Natural Resources.

F. Werth, M.Sc., Senior Economic Analyst, Alberta Oil Sands Technology and Research Authority, Alberta Department of Energy and Natural Resources.

S. Winzer, Senior Staff Specialist, Great Canadian Oil Sands Limited.

P. Ziemkiewicz, Ph.D., Reclamation Research Officer, Scientific and Engineering Services and Research Group, Alberta Department of Energy and Natural Resources.

All design and documentation within this report was jointly prepared by the Consultants. The staff were periodically rotated between the Calgary and Cologne offices. The project acknowledges the contributions of the following Techman Ltd. staff:

L. Teichgraber, P. Eng., Project Manager, who provided overall administrative and technical management for the project for both Techman Ltd. and Rheinbraun - Consulting, GmbH.

J. Hrouda, P. Eng., Mine Design Engineer, who developed details for many aspects of the mine plans, tailings disposal, prepared soil manufacture and mine simulations.

H.J. Leistra, Cost Engineer, who assembled many of the costs developed in the study.

B. Martens, B.Sc., Wildlife Biologist, who assisted in searching documents for revegetation experience and reclamation ideas.

F.D. McCosh, P. Eng., Geological Engineer, who provided much of the background for regional geology, geomorphology, hydrology and geotechnics.

D.R. Mudry, Ph.D., Aquatic Biologist, who provided the aquatic assessments for the selected mine sites as well as managing Techman's Environmental Department through the various phases of the Department's contribution to the study.

V.S. Pobran, P.Eng., Geologist, who assisted in preparing the base geologic data and developing geotechnical concepts for mine planning purposes.

J. Porteous, P. Eng., Minerals Engineer, who assisted in developing process and cost details for tailings pumping and treatments.

D.E. Rose, P.Eng., Chemical Engineer, who provided the plant process technology, and many aspects of tailings disposal system design and costs.

A. Schori, M.Sc., Pedologist, who provided technical review of the reclamation methods contained in the study.

R. Valleau, P.Ag., Pedologist, whose field reclamation experience was used in assessing the problems of oil sands revegetation.

L.E. Watson, B.Sc., Terrestrial Biologist, who examined most of the existing oil sands reclamation literature and worked closely with the engineers to design reclamation techniques and mine plans.

The project acknowledges the technical contributions of the following Rheinbraun - Consulting GmbH staff:

W. Vogt, Ph.D, Dipl.-Berging., Head of Mine Planning Department (RC), who managed the project for RC and contributed to the overall development of mine plans, reclamation concepts and costs.

L. Dilla, Forestry Specialist, Head of Department of Forest Management (RB), who contributed to developing reclamation and revegetation concepts, especially with respect to prepared soil requirements and reforestation.

H. van Leyen, Ph.D, Dipl.Ing., Head of Mechanical Department (RC), whose experience with oil sands mining equipment was put to use in assessing various operating techniques of oil sands mining equipment and directing the work of the Department staff.

C. Loegters, Ph.D, Dipl.Ing., Divisional Director (RB), who acted as project manager during Dr. Vogt's illness and contributed to the early phases of mine planning.

G.G.P. Milde, Ph.D., Chief Geologist and Professor of Hydrogeology (RC), who provided a general review of the groundwater and dewatering requirements for the project.

H. Neffgen, Ing.(grad), Chief Mechanical Engineer (RC), who assumed the leading role in selecting, sizing and costing many items of machinery required for mining and reclamation activities described in the mine plans presented in this study.

E. Petzold, Ph.D, Agriculturalist, Head of Department of Agriculture (RB), who assessed Rheinbraun's reclamation experience in lignite mines in terms of the prepared soil and revegetation requirements of mines in the Athabasca oil sands.

The project further acknowledges the contribution of Mr. H. Weise, Dipl.Ing., Vice President of Rheinbraun - Consulting (U.S.) Inc., and Mr. E.J. Blazenko, P. Geol., Vice President of Operations for Techman Ltd. as senior advisors to the project. Both executives provided technical review and attended the meetings with the client and Steering Committee as senior representatives of the joint venture partners. Mr. Cliff Berry, Reclamation Consultant was instrumental in providing advice to the project staff based on his many years of practical experience in oil sands mine reclamation.

The project also acknowledges the consideration given by the field staff of Great Canadian Oil Sands Limited and Syncrude Canada Limited in answering questions by the consultants' staff on various field trips to the two operating oil sands mines, the permission granted by Shell Canada Resources Limited to examine their test pit site, and the contribution of geologic data by the Energy Resources Conservation Board.

The contents of this report represent the opinion of Techman Ltd. and Rheinbraun - Consulting GmbH with regard to the work required by the terms-of-reference contained in the contract. The Consultants attempted to meet the varied interests and requirements of both the Client and the Project Steering Committee in the most accomodative fashion possible. The subject area of this project was very extensive and consequently the Consultants were required to exercise considerable amounts of judgement in allocating the level of effort allowable in addressing any given problem.

It should also be noted that neither Techman Ltd. nor Rheinbraun-Consulting GmbH are the agent, in any manner, of equipment manufacturers and contractors nor do the Consultants represent any corporation or individual with interest in oil sand leases, investments, processes, patented technology, special interest groups and like.

CONCLUSIONS

The following conclusions are highlighted:

1. Reclamation of oil sands mines can be considered to be composed of two elements: the creation of reclaimable surfaces and the revegetation of reclaimable surfaces. The former component is by far the more expensive element of a reclamation plan; the revegetation component is relatively inexpensive by comparison. Often, in assessing costs of reclamation, efforts are concentrated on the cost of revegetation, whereas it is currently more important to devise operating methods for oil sand mining operations that will maximize the creation of reclaimable land surfaces. The current emphasis in reclamation should be shifted from revegetation to the optimization of mining and tailings disposal schemes that maximize the creation of dry reclaimable land, and that optimize prepared soil manufacturing techniques. Revegetation efforts must complement the field conditions which are created by materials handling techniques available to the oil sands mining operation.

2. The mine plans developed at each level of reclamation reflect the extent to which each mine can be reclaimed by integrating various tailings disposal concepts with various mining schemes. Thus, the mine plans at the Minimum Level of Reclamation represent the lowest achievement with respect to reclaimability, the Improved Level represents a greater reclaimability, and the Enhanced Level represents plans where the mine is totally reclaimable.

3. The overall quality of reclamation at any given level is largely determined by the prepared soil quality and depth. The quality and depth were selected for each level of reclamation to be progressively better and deeper, respectively. Consequently, the quality of reclamation parallels the implications of the qualitative adjectives "minimum", "improved" or "enhanced". Nonetheless, it is possible to have a very high quality surface treatment in a Minimum or Improved

Level plan. It was decided not to include this option in order to minimize the number of variations in mine plans that would have to be prepared in this study.

4. The reclaimability of an oil sands mining and extraction project is most affected by the choice of tailings disposal system and the extent of integration of the tailings disposal systems with the overall pit plan. In general, reclaimability levels are greatest with dry tailings and lowest with wet tailings. The concentration of tailings sludge achieved by storing sludge in a single sludge pond represents the single greatest improvement possible with a wet tailings disposal system. Tailings disposal systems where sludge is treated (dewatered) possess a level of reclaimability slightly greater than those in which the wet tailings disposal systems incorporate sludge rehandle only. The three basic categories of tailings disposal concepts can be integrated with pit plans to various degrees depending on the shape of the ore body, the size of the ore body, the general mine layout, and the development schedule of the mine.

5. The shape and size of the ore body dictates the mine layout and development schedule and, to a great extent, the feasible tailings disposal options. Longitudinally extended ore bodies (Ore Body No. 2 in this study) have advantages in that in-pit dykes can be more economically and conveniently constructed. Small, rectangular or triangular shaped ore bodies (Ore Body No. 4 in this study) result in a high volume of out-of-pit tailings storage requirements and in a high volume of in-pit dyke construction. Both these factors lead to high tailings disposal costs. Generally, tailings disposal costs are high for a small ore body operated independently of neighbouring ore bodies. Large uniform ore bodies also have high tailings disposal costs, but a greater portion of the cost is attributable to in-pit dyke construction. Large irregular ore bodies (Ore Body No. 1 in this study) provide ample opportunity for the efficient disposal of wet tailings. Using the area of wet pond surface remaining as a measure of unreclaimable surface area, the highest reclaimability is associated with the large but irregularly shaped ore body, and the lowest with the small but rather uniformly shaped ore body.

6. It is desirable to store as much tailings, especially sludge, in-pit as is technically possible and economically reasonable. Overburden and tailings sand can be placed out-of-pit with less risk and often with less overall cost. As well as reducing the risk of pollution by the failure of a containment structure containing wet tailings, this arrangement allows easier rehandle of wastes stored out-of-pit in the event that rehandle of waste becomes a future requirement. The removal of a tailings pond containing sludge and saturated sand appears to be the technically less desirable option. The option for rehandling of overburden, reject and tailings wastes has not been considered for any of the twelve mine plans developed in this report.

7. The overall efficiency of an out-of-pit tailings pond can be optimized by the proper selection of size, height, shape and location of the pond. However, the out-of-pit tailings pond can be properly designed only once the in-pit pond requirements have been determined and integrated into the overall detailed pit development scheme. In this manner, in-pit waste storage is maximized while the out-of-pit storage is minimized. The disposal of tailings at a considerable distance from the ore body outside of the suggested surface mineable limits was not considered for any of the twelve developed mine plans. Nonetheless, all the tailings ponds are sited on barren or currently very uneconomical oil sands as near to the pit and extraction plants as possible.

8. Wet tailings disposal concepts can take many forms, especially when the site-specific requirements of an ore body are considered. In mines where the extraction plant produces wet tailings, the major improvement with respect to reclaimability is achieved by concentrating sludge within one pond and sanding in all other ponds formed during the life of the operation. Sludge treatment, a dewatering procedure, will result in a small improvement in reclaimability but at a high cost of treatment and overburden dyke construction. For this study, the potential for sludge treatment was considered only for tailings produced by the Clark Hot Water Process.

9. The study indicates that the cost of reducing the sludge surface area to approximately 50% of the area required before dewatering will increase the per barrel cost of crude by nearly one dollar. Consequently, there is considerable incentive to investigate dry extraction processes rather than develop sludge dewatering processes. Combining sludge treatment with bitumen recovery from the sludge will improve the economics of partial sludge dewatering.

10. The detoxification, or rehabilitation of tailings water and sludge, is important with respect to improving reclaimability. The intent must be to make waters satisfactory for release to existing natural watercourses or for recycle. The thickening of sludge and possibly even the detoxification of pond surfaces will be required before a sludge pond surface can be reclaimed.

11. The likelihood of a tailings pond or a sludge pond filling with precipitation is unknown. If water does accumulate, dyke failure due to erosion is inevitable without perpetual maintenance. Reclamation objectives must be directed to eliminating all such ponds. However, current extraction technology appears to make the elimination of all sludge ponds impractical and consequently the solution towards such elimination must be approached on a more regional basis. Currently, the objective should be to minimize the areal extent of wet pond surfaces.

12. Mines utilizing dry tailings disposal will achieve total reclaimability at a cost equivalent to those using a wet tailings disposal. However, if the extraction plant capital and operating costs are higher than those of a hot water extraction plant generating the wet tailings, the overall costs per barrel produced will increase proportionally.

13. Certain characteristics were assumed for the dry tailings in order to produce mine plans and estimate costs for oil sands mining projects with extraction plants producing dry tailings. Amongst the assumptions was that the dry tailings would be transportable by conveyor and placeable by spreaders. Consequently, the backfilling

of the pit would resemble the backfilling procedures used in most large mines where conveyors and spreaders are employed. By generating dry tailings, the possibility exists to make the entire surface disturbance caused by the oil sands mining and extraction process reclaimable. The cost of the mining, tailings disposal and reclamation of such a mine are estimated to be equivalent to that of an operation where wet tailings are produced and sludge is concentrated by rehandling into a single sludge pond.

14. The addition of water to dry tailings to facilitate pumping is considered unnecessary and operationally undesirable. The slurrying of dry tailings would likely create wet tailings disposal problems similar to those associated with the Clark Hot Water Extraction Process, mainly the formation of unreclaimable sludge ponds. Additional costs of dyke construction and the slurrying itself make direct conveying and dumping with spreaders a more feasible method of dry tailings disposal.

15. It appears that dry tailings could be blended with overburden and reject to provide improved backfill slope stability and to allow for the selective placement of certain materials, for example a selected overburden layer on top of the sand or burial of materials detrimental to surface reclamation.

16. Mines with extraction plants producing dry tailings streams will minimize total surface disturbance since the amounts of waste stored in out-of-pit ponds and waste dumps will be greatly reduced. As well, the overall volume of dry tailings is less since little water would remain in the dry tailings.

17. The backfilling of mines with dry tailings can begin as early as 2 years after start of production. Mines with extraction plants producing wet tailings cannot begin backfilling with tailings until 10 or 12 years after start of production.

18. The examination of twelve mine plans indicates that certain trends are likely with respect to the application of draglines and bucket wheels as prime excavators. Most important, the particular type of excavator selected does not affect the potential for reclamation. However, principles of economy of scale with respect to equipment do affect the overall mining and tailings disposal costs.

19. Rather than utilizing windrows and bucket wheel reclaimers in the dragline schemes in this study, large-sized hoppers are used to avoid double handling of oil sands. The cost of mining is expected to be different if windrows and bucket wheel reclaimers are used to rehandle oil sands.

20. The application of the draglines appears more attractive at the smaller mine size, i.e., approximately 80,000 BPCD or less. Costs for dragline systems increase rapidly for larger mines, primarily due to the disadvantageous capital and operating costs of the entire materials handling system. At the 60,000 BPCD size, the dragline system is slightly more attractive than the bucket wheel system. At this mine size one bucket wheel excavator and two draglines are capable of handling the entire excavating requirements. In the bucket wheel plan at the 60,000 BPCD size, three bucket wheel excavators are needed. The conveyor and spreader systems associated with both types of prime exavators are very similar in the 60,000 BPCD mine size. At the 120,000 BPCD mine size the application of one bucket wheel excavator and four draglines as well as roughly fifty percent more conveyors in the dragline plan, compared to three bucket wheels in the bucket wheel plan, results in an overall higher capital and operating cost for the dragline system. At the 240,000 BPCD mine size, the costs are even more unfavourable toward a dragline system due to the more extensive conveyor systems required. The optimal bucket wheel mine size appears to be in the neighbourhood of 120,000 to 150,000 BPCD.

21. There are distinct operational limits within which a prime excavator should be applied. The height of the operating bench is the major factor. Attempting to remove too thin a bench is as harmful to productivity costs as trying to remove too thick a bench. In the ore bodies studied in this report, three benches are required in both the bucket wheel and dragline mines. In all the plans, the overburden is removed by bucket wheels. Single oil sands benches are impractical for the ore bodies examined in this report.

22. Two distinct conveyor layouts are possible in an oil sands mine: parallel and slewing. Generally, parallel systems require more conveyors than do slewing systems. Consequently, slewing systems are more economical to operate. The advantage of the parallel system, in some ore bodies, is that the mined-out pit can be utilized earlier for backfilling with tailings. Slewing systems are easily applied to most ore bodies but there are ore bodies where only parallel systems could or should be used.

23. In a multiple bench mining operation, the conveying of overburden, reject and oil sands is the most expensive single element of the mining operation, exceeding even the cost of excavation. Consequently, planning must give due consideration to this aspect of the development.

24. The separation of centre reject from within the ore may be a requirement in order to maintain a specified plant feed grade oil sand. The calculation of mining quantities for all twelve mine plans in this study is based on the presence of only one large centre reject band. The finer separation of centre reject possible with bucket wheels results in less dilution of the pay zone horizon but an overall lower total bitumen yield. The greater mining loss and dilution expected with the draglines results in larger plant feed volumes at a lower average grade but an overall higher bitumen yield. Dilution material is assumed to contain some bitumen. However, the production of tailings will be higher for the dragline system due to both increase in plant feed quantity and higher fines content of the plant feed resulting in greater sludge quantities generated. The decision to select either type of excavator cannot be made on selectivity. Capital and operating costs are the major concerns in prime excavator selection.

25. Working of bucket wheels near the pit floor is not a drawback to the application of the bucket wheel excavator. The excavator can be operated to remove pay zone below the crawler grade by deep cutting or deep cutting with ramping. Pockets of oil sand below the reach

of the machine can be removed by other equipment such as a small dragline. In a multiple bench dragline system, the bench height for the lower dragline may also require adjustment to allow the dragline to remove similar deep pockets.

26. The spreading of prepared soil on graded reclamation areas requires mobile equipment regardless of the overall mining method. It is expected that medium-sized off-highway mining trucks will transport the prepared soil from the field depots to the reclamation site. This spreading activity will occur both in winter and summer, stopping only during spring thaw or rain.

27. Three distinct prepared soil manufacturing methods were analyzed and costed. The prepared soil manufacturing methods were matched to the level of reclamation to reflect overall improvements with respect to revegetation potential. At the Minimum Level of Reclamation, the prepared soil manufacture consists of separately trucking muskeg and overburden onto the reclamation site. After thawing and sufficient drying, the muskeg and overburden are spread by dozer. Tailings sand is mixed with overburden and muskeg to form a total prepared soil thickness of 0.6 m. Rototilling is followed by planting of the appropriate revegetation plant species. At the Improved Level, the prepared soil manufacture is enhanced by utilization of a layered blend pile. Alternate layers of selected overburden and muskeg are spread in thin layers. When required for reclamation, a dozer or scraper cutting across these layers further blends the material. The mixed muskeg and overburden is hauled to the reclamation site and spread to a thickness of approximately 1 m. The most uniform and highest quantity of prepared soil is expected at the Enhanced Level where muskeg is obtained from a hydraulic muskeg mine and suitable overburden diverted from the mine conveyor system via a stacker into blend piles. Prepared soil is removed from the blending yard with a small bucket wheel reclaimer and associated conveyor to field depots. The prepared soil is trucked onto the reclamation site as required for reclamation.

28. The direct costs for reclamation activities such as prepared soil manufacture, transport, spreading and basic revegetation vary from \$0.025 to \$0.056 per barrel of synthetic crude at the Minimum Level, from \$0.121 to \$0.185 at the Improved Level, and from \$0.119 to \$0.193 at the Enhanced Level. Economy of scale is realized with respect to prepared soil manufacture at the Enhanced Level as the mine size increases. The 240,000 BPCD mine has a cost of \$0.12 per barrel of synthetic crude for reclamation activities, while for the 120,000 BPCD mine, the reclamation cost is \$0.20 per barrel of crude.

29. The additional cost of revegetating to commercial forest adds from \$0.005 to \$0.165 per barrel for the mine plans developed in this study. Revegetating to non-commercial forest adds \$0.004 to \$0.012 per barrel.

30. The muskeg to overburden ratio assumed in the manufacture of prepared soil in the study is 1:2 by volume. This is done in order to arrive at conservative prepared soil manufacturing costs. It is expected that, in most situations, the quantity of muskeg should be less.

31. Embankment slopes of both tailings dykes and waste dumps must be shallow enough to permit spreading of prepared soil by dozer. Overall slopes of 4:1 and 3:1 between berms (roadways) are considered acceptable for the prepared soil spreading activities described in this report.

32. In the Muskeg River area, good quality fine-textured overburden is exceedingly scarce. Bearing this in mind, great care should be taken to select the most suitable overburden during the stripping operation and to combine the overburden with muskeg and tailings sand to arrive at the best possible prepared soil mixture. Adequate mapping of muskeg and overburden deposits within the proposed mine boundaries is required before a reclamation plan can be prepared. The reclamation plan must reflect the availability of suitable muskeg and overburden throughout the life of the mine. If suitable materials for prepared soil manufacture are not available in the

quantity desired for the selected land use at any given time in the life of the mine, then the reclamation must be postponed or the land use changed to allow reclamation to be carried out with lower quality and quantities of materials.

33. The quality of prepared soil or rooting medium is of utmost importance to minimize the risk of "crop" failure and the amount of maintenance required on an annual or longer basis. As experience is gained with revegetation using various overburden, muskeg and sand ratios over a variety of sub-surfaces and over a range of depths, it will be possible to balance maintenance cost against the cost of improving the prepared soil quality or increasing its depth. Such experience information should be well documented and widely disseminated by the industry, government and research agencies involved with oil sands reclamation.

34. The period of fertilization required before a self-sustaining vegetation can occur is not yet known. However, it is strongly suspected that the period will be significantly reduced as the prepared soil thickness is increased from very shallow to 1 m in depth.

35. Examination of field experience to date of the oil sands industry and researchers strongly indicates that approximately one metre of prepared soil is required for self-sustaining forests, especially commercial-quality forests.

36. The final land use plan must be based, firstly, on the availability of materials suitable for prepared soil manufacture and, secondly, on a desired regional land use plan. Reclamation guidelines based on the available materials, the sub-surface to be reclaimed, and potential land uses should be developed to allow formulation of decisive reclamation plans in oil sands mines. Prolonged negotiation is counter-productive with respect to generating workable and satisfactory reclamation plans.

37. End-pits will be particularly significant for the maximization of reclaimable land surfaces. The final end pit from an earlier mine should be considered for use by a later mine for tailings disposal, thus greatly increasing regional reclaimability. End-pits may also prove to be effective as plant make-up water reservoirs or for the controlled disposal of saline mine water.

38. The advantages of simultaneously operating extraction plants using the Clark Hot Water Process and a yet-to-be-developed dry process should be investigated with respect to sludge disposal. Sludge could be incorporated into the dry tailings stream or regional sludge containment structures could be built, in part, with dry tailings. The concentration of sludge, either for reprocessing in the future or for reducing the areas of unreclaimable surfaces, is required on a regional basis since no methods currently exist which would allow the reclamation of sludge ponds.

39. The twelve mine plans developed for this study were simulated by use of computer programs. The programs were utilized to manipulate geologic data, to simulate mine operations, to provide information for tailings disposal, and to summarize costs. It appears unlikely that an adequate number of development options can be examined in sufficient detail without simulation aids in order to optimize the mine plan for a given ore body. The preparation of regional oil sands development concepts will undoubtedly require computerized planning aids.

40. A regional mine development plan is required in order to maximize oil sand recovery and creation of reclaimable land surfaces, and to minimize detrimental interference between mines. The determination of realistic ore body boundaries and the determination of areal requirements of likely neighbouring mine developments should be completed prior to extensive detailed mine planning on any given ore body. A greater emphasis should be paid to areas where ore bodies are clustered.

41. The cost of reclamation in an oil sands mining operation is the summation of various operational costs that are likely to be affected by meeting a given reclamation objective. For this purpose five cost centres, of which field reclamation cost is only one, were defined in this study. Direct field reclamation costs accounted for 2% to 12% of the total operating costs estimated in this study, or for 0.5% to 3% of the total oil sands mine operating costs excluding taxes, royalties and interest.

42. The overall cost summary by mine and cost centre (see following table) indicates that the costs of the mines at the Minimum and Enhanced Levels are similar but that the cost at the Improved Level is considerably higher. Next to overburden, reject and oil sands handling, tailings disposal is the most expensive field operation. The establishment of ultimate land use resources, i.e. field reclamation activities, is the third highest field cost. Staff for muskeg removal, overburden stripping, oil sands mining, tailings disposal and reclamation are grouped into one cost centre. A table summarizing the cost centre follows. The reader is warned not to draw conclusions from this table without first reading Chapter 6.0 and Section 10.7 of the report.

43. Major gaps currently exist with respect to the capacity of the industry and the goverment to solve problems related to mining, extraction, tailings disposal and reclamation of oil sands. This report has outlined some major items of research that are urgently needed in order to assess possible operating alternatives for the industry. These are summarized in Chapter 12.0 of this volume.

COST CENTRE SUMMARY BY MINE AND COST CENTRE ($/bbl)

COST CENTRE*	1	2	3	4	5	6	TOTAL
ORE BODY NO. 2							
Dragline Scheme							
1. Minimum	0.0588	0.0475	1.3396	0.4465	0.0501	0.1870	2.1294
2. Improved	0.0588	0.0455	1.3434	1.2945	0.1273	0.1870	3.0565
3. Enhanced	0.0699	0.0455	1.3434	0.3072	0.1712	0.1904	2.1276
B.W.E. Scheme							
4. Minimum	0.0436	0.0464	0.9026	0.4468	0.0253	0.1959	1.6606
5. Improved	0.0436	0.0464	0.9026	1.1110	0.1214	0.1959	2.4209
6. Enhanced	0.0530	0.0405	0.9026	0.2210	0.1931	0.1995	1.6097
ORE BODY NO. 4							
Dragline Scheme							
7. Minimum	0.0681	0.0590	1.2645	0.4509	0.0412	0.3531	2.2369
8. Improved	0.0685	0.0582	1.2682	1.0428	0.1799	0.3531	2.9709
B.W.E. Scheme							
9. Minimum	0.0656	0.0596	1.2683	0.4444	0.0429	0.3568	2.2377
10. Improved	0.0657	0.0589	1.2807	1.0545	0.1846	0.3568	3.0012
ORE BODY NO. 1							
B.W.E. Scheme							
11. Minimum	0.0455	0.0517	1.1145	0.4034	0.0556	0.1499	1.8207
12. Enhanced	0.0580	0.0419	1.1145	0.3257	0.1194	0.1531	1.8126

* COST CENTRE 1: Civil Construction-Type Activities
COST CENTRE 2: Removal of Organic Materials & Soils
COST CENTRE 3: Overburden, Reject, Oil Sands Handling
COST CENTRE 4: Tailings Disposal
COST CENTRE 5: Establishment of Ultimate Land Use Resources
COST CENTRE 6: Supervision, Technical Services

NOTE: Only direct costs are shown. Consult Section 10.6 for further explanations of cost centres and Section 10.7 for detailed comparative analysis of costs.

GLOSSARY

Backcast: Waste material directly dumped onto the pit floor by the bottom bench dragline (see Figure 1.3.3-15)

Backfill: Waste materials conveyed into the mined-out pit and deposited by spreaders. (see Figures 1.3.3-13 and 14)

Beaching: Forming of tailings sand beach against a dyke by spigotting tailings slurry (see Figure 1.3.3-21)

Bitumen: Hydrocarbon substance occupying the space between sand grains of oil sand (see Figure 5.1.4-1), and removed from oil sand by the extraction process.

Blending Pile: A pile composed of alternate layers of muskeg and overburden. Dozers or bucket wheel reclaimers mix the components into prepared soil.

BPCD: Barrels per calendar day - rate at which synthetic crude is produced by the process plant

Bucket Wheel Excavator (BWE): Prime excavator digging by means of buckets attached to rotating wheel, and capable of excavating a continuous stream of material (see Figures 1.3.3-5, 1.3.3-6, and 5-5.1)

Bucket Wheel Reclaimer (BWR): Similar to bucket wheel excavator but used for rehandled material only. When designed for the sme output, a BWR is lighter in structure than a BWE. (see Figure 1.3.3-7)

Capital Cost: Purchase cost of the initial units of equipment, buildings, parts, supplies, etc. Also includes replacement units of equipment such as trucks, dozers, scrapers, etc.

Cation Exchange Capacity: The ability of soil particles to exchange the positive ion nutrient elements necessary for plant growth.

Commercial Forest: Forest which has grown to a level of maturity sufficient to support a forestry harvesting operation, as opposed to non-commerical forest which, because of age, condition or site limitations, has not achieved a growth level sufficient to support a harvesting operation, and unproductive forest and brushland which, because of severe site limitations and tree type will not normally become commercial forest, although other uses are not precluded (i.e. wildlife habitat).

Commercial Forest Plantings: Plating of the species of tree that may eventually be profitably harvested.

Comercial Quality Timber: Mature forest consisting of the species and grade necessary for profitable harvesting.

Conveyor: Steel cable reinforced rubber belt supported on rollers capable of the continuous movement of large volumes of mined materials as well as solid waste. (see Figure 1.3.3-11)

Cost Centre: Several related activities grouped together for cost comparison purposes (eg. Tailings disposal)

Cost Sub-centre: A distinct operational activity separated from others for purpose of costing.

Critical Habitat: A segment of habitat which has singular importance in the survival of one or more species.

Distribution Point: Arrangement of conveyor termini allowing oil sand to be deposited on conveyors going to the plant, and reject and overburden onto conveyors going to the waste dump. (see Figure 1.3.3-12)

Dragline (D/L): Prime excavator digging by means of a large bucket suspended from boom. (see Figure 1.3.3-8 and 9)

Dragline Hopper: Large hopper enabling dragline to load onto belt conveyor, and buffering the cyclic nature of dragline production into a continuous stream of material necessary for conveyor operation. (see Figure 1.3.3-11 and 5.5-2)

Drive Station: A structure usually located at the front end of a belt conveyor segment, and whcih houses electric motors and the belt tensioning system. The drive station propels the conveyor belt and provides the required lift to dump materials onto tail end of the following conveyor. (see Figure 1.3.3-11 and 5.5-5)

Dry Tailings: Dry sand and fines produced in the bitumen extraction plant by a dry process. Dry bitumen extraction and dry tailings are assumed at the Enhanced Level of Reclamation. (see Figue 1.3.3-16)

Dyke Cell Construction: Tailings sand hydraulically-placed and compacted by dozers within a cell of 1-2 m high dozed-up dykes. (see Figure 1.3.3-17 to 20)

Emissions/Effluents: By-products of industrial activities which are released into the environment (air, water, land), often with negative biological effects.

End Land Use Objectives/Capability: The determination for lands, following project completion, of an appropriate use for which the land and its surroundings have both a need and a capability.

End-pit Lake: Residual mined-out pit which cannot be backfilled without extensive rehandle of previously placed materials and which is filled with water as part of the reclamation scheme.

Erosion Control: Stabilization of slope surface (dykes, pit walls, waste dumps) by revegetation or other means.

Extraction Plant: The process facilities associated with the extraction of bitumen from the mined oil sand.

Feasibility - technical: Possible within the scope of current technology.

Feasibility - economic: As above, but also currently profitable.

Impact Mitigation: An interaction between the natural environment and an activity of man, and a concurrent action or precaution taken to minimize or eliminate the effects of the action.

In-pit Tailings Pond: Tailings pond located in the mined-out pit. See also definition of tailings pond.

Leaching: Water infiltrating and passing through soil, washing out nutrients.

Level of Reclamation: A measure of reclaimability defined in terms of tailings disposal and surface reclamation characteristics. The term is related to the sophistication of materials handling techniques employed in the mine.

- Minimum: Used in the study associated with wet tailings. Tailings sludge may or may not be rehandled into one pond, and prepared soil is mixed to a depth of 0.6 m at reclamation site and revegetated for grass, shrubs and/or non-commercial tree species.

- Improved: Used in the study associated with wet tailings but with sludge treated (dewatered) and concentrated in one pond. Prepared soil is mixed at strategically placed mixing piles, spread 1.0 m thick onto reclamation sites and revegetated. Expected to be suitable for reforestation.

- Enhanced: Used in the study associated with an extraction process producing dry tailings. All waste materials are solids and are dumped with spreaders and conveyors. The best quality prepared is soil centrally mixed and then spread to the depth of 1.0 m and revegetated. Expected to be suitable for establishment of commercial forest.

Mining Bench: A step-like division of total mining depth. Its height is usually governed by the prime excavator's reach, mine scheduling, plant feed requirement, and geotechnical problems (slope stability), etc. (see Figure 1.3.3-15)

Mining Face: Sloping surface from which oil sands, overburden or reject are removed by prime excavators.

Mining Scheme: Refers to choice of prime excavators used. In some cases in this study it is also associated with different tailings disposal schemes used (i.e. Ore Body 2)

- bucket wheel schemes: Only bucket wheel excavators are used as prime excavators. Three benches of approximately equal height are employed regardless of the position of the overburden - oil sand interface. (see Figure 1.3.3-15)

- dragline schemes: Bucket wheel excavator removes overburden only. The oil sand zone is mined out in two benches of approximately equal height by draglines. (see Figures 1.3.3-15)

Moisture Holding Ability: The ability of soil or granular substances to absorb and hold water and make it available to plants.

Muskeg Dewatering: Removal of water from muskeg by means of array of ditches. (see Figures 1.3.3-1 and 2)

Muskeg Dewatering Plant: A facility used to remove excess water from muskeg slurry supplied by the muskeg mine for prepared soil manufacture. Applicable to the Enhanced Level plans only.

Muskeg Dump: Temporary or permanent pile of muskeg removal from mine or plantsite. Approximately 4 m high. (see Figure 1.3.3-4)

Muskeg Mine: Source of muskeg for prepared soil at the Enhanced Levels. Muskeg is hydraulically mined and piped as slurry to the dewatering plant. (see Figure 5.4.3-7)

Muskeg/Peat: Organic material derived from bogs, swamps or other inundated areas of typically poor drainage. It is of fibrous organic texture.

Muskeg Rehandle: Trucking of muskeg from muskeg dumps to reclamation sites (Minimum Level) or to the prepared soil blending pile (Improved Level).

Oil Sand: Four-phase hydrocarbon solid consisting of a solid phase (predominantly sand), liquid phase (water), gaseous phase (predomininatly carbon dioxide, nitrogen, and methanol), and viscous hydrocarbon phase (bitumen) (see Figure 5.1.4-1)

Operating Cost: Cost required to maintain and operate equipment, including operators, fuel and oil, repair and overhaul labour and parts, tires, tracks, conveyor belt and idler replacement, etc.

Ore Body: Oil sands deposit with GRAMT and R-Factor above certain cut-off value, which may be slightly modified to suit mining or tailings disposal. Three ore bodies were used for detailed planning in this study.

Ore Body 1: Used in 240,000 BPCD mine. It is composed of two parts mined concurrently.

Ore Body 2: Used in 120,000 BPCD mine. Delineation of the ore body for the dragline schemes is slightly different than that used for the bucket wheel schemes.

Ore Body 4: Used in 60,000 BPCD mine. Delineation of the ore body is the same for both dragline and bucket wheel schemes.

Overboarding: Similar to beaching. Term used for spigotting tailings slurry on top of beached sands in the sanding-in operation. (see Figure 1.3.3-21)

Overburden: Any material (including top reject) that overlies the uppermost pay zone.

Overburden Dyke: A compacted earth structure built by trucks, scrapers, dozers and compactors. Sometimes it is possible to supply suitable overburden to the dyke location by conveyor and spreader. An overburden dyke is used only when hydraulic tailings sand dyke building is not possible (primarily for dewatered sludge pond dykes).

Overburden Face: Mining face from which overburden is being excavated. In all mining schemes a B.W.E. is involved.

Overburden Rehandle: Trucking of overburden from waste dumps to the reclamation site (Minimum Level) or to the prepared soil blending pile (Improved Level).

Oversize Reject: Wet mixture of boulders, pebbles, pieces of shale and clay rejected from the wet process extraction plant (between tumblers and primary separation tanks), and trucked to the disposal site.

Out-of-pit: Designation used for tailings ponds or waste dumps constructed on ground surfaces underlain by uneconomic oil sands, away from the mined-out pit.

Outside: Sometimes used to describe out-of-pit ponds and dumps. The former term (out-of-pit) is preferred.

Parallel Mining: Mine design in which the mining faces remain parallel to each other. (eg. mine design of Ore Body 2 - Dragline scheme)

Pay Zone: Oil sands of over 5% bitumen and 1.52 m thickness, including centre reject (less than 5% bitumen) less than 1.52 m thick.

Pit: Oil sands mine worked by surface excavation.

Pit Wall: Side slopes of pit remaining after overburden, reject and oil sands removal.

Plant Feed: Oil sands supplied to the extraction plant by the mining operation including mine dilution from reject/pay zone interfaces and centre reject too thin to be separated by excavators.

Plant Site: Area designated for extraction plant, upgrading plant, plant feed surge pile system, and synthetic crude storage, etc.

Prepared Soil: Mixture of muskeg, overburden and, where applicable, tailings sand, used as growth medium for reclamation purposes.

Present Value of Costs: The sum of yearly operating and capital costs discounted at a specific interest rate each year between the date of occurrence and the beginning of the project.

Reclamation: The act of bringing disturbed land back into biological production or some other acceptable use (i.e. revegetation, creation of lake, etc.)

Reclamation Maintenance: Activities necessary to ensure that reclaimed sites do not regress, erode or lose production.

Recycle Water: Water which separates from tailings slurry and forms a sufficiently deep clarified layer on the tailings pond surface that it can be pumped back for reuse in the extraction process. (see Figure 1.3.3-16)

Reject: Low grade oil sands or barren material that is not defined as ore.

- top reject: low grade oil sand (<5%) located above the uppermost pay zone, and removed as overburden.

- centre reject: layer of low grade oil sands (<5%) or other waste material within the pay zone. This layer is removed and conveyed to waste dumps or backcast onto the pit floor.

- bottom reject: low grade oil sands (<5%) located below the lowermost pay zone, and left unmined.

Rooting Depth/Soil Depth: The depth to which roots penetrate, either because of natural habit or soil obstruction, versus the total depth of material capable of supporting vegetation, whether or not roots are present.

Saline Water: Waters especially high in dissolved sodium, which is toxic to many animals and plants. Saline water is sometimes present in deep aquifers and must be safely removed and disposed of to allow mining and safeguard the environment.

Sanded-in: Term used in association with tailings ponds from which tailings sludge has been removed and tailings sand allowed to completely fill the enclosure.

Seepage: The slow percolation of water through permeable or semi- permeable materials; often associated with tailings water moving through impounding dykes, or groundwater surfacing in a discharge zone.

Shunting Head: Extendable termini of conveyor drive which allows selective dumping onto several other conveyors. A shunting head is an essential part of a distribution point (see Figure 5.5-7)

Slewing Mining: Mine design in which the mining faces rotate around a point (eg. mine designs of Ore Body 4)

Sludge: Fluid mixture of water, clay and silt particles, bitumen, caustic soda and other chemicals, and overall, the most important factor to consider in the reclamation of mines where wet tailings are present.

Sludge Pond: The pond designed to contain all sludge. The sludge pond is preferably the deepest mined-out pit, to reduce impact on the environment. (see Figure 1.3.3-16)

Sludge Teatment: Operation which removes approximately 50% of the water and most of the bitumen from sludge.

Soil Quality: The characterization of a soil, based on chemical and physical parameters, which defines the end use to which that soil may be put.

Spreader: Crawler-mounted structure with a long boom and belt for the dumping of materials into stable piles or backfill (see Figures 1.3.3-13, 1.3.3-14, and 5.5-3).

Stacker: Similar to spreader but usually smaller or stationary, or mounted on rails (see Figure 5.4.3-8).

Starter Dyke: Compacted overburden structure that must contain sludge and water remaining from hydraulic construction of the sand tailings pond dyke in the initial year or two of operation (see Figure 1.3.3-16 and 23)

Surficial Materials: Those materials found on the surface of the land, and which are placed by recent geologic phenomena such as glaciers, water or wind. Surficial materials are usually a relatively unconsolidated overburden type.

Synthetic Crude: Crude oil produced from extracted bitumen by the upgrading plant.

Tailings Slurry: Waste from primary separation of bitumen and froth treatment. Slurry separates into sand, sludge and recycle water in tailings pond. (see Figure 1.3.3-18)

Tailings Pond or Out-of-pit Tailings Pond: A structure for impoundment of tailings produced in Clark's Hot Water extraction process. Usually refers to pond located on ground surfaces underlain by uneconomic oil sands (see also definition of an in-pit tailings pond. (see Figure 1.3.3-24)

Treated (dewatered) Sludge Pond: Pond (usually in-pit) used at the Improved Level of Reclamation which contains partially dewatered sludge, and which requires overburden dykes because run-off from hydraulic dyke construction would otherwise dilute treated sludge.

Tripper Car: Crawler-mounted structure closely, coupled with spreader, which creates a loop in belt conveyor necessary for material removal from belt conveyor anywhere along its length (see Figures 1.3.3-13, 1.3.3-14, and 5.5-4).

Unit Cost: The specific cost, operating or capital (eg. $\$/m^3$, $\$/(m^3xkm)$; etc.) used in calculation of costs for known yearly quantities, time, distance, etc.

Upgrading Plant: The process facilities which convert extracted bitumen into synthetic crude.

Waste Dump: A pile of waste materials, such as overburden and centre reject, deposited by spreaders and conveyors. The waste dumps are constructed on ground surfaces underlain by uneconomic oil sands.

1.0 OVERVIEW

1.1 PURPOSE AND SCOPE OF THE STUDY

The impact of actions taken to reduce undesirable environmental effects can be measured both biophysically and economically. However, the determination of "added benefit" and "associated cost" is more complex, with many subjective elements. This study attempts to define not only the methods required to achieve various degrees of reclamation in the Athabasca oil sands, but also to show how the benefits and costs of such reclamation can be objectively measured.

Objectives of the study are listed below:

- To illustrate the application of recommendations respecting development and reclamation by designing "model operations" for three oil sands mines delineated by actual field drilling data, with process capacities of 60,000 BPCD*, 120,000 BPCD, and 240,000 BPCD, respectively, after extraction and upgrading losses.

- To examine mining and reclamation schemes in detail within the context of "minimum", "improved", and "enhanced" levels of reclamation; each representing operations using "wet", "dewatered", and "dry" tailings systems, respectively, in order of decreasing reclamation difficulty.

- To determine the additional materials handling costs incurred in oil sands mines when selected reclamation alternatives are implemented, and to perform a cost-benefit analysis using rational economic units such as dollars, hectares, cubic metres, etc. per barrel of synthetic crude oil produced.

- To define the technical limitations of materials handling and overall mine planning with respect to creating or carrying out reclamation options.

* BPCD = barrels per calendar day.

- To suggest typical materials handling activities and their comparative merits in oil sands mine reclamation.

- To determine whether the choice of major mining equipment substantially affects the success of reclamation efforts.

- To recommend the specific techniques that must be incorporated into an operator's mining methods to ensure successful reclamation.

- To compare the advantages and disadvantages of oil sands mines using the traditional "wet" tailings pond with those using conceptual "dewatered" or "dry" tailings disposal systems.

- To develop an understanding of the major factors that dictate the reclamation potential of oil sands mines, and to describe the impact of these factors on oil sands development regionally.

- To estimate the direct energy consumption of an oil sands mine that incorporates recommended reclamation objectives.

- To apply the combined experiences of Techman Ltd. - Rheinbraun-Consulting GmbH, oil sands mine operators, and other concerned investigators, to the problem of reclaiming oil sands mines in Alberta.

- To prepare guidelines defining the information required in the preparation of an oil sands mine development and reclamation plan.

This report presents the results of the study in the following manner:

In Chapter 1.0, Overview, a summary of study objectives is followed by a brief discussion of reporting methods and special project considerations, and then by reviews of the oil sands mining operations of both Great Canadian Oil Sands Ltd. and Syncrude Canada Ltd.

Chapter 2.0, Definition of Study Area, reviews basic drilling information, and discusses the criteria used in the selection and sizing of the oil sands ore bodies eventually considered in the study.

Chapter 3.0, Review of Information Concerning Selected Mining Area, highlights prevailing biophysical conditions at the sites selected, and includes, as well, a review of experience to date on oil sands mine reclamation.

Chapter 4.0, Definition of Levels of Reclamation, identifies the features that define the three levels of reclamation (Minimum, Improved and Enhanced) used throughout the study.

Chapter 5.0, Mine Development Criteria, reviews basic geologic and engineering criteria, and explains in detail the major design considerations for tailings disposal, reclamation techniques, and equipment that must be understood prior to detailed mine planning.

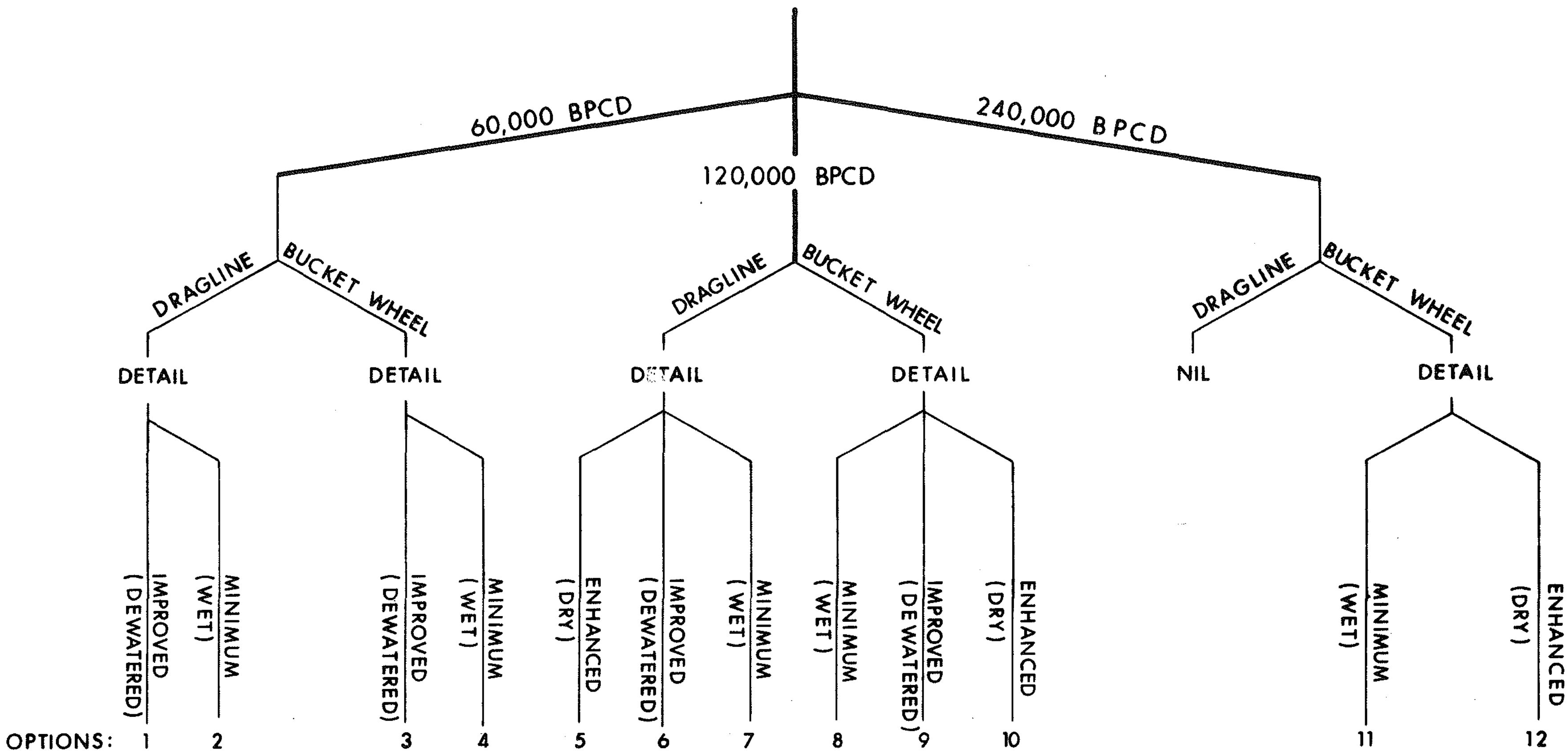
Chapter 6.0, Costing Methods, explains the basis for all costing performed in the study, and specifies all component activities considered in estimating the "net cost of reclamation".

Chapters 7.0, 8.0 and 9.0, Concepts and Costs of Development and Reclamation (of 120,000 BPCD, 60,000 BPCD, and 240,000 BPCD Oil Sands Mines, respectively), present an overall development concept for each scheme, as well as detailed mine plans, technical explanations, and cost details for each of the twelve options delineated for inclusion in the study (Figure 1.1-1).

Chapter 10.0, Major Factors in the Development and Reclamation of Oil Sands Mines, draws both on the experiences of current operations as well as on oil sands mine modelling results, and summarizes the major factors influencing oil sands mine development and reclamation.

Chapter 11.0, Recommendations Towards Reclamation Guidelines, briefly sets forth recommendations for reclamation guideline development.

Chapter 12.0, Recommendations Towards Further Applied Investigation of Oil Sands Mine Development and Reclamation, recommends the study of several areas of concern, which the Consultants feel would generate information allowing more efficient development of the Athabasca oil sands.



FINAL DELINEATION OF MODELLING OPTIONS

FIGURE 1.1-1

1.2 EXECUTION OF THE STUDY

An in-depth literature search commenced early in September, 1977, and continued through June, 1978. The objective of the search was to obtain background data for use in the creation of mine development and reclamation plans for the study areas. Simultaneously, a geologic interpretation of potential mine sites was conducted, as described in the first portion of Chapter 2.0. The ERCB assisted in this regard by supplying data for interpretation by Techman Ltd. In March, 1978, the Project Steering Committee approved three mine sites for further detailed study. From March through June, 1978, detailed assessments of geologic and biophysical data were made. The results of these assessments are detailed in Chapters 2.0, 3.0 and the first half of Chapter 5.0. The development of a set of computer programs designed to assist in the data planning efforts also occurred during this period.

Detailed mine planning was started in July, 1978, and continued until the end of March, 1979. Mine plans utilizing draglines, bucket wheels or combinations of wheel and dragline were developed. During this period the concepts of "minimum", "improved" and "enhanced" levels of reclamation were further evolved.

The final determination of the "net cost of reclamation" required that the costs of major component operations (resulting from the application of both the dragline and bucket wheel mining techniques at the three levels of reclamation) be detailed and then compared. The initial list of component activities was progressively modified and finalized to include 44 cost items. Costing began in July, 1978, and was completed in April, 1979.

The complexity of the work required that the activities of the staff of Techman Ltd. and RHEINBRAUN-Consulting GmbH (hereinafter identified as the "Consultants" or "Techman/RC") be integrated at all stages of the study. The contribution of technology during the course of the study was as follows:

Major activities of Techman Ltd.

- literature search
- interpretation of geologic data
- development of mine simulation models
- tailings disposal techniques
- dragline mine planning
- mobile equipment applications
- reclamation techniques
- costing

Major activities of RHEINBRAUN-Consulting GmbH

- bucket wheel mine planning, and equipment layout
- applications of bucketwheels, spreaders, reclaimers, conveyors
- mobile equipment applications
- reclamation techniques
- costing

The sheer volume of data needed for reliable mine planning, and the fact that using average rates of production, yields, etc. leads to erroneous scheduling of mining and tailings disposal activities, necessitates the use of mine simulation programs. Once a mining sequence has been conceptualized, the scheme can be tested to prove its workability and the results used to calculate the costs of operating the mine.

The production of overburden, oil sands, reject, tailings sands, and tailings sludge are summarized by computer, and the feasibility and schedule of in-pit tailings disposal and backfilling are determined. The viability of any given mining and reclamation concept is dependent on the ability of the designer to accurately schedule mass movements occurring during the life of the mine.

The meanings of the reclamation terms "minimum", "improved", and "enhanced", as defined in this report, reflect general types of materials handling concepts analogous to three distinct types of tailings disposal: wet, dewatered, and dry.

The term "minimum" is associated with a mining operation generating a wet tailings product, such as is produced by Great Canadian Oil Sands Ltd. and Syncrude Canada Ltd. The associated reclamation effort specified for the Minimum Level of Reclamation in this study results in the least enduring revegetation and the most inflexible land use.

The term "improved" is used in conjunction with an operational practice in which tailings sludge is processed to remove bitumen, and the sludge is partially dewatered prior to disposal. By reducing the total tailings quantity and thickening the sludge, a much greater potential for ultimate reclamation of the sludge disposal site is created.

The "enhanced" level of reclamation is associated with an operation whose tailings products are dry, i.e. transportable by belt conveyor. Since this is the most futuristic operating option examined in this study, reclamation techniques offering the best chance of long-term revegetation success are used in conjunction with it.

The terms as defined may appear to indicate a certain quality of reclamation. This is not the intent, since any number of attributes such as the cost of reclamation, the economics of post-mining land use, the durability of vegetation, the aesthetics of the final landform, the thickness of replaced soils, the final land use possibilities, the extent of surface disturbance, and many other factors might be stressed. In this study, it is clearly shown that the method of tailings disposal controls the overall mining and reclamation plan, and that the quality of reclamation done on dry land areas remains independent of most other activities involved in the mining of oil sands.

The practical application of the three types of tailings disposal techniques is discussed in Section 5.3. The reclamation techniques, explained in Section 5.4, were evolved to be compatible with the mining and tailings disposal concepts demonstrated in the twelve detailed mine plans generated for this study.

The detailed mining and reclamation plans generated in this study utilize various mining methods. Some of the plans utilize techniques currently in use at the two operating oil sands mines. Since field conditions prevailing at the study sites differ from those at current operations, some plans are based on operating techniques that have not been employed to date. In other instances, alternative operating techniques are required by the reclamation objectives set in the study. In the following sub-section, an overview is provided of the two operating oil sands mines in Alberta, followed by a short review of the major operating techniques employed at these mines and in the mine plans presented in this report.

1.3 TECHNICAL OVERVIEW OF OIL SANDS MINING

1.3.1 TECHNICAL OVERVIEW OF THE GREAT CANADIAN OIL SANDS LTD. OPERATION[1,2,3,4,5]

Introduction

The oil sands mining operations of Great Canadian Oil Sands Ltd. (G.C.O.S.) are located on Bituminous Sands Lease No. 86, which covers an area of about 1,830 hectares and is situated on the west bank of the Athabasca River about 34 km north of Fort McMurray, Alberta. An additional 618 hectares of land has been sub-leased by agreement with Syncrude Canada Ltd. (Lease 17A).

The G.C.O.S. operation was the first major scheme to produce a synthetic crude oil from the Athabasca oil sands on a commercial scale. The level of production initially approved by the Oil and Gas Conservation Board was 31,500 BPCD; this was subsequently raised to 45,000 BPCD. Following government approval of the project in 1964, construction was begun, and the operation was commissioned in September, 1967. After several years of operation, improvements in equipment reliability and operating efficiencies allowed G.C.O.S. to make application to increase production to 65,000 BPCD, and the Board gave its approval in December, 1973.

Economic Geology

The overburden is soil and muskeg at the surface, underlain by glacial drift consisting of boulder sands, sands and boulder clays, followed by grey to green glauconitic sands and shales of the Clearwater Formation, and top reject (oil sands with less than 8 % bitumen) resting on the ore zone. Within the present pit limits, which enclose about 90 % of Lease No. 86, the overburden averages 16 m but varies in thickness from 0 to 46 m.

The ore body consists of a basal fluvial sand overlain by lagoonal deposits which have been channelled into and refilled with silty beds. The amount of bitumen in the sand is not homogeneous, but varies from zero to 18 % by weight. In addition, there are numerous thin beds of clay which contain no bitumen. There is a noticeable increase in the coarseness of the sediments and bitumen content towards the base of the deposit. The thickness of economic oil sands averages 40 m and ranges from 0 to 72 m. The bitumen content of the economic oil sands zone averages 11.5 to 12 % (weight percent on a dry basis). The fines content (weight percent of fines in the mineral fraction) of the oil sands varies from about 5 to 45 % and averages about 16 %.

The ore zone rests on Devonian-age formations of limestone, shale, salt, dolomite and anhydrite.

Muskeg Removal

Muskeg overlying the ore body must be drained before removal is possible. This is achieved by digging an extensive network of ditches and allowing the muskeg to drain naturally for about 2 years. The muskeg is excavated and transported during the winter months with a fleet of Marathon - Letourneau 11 m^3 front-end loaders and Wabco 150 B trucks. The muskeg is stored in specially-constructed impoundment structures. Only a small part of the muskeg is to be used for the land reclamation program. The height of muskeg waste dumps is limited to 30 m.

Overburden Stripping

Approximately 80 % of the overburden is used to build dykes inside the pit to contain future extraction plant tailings. At present, approximately 9.94 million m^3 of overburden are removed per year. The fleet of 11 m^3 front-end loaders and 136-tonne trucks is used for overburden excavation and hauling for about eight months of the year. This operation is supplemented by two bucket wheel excavators (BWE) loading into trucks. A Bucyrus-Erie BWE with a rated output of 3,630 tonnes/hr. has been used since 1970 to load part of the truck fleet during the summer months. Recently, a new Orenstein and Koppel BWE with a rated output of 4,900 tonnes/hr. has been used during summer and winter operations to load overburden into the truck fleet.

Mining Operation

The G.C.O.S. mining operation utilizes bucket wheel excavators for mining, and belt conveyors for transporting oil sands plant feed to the extraction plant. The main mining units are two Orenstein and Koppel LMG bucket wheel excavators with a rated output of 3,990 tonnes/hr. each operating on parallel benches; one bench leading the other. The average thickness of the ore body is about 40 m. The lead bench is usually 23 m high and the trailing bench varies in height. At an ore grade of 12 %, approximately 104,300 tonnes of oil sands must be mined in one day to produce 45,000 BPCD of synthetic crude oil. The mineable oil sands are preblasted to facilitate BWE excavation, and slopes are cut to an angle of 40° to 45°. The G.C.O.S. pit is essentially dry with no water problems caused by high pressure aquifers at the base of the McMurray Formation, as encountered in other areas of the Athabasca oil sands.

The oil sands excavated by the BWE are deposited on a face conveyor via the conveyor system installed on the BWE and a connecting belt wagon. These 1,524 mm wide conveyors run parallel to the mining face and are about 1,525 m long. Each BWE makes two passes of 1,220 m to 1,525 m long and 43 to 46 m wide, taking two to three weeks for each pass. The face conveyors must be moved every three months to keep up with the mine advance. The two face conveyors feed onto two trunk conveyor systems, which carry the mined oil sands to feed bins at the extraction plant. One trunk conveyor extension must be made for every two face-conveyor moves.

Bitumen Extraction and Upgrading

Crude bitumen is extracted from the oil sands plant feed using the Clark Hot Water Extraction process and dilution centrifuging. The bitumen produced is not suitable for market as a refinery feed stock, and must be upgraded before it can be shipped in a conventional pipeline. The raw bitumen is subjected to a delayed coking operation following diluent recovery. Vapors from the coking drums are fractionated into gas, naphtha, kerosene and gas-oil components. After further processing, the latter three products are blended to form a synthetic crude oil, which is trans-shipped to Edmonton through a 428-km pipeline for eventual shipment to eastern Canadian and U.S. refineries.

Tailings Disposal

Initially, extraction plant tailings were disposed in a tailings pond constructed immediately adjacent the Athabasca River. A starter dyke was constructed along an island in the river (Tar Island) and across a small channel. The dyke was subsequently raised with the coarse fraction of the tailings stream to form a structure that would contain sludge and fines. This pond was designed to store tailings until sufficient space became available in the mined-out area to allow for in-pit disposal; at present, its dyke reaches a height of over 82 m.

Recently, G.C.O.S. has commenced disposal of tailings in the pit. A large dyke composed of overburden material was constructed across the pit to form the first in-pit pond. Continued in-pit disposal will be accomplished by construction of a series of overburden dykes as mining progresses to the northwest. Current G.C.O.S. plans propose the development of various tailings ponds, some topped with sand and others filled with sludge, and a final pit which will not be backfilled.

Approximately 1,500 litres per second (L/S) of tailings in the form of water (50 %), bitumen, sand, silt, and clay are pumped by multistage centrifugal pumps to the disposal area. Clarified water is recycled to the extraction plant for use as make-up water in the process. Unsettled clays are present in this recycle water so the quality must be carefully regulated.

1.3.2 TECHNICAL OVERVIEW OF THE SYNCRUDE CANADA LTD. OPERATION[1,5,6]

Introduction

Syncrude Canada Ltd., through its participants, controls mining rights on Bituminous Sands Leases 17 and 22, in the Mildred Lake area north of Fort McMurray, Alberta. These leases cover an area of about 39,593 hectares and the lease ownership is as follows:

Imperial Oil Limited	31.25 %
Canada-Cities Service Ltd.	22.00 %
Gulf Oil Canada Limited	16.75 %
Government of Canada	15.00 %
Government of Alberta	10.00 %
PanCanadian Petroleum Ltd.	5.00 %

The Syncrude project is the second major commercial oil sands mining scheme in the Fort MacMurray area. The Syncrude project began in the late 1950's. Syncrude Canada Ltd. first applied to the Alberta Oil and Gas Conservation Board in 1962 for permission to build a synthetic crude oil facility capable of producing 100,000 BPCD. This was refused because of Alberta government policy, which was designed to assure conventional oil supplies a sufficiently large market. The second application in 1968 was again delayed due to possible uncertainties related to the Prudhoe Bay, Alaska discoveries. In 1969, Syncrude received permission to proceed with plans for a plant, and in 1971, authorization was given for this plant to produce 125,000 BPCD of synthetic crude oil by 1984.

Economic Geology

The prime mining area is located immediately adjacent G.C.O.S Lease No. 86, and the overall economic geology of the Syncrude deposit is similar to that of the previously discussed G.C.O.S. ore body. The overburden is soil and muskeg underlain by glacial drift (Pleistocene material), sands and shales of the Clearwater Formation, and uneconomic oil sands with less than 6 % bitumen (top reject). Within the prime mining area, the overburden averages about 23 m, but varies from 1.5 to 43 m.

The cutoffs used by Syncrude for extraction plant feed are 6 weight percent of bitumen and, in general, a minimum 1.5 m zone thickness. Low-grade oil sands and non-hydrocarbon bearing materials occurring in the ore body in beds thinner than 1.5 m are included in the plant feed. If thicker than 1.5 m, they are classed as "centre reject" material and handled as waste.

The cumulative thickness of the economic oil sands averages about 40 m, but ranges from 6 to 55 m. The average in situ bitumen saturation is about 11.6 % and the average fines content of the oil sands is about 15 %.

Muskeg Removal

Muskeg exposed by site clearing is drained by a systematic array of ditches approximately 1.5 m wide and 1.5 m deep. In the absence of actual measurements, the muskeg is estimated to average 1 m in thickness. Actual thicknesses range from zero to in excess of 4.5 m. After drainage, the muskeg is removed by a truck and loader operation similar to that of G.C.O.S. This work is generally carried out during the winter months, when the muskeg is frozen and trafficability is improved. Muskeg is stored in piles located around the site for use in reclaiming land disturbed by construction and mining activities.

Mining Operation

The Syncrude operation uses draglines for mining, bucket wheel reclaimers for loading, and belt conveyors for transporting mined oil sands to the extraction plant. For about the first five years of operation, the overburden and the oil sands will be excavated with four-61 m^3 draglines (2-Marion model 8750 and 2-Bucyrus Erie model 2570 W). Techman/RC estimates that each machine is capable of producing about 14,000,000 bank m^3 (based on 230,000 m^3 per year per m^3 of bucket capacity) annually. The draglines will operate from a prepared working bench along each side of the initial opening boxcut. The present mine design requires the draglines to 'chop cut' overburden above the working bench. The draglines mine 24 m wide strips in a north-south direction outward from the opening cut. The feed-grade oil sands excavated from below the working bench are placed in windrows parallel to the pit wall, and the waste material is cast directly onto the pit floor of the mined-out area. The highwall is designed to be excavated at an overall angle of 50° from toe to crest. Water sands, which underlie the oil sands throughout much of the mining area, contain saline aquifers that are under considerable piezometric pressure. Hydrological studies have shown that it is necessary to depressurize the basal aquifers well in advance of mining. Depressurization wells have been installed around the perimeter of the mining area. At an ore grade of 11.5 %, approximately 235,800 tonnes of oil sands must be mined daily to produce 130,000 BPCD of synthetic crude oil.

The oil sands placed in windrows by each dragline, are reclaimed by the bucket wheel reclaimer that is paired with the dragline. The four reclaimers, with a rated output of 4,990 tonnes/hr. each, were manufactured in Germany by Orenstein and Koppel LMG. They discharge, via a bridge conveyor, onto a conveyor system that consists of four-1,829 mm wide shiftable face conveyors running parallel to the pit; and four-1,829 mm wide collecting conveyors that transfer the plant feed to a radial stacker arrangement immediately adjacent the extraction plant. The face conveyors are about 4,270 m long, and are skid-mounted so that they can be shifted in conjunction with the mining advances.

Bitumen Extraction and Upgrading

Syncrude Canada Ltd. utilizes the Clark Hot Water Extraction process and dilution centrifuging to separate bitumen from the plant feed. The materials handling and process unit operations contained in the extraction plant are similar to those utilized by G.C.O.S., except that the Syncrude extraction operation was designed to process approximately twice the tonnage of oil sands.

The upgrading plants receive diluted bitumen from the extraction plant and upgrade it to a mineral-free crude oil with reduced nitrogen and sulfur content. Upgrading is accomplished using a fluid coking process to produce gas, butanes, naphtha, gas-oil and coke. The naphtha and gas-oil is hydrotreated and blended to form a synthetic crude oil which is suitable as a conventional refinery feed stock. A 559 mm diameter oil pipeline links the plant with facilities in Edmonton.

Tailings Disposal

A large area within the Beaver Creek valley north of the plant site was selected for a tailings disposal area. The site covers an area of approximately 2,800 hectares and has a volume sufficient to contain at least eight years of tailings. Three starter dykes were constructed in the pre-production phase across the valley. The main dykes will be raised by the upstream 'step-over' method to an ultimate maximum crest height of 80 m with sand tailings. When sufficient space becomes available in the pit tailings will be diverted to the mined-out area. Sludge and water from the in- pit disposal area will be pumped to the main tailings pond in the Beaver Creek valley. Prior to in-pit disposal of tailings, a portion of the mined-out pit will be used to store overburden. At this time, it is not clear whether the out-of-pit tailings pond will remain or if it will, in due time, be removed to allow mining of oil sands covered by the pond.

At full production, the Syncrude extraction plant will generate approximately 103 million m^3 of tailings per year of which approximately 48 million m^3 is water to be recycled through the extraction plant.

1.3.3 GENERAL DESCRIPTION OF CURRENT AND PROPOSED MINING METHODS

A brief overview (Figure 1.3.3-1 to Figure 1.3.3-24) of basic operating techniques utilized at current operating mines and proposed for future operations follows.

Figure 1.3.3-1

Prior to the start of mining, surface drainage is established by cutting ditches through the muskeg using small draglines or large backhoes. A dewatering period of three to five years prior to overburden removal may be required. Trees are cleared during the winter. Ditching may be done both in winter and in summer, depending on local field conditions.

Figure 1.3.3-2

A network of ditches drains the surface water. Spacing depends on the characteristics of the local material as well as its depth. Overburden dewatering is done by means of wells.

Figure 1.3.3-3

Muskeg is excavated using rubber-tired front-end loaders, and is transported to the storage sites with off-highway mining trucks. This operation is done during the winter to take advantage of the improved trafficability of the frozen terrain. The muskeg removed is not completely frozen and may vary in depth locally.

Figure 1.3.3-4

Muskeg is transported by trucks to storage sites: permanent, if not needed for reclamation, or temporary, if needed later for reclamation.

Figure 1.3.3-5

The G.C.O.S. operation employs two BWE's to load overburden and oil sands. The overburden BWE dumps directly into large off-highway trucks, which transport the material either to overburden dumps or to construction sites (earth-filled dykes). The two oil sands BWE's load directly into belt wagons, which in turn load the oil sands onto conveyors.

Figure 1.3.3-6

The high cut is the typical mode of operating a BWE. The excavating wheel rotates (in a clockwise direction in the photograph) while the boom of the machine slews along the face at the selected height for about ninety degrees. With each swing a thin slice of material is removed. The machine and its conveyor bridge move a short distance forward and slew in the reverse direction. After a series of these moves the boom is lowered, and the series of slewing cuts is repeated. Material can be excavated below the level of the crawlers by lowering the boom and cutting an arc-shaped trench in front of the excavator. The machine then moves forward as before. A deep cut can also be achieved by reversing the buckets and changing the direction of rotation of the wheel. Either a bridge (as shown in the photograph) or a belt wagon (as shown in Figure 1.3.3-5) is used to transfer the material onto conveyor belts. The bridge and belt wagon allow for variation in distance between the excavator and the conveyor. Bucket wheel reclaimers (BWR) used at Syncrude Canada Ltd. operate in the same fashion.

Figure 1.3.3-7

Draglines may be employed to excavate oil sands. The photograph shows a dragline at the Syncrude mine in position to dig oil sands and stack the material in a windrow alongside the excavation. A BWR loads the material from the windrow via a conveyor bridge onto a conveyor paralleling the excavation.

Figure 1.3.3-8

A dragline digs material from below the base of the machine by dragging the bucket upward along a sloping working face. Once the bucket is full, the bucket is hoisted upward and the dragline begins to rotate.

Figure 1.3.3-9

When the machine has rotated so that the bucket has reached a specified dumping point, the open end of the bucket is lowered, and the material flows out of the bucket. The photograph shows the dragline rotating towards the windrow. The windrow serves as a surge storage between the mine and the oil sands extraction plant.

Figure 1.3.3-10

As well as being placed into a windrow for later removal, the oil sands may be loaded directly onto a belt conveyor via an adequately-designed mobile hopper. This would eliminate double handling of the mined material. The photograph shows a hopper being used to load coal onto a belt conveyor. In an oil sands mining operation, such a hopper would not only be much larger, but also be designed to operate somewhat differently. The size would be such that a free-flowing face could always be maintained. If oversized material were expected, a grizzly would be moved into position. A combination of apron feeder and conveyor belt would be used to move the material between the hopper and the belt conveyor transporting the material to the extraction plant.

Figure 1.3.3-11

Conveyors situated on various mining benches transfer the oil sands and overburden to major trunk conveyors leading to the extraction plant and to the overburden dump. The material from one conveyor is dropped onto the next conveyor. Such transfer points are also needed whenever a change in direction of the conveyor is required or when the conveyor length has reached its practical operating limit. The electric motors that drive the conveyor coming from the mining bench can be seen in this photograph.

Figure 1.3.3-12

In a multiple-bench mining operation, various types of material may be excavated on each bench. This material must be directed to various locations. The re-directing of the various materials is made at a conveyor distribution point. Conveyors are arranged so that the adjustable end of the conveyor bringing the material can cross over the conveyors receiving the material. A controller selects the conveyor onto which the incoming material is to be dumped by lengthening or shortening the incoming conveyor.

Figure 1.3.3-13 and 1.3.3-14

In an oil sands mine various types of material may be conveyed: overburden (OB), low grade oil sands from above the pay zone (TR), plant feed-grade oil sands (PF), or oil sands with an unacceptably low grade from within the pay zone (CR). All material other than the plant feed is directed to waste disposal areas located either in-pit or out-of-pit. Material is placed into these dumps by means of a spreader. Spreaders may stack materials below the base of the machine (low dump) or above the base of the machine (high dump).

Figure 1.3.3-13 shows a spreader operating in the low dump mode. Figure 1.3.3-14 is an example of a spreader operating in the high dump mode.

In the future it may be possible to design oil sands extraction plants to produce dry tailings. Tailings ponds would become unnecessary, and the tailings sand would be placed by means of spreaders as shown.

Figure 1.3.3-15

The type and arrangement of prime movers (the major digging machines) will vary between oil sands mines. The choices of GCOS and Syncrude have already been described. The selection of prime excavators as employed in the mine plans studied in this report is shown for both the dragline and the bucket wheel mines.

In the dragline mines, overburden is dug by BWE and removed via conveyor. The oil sands are mined in two benches by draglines, which dig the oil sands from in front of and below the elevation of the base of the machine. The oil sands are dropped into a hopper which feeds onto a conveyor. In the bottom bench only, waste (material not suitable for plant feed) is dropped onto the pit floor by back-casting with the dragline.

In the bucket wheel mines, both the overburden and the oil sands are removed by BWE. When the pit bottom is weak or wet the BWE removes the lowest portion of the payzone by deep cutting (excavating at an elevation below the crawler level of the excavator). The remainder of the oil sands is dug by means of a high cut, as shown, by the BWE on the 1st and 2nd benches.

Figure 1.3.3-16

Three types of material must be returned to the void created by the removal of overburden and the excavation of oil sands: overburden, tailings (either wet or dry), centre reject, and oversize reject. At the Minimum and Improved Levels of Reclamation, wet tailings are returned to the pit by pipeline and stored as sand, sludge, and water. A portion of the tailings sand is also used to construct containment dykes. Placement of overburden and reject is done by spreader. At the Enhanced Level of Reclamation the tailings are conveyable, and are placed along with the overburden and reject by spreader.

Figure 1.3.3-17 and 1.3.3-18

Unless a completely enclosed void such as a mined-out pit is available, some containment structures are required. Those structures, known as dykes, can be constructed from overburden or from tailings sand. The availability of these materials will determine the relative quantities of each material used. Dyke construction using the sand portion of the tailings stream is achieved by allowing the sand to settle within a settling cell, and allowing the sludge and water to overflow from the cell. Dozers form low perimeter dykes by pushing sand into a continuous encircling pile. The tailings line is then directed into the cell from one end, and the fluid portion is removed at the other end of the longitudinal cell. The sand which settles in the cells is compacted by the dozers.

Figure 1.3.3-19 and 1.3.3-20

The mined-out portion of an oil sands mine can be used to store tailings. If desired, portions of void can be filled with sand, while other portions can be filled with sludge. Figure 1.3.3-19 shows a mined-out area being completely filled with settled tailings sand. Figure 1.3.3-20 is from the opposite side of the same pond, and shows a dyke being constructed both of tailings sand and overburden. To the left side of the photograph can be seen a dyke constructed only from overburden.

Figure 1.3.3-21

Dyke construction can only occur during that portion of the year when freezing will not hinder the separation of sand, sludge and water.

During the remainder of the year the tailings are spigotted into the pond from the end of a tailings line. Separation of sand, sludge and water occurs as the sand drifts below the liquid portion of the pond. The slope of such beached sands is very shallow.

Figure 1.3.3-22

Roughly two-thirds of the material handled in an oil sands mine is sent to the extraction plant for the removal of bitumen, and ninety per cent of this material is returned as tailings sand and clay. In addition, large volumes of water have been added if a hot water process is used. Hot tailings are transported from the extraction plant via pipelines to the tailings pond. The number of lines as well as the diameter will vary with mine size. The mines designed in this study use 5 to 7 lines from 500 mm to 610 mm in diameter. The photograph shows some of the tailings lines in use at the GCOS operation.

Figure 1.3.3-23

Usually the construction of a dyke using tailings sand cannot begin unless a starter dyke is built. This dyke is constructed from selected overburden, and forms the base upon which the sand portion of the dyke is constructed.

Figure 1.3.3-24

Dykes can be constructed to considerable height. The GCOS dyke shown is nearly 100 m high. Such ponds could eventually be completely filled with tailings sand if another area of the mine is utilized to store only sludges.

(G.C.O.S. — Photo L.T.)

Figure 1.3.3-1 Dewatering ditch in muskeg.

(Syncrude — Photo J.H.)

Figure 1.3.3-2 An array of muskeg dewatering ditches.

(G.C.O.S. — Photo H.N.)

Figure 1.3.3-3 Excavating muskeg during the winter with front-end loader and hauling with off-highway truck.

(Syncrude — Photo J.H.)

Figure 1.3.3-4 Surface of a muskeg dump.

(Photo G.C.O.S.)

Figure 1.3.3-5 Bucket wheel excavators (BWE), belt wagons, and plant feed conveyors at the G.C.O.S. mine.

(Photo R.C. Nr 18G513)

Figure 1.3.3-6   Bucket wheel excavators digging at working faces.

(Syncrude — Photo J.H.)

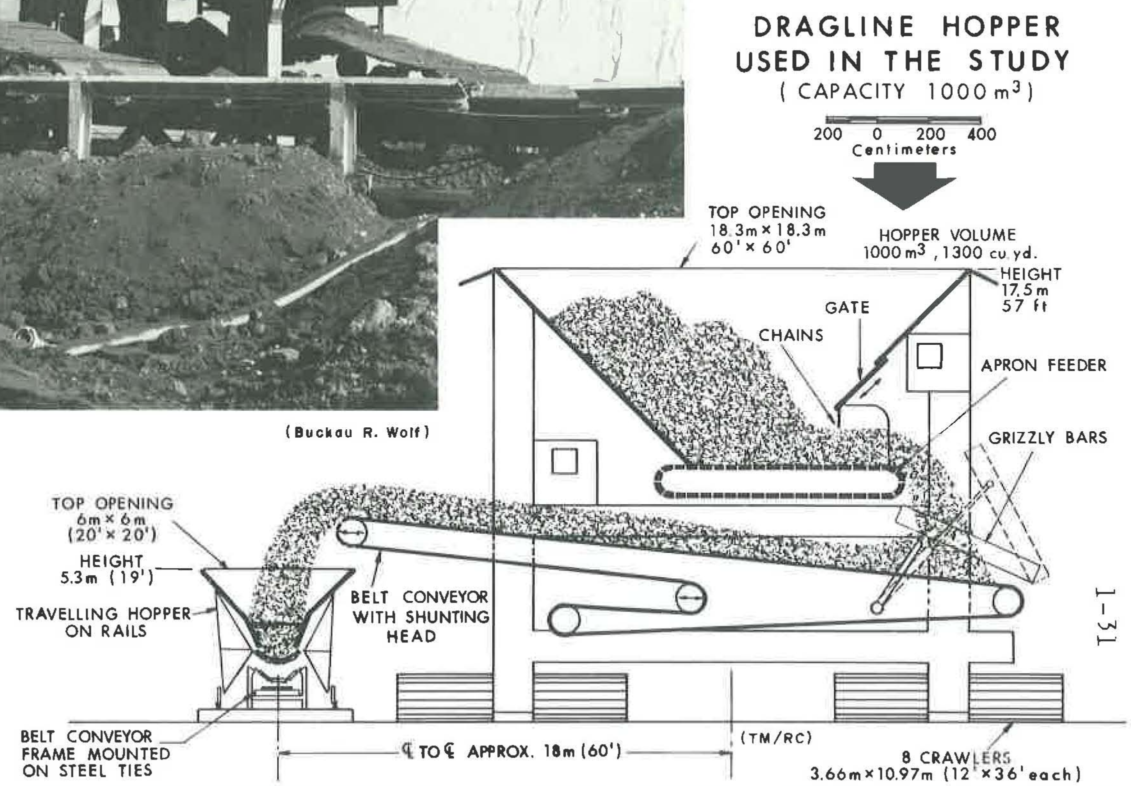
Figure 1.3.3-7 A single bench dragline mining and bucket wheel reclaimer loading belt conveyor at the Syncrude mine.

(Syncrude — Photo W.V.)

Figure 1.3.3-8 A dragline bucket being filled with oil sands by dragging along the working face.

(Syncrude — Photo W.V.)

Figure 1.3.3-9 A dragline swinging into position to drop oil sands from bucket into a windrow.



DRAGLINE HOPPER

FIGURE 1.3.3 - 10

(Photo R.C.)

Figure 1.3.3-11 A conveyor transfer point with conveyor drive station.

(Photo R.C. Nr 18H554)

Figure 1.3.3-12 A conveyor distribution point.

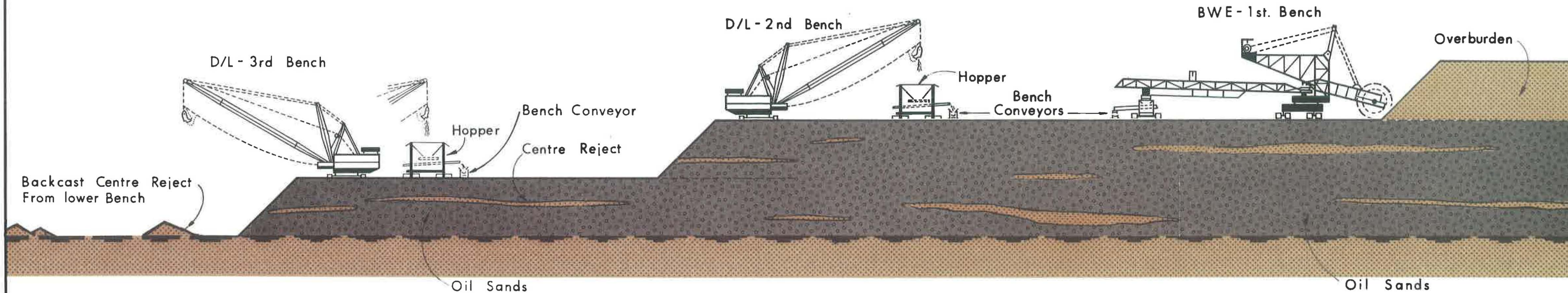
(Photo R.C.)

Figure 1.3.3-13 Spreader operating in low dump mode.

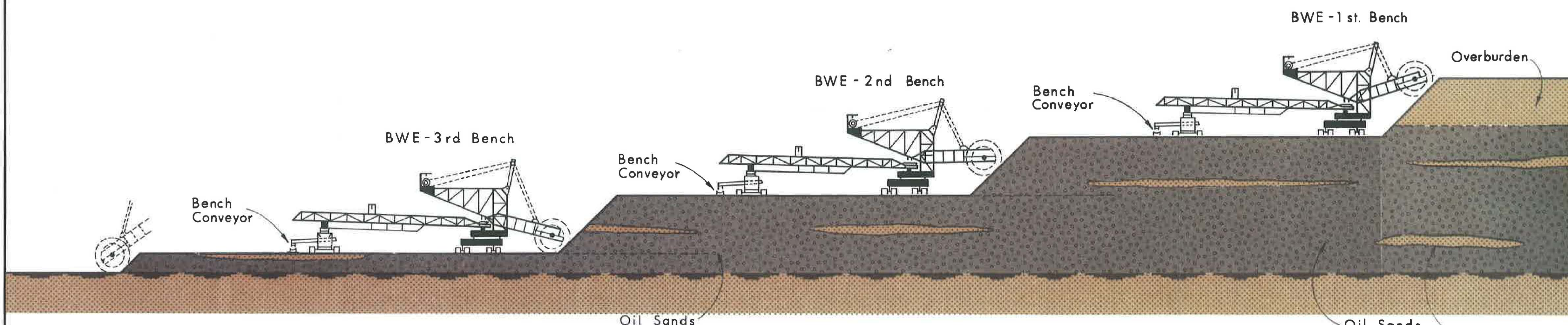
(Photo R.C.)

Figure 1.3.3-14 Spreader operating in high dump mode.

ARRANGEMENT OF OVERBURDEN REMOVAL AND OIL SANDS MINING EQUIPMENT AS EMPLOYED IN THE DETAILED MINE PLANS PRESENTED IN THIS REPORT



SCHEMATIC SHOWING ARRANGEMENT OF PRIME EXCAVATORS IN THE DRAGLINE MINING SCHEME



SCHEMATIC SHOWING ARRANGEMENT OF PRIME EXCAVATORS IN BUCKET WHEEL MINING SCHEME

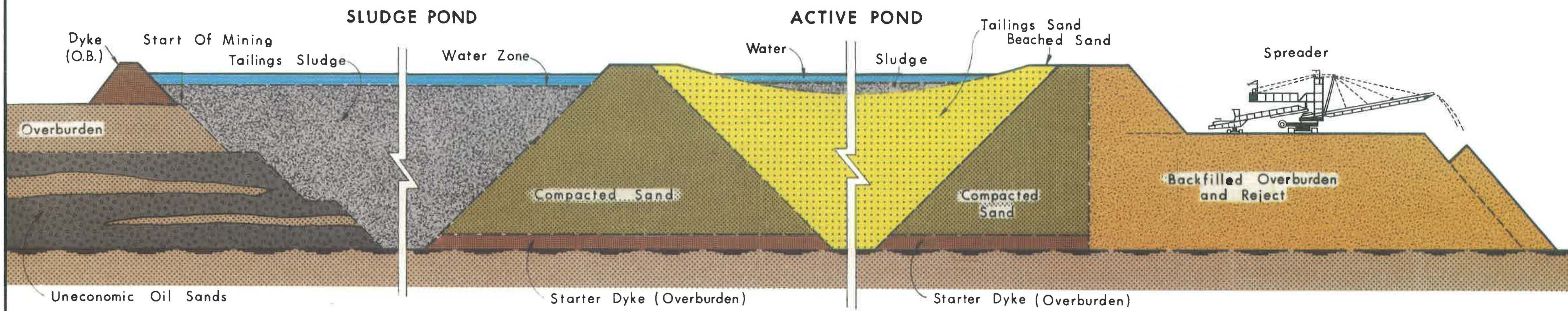
COMBINATIONS OF PRIME EXCAVATORS USED IN THE STUDY:

120,000 BPCD - 1 BWE & 4 DRAGLINES
120,000 BPCD - 3 BWE
60,000 BPCD - 1 BWE & 2 DRAGLINES
60,000 BPCD - 3 BWE
240,000 BPCD - 6 BWE

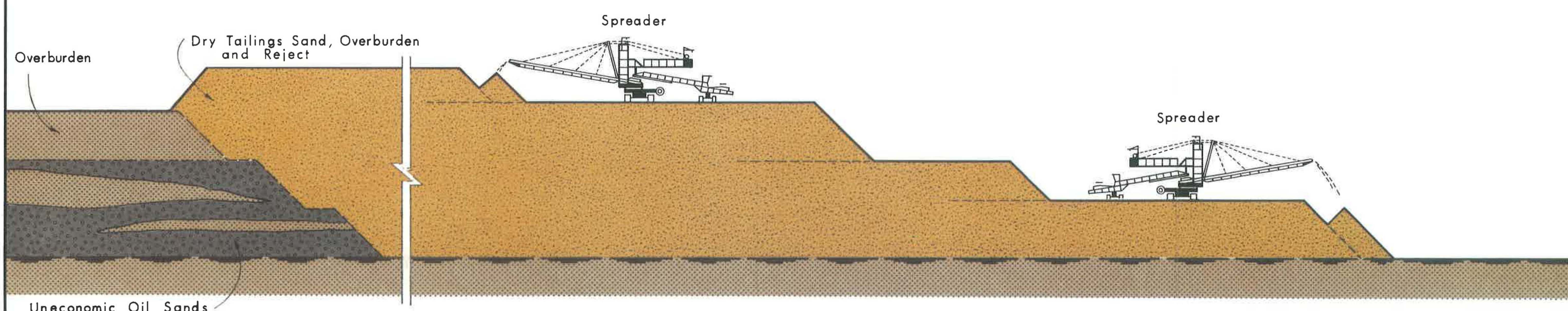
FIGURE 1.3.3-15

NOT TO SCALE

TYPICAL SEQUENCES OF TAILINGS DISPOSAL AND MINE BACKFILLING AS EMPLOYED IN THE DETAILED MINE PLANS PRESENTED IN THIS REPORT



SCHEMATIC SHOWING ARRANGEMENT OF TAILINGS DISPOSAL PONDS AND MINE BACKFILLING AT THE MINIMUM AND IMPROVED LEVELS OF RECLAMATION



SCHEMATIC SHOWING ARRANGEMENT OF MINE BACKFILLING WITH DRY TAILINGS, OVERBURDEN AND REJECT AT THE ENHANCED LEVEL OF RECLAMATION

FIGURE 1.3.3-16

NOT TO SCALE

(Photo G.C.O.S.)

Figure 1.3.3-17 Dozers constructing tailings dyke cells.

(Photo G.C.O.S.)

Figure 1.3.3-18 Tailings being dumped into cells from one of the tailings pipelines.

(Photo G.C.O.S.)

Figure 1.3.3-19 View of in-pit cell construction at the G.C.O.S. mine.

(G.C.O.S. — Photo J.H.)

Figure 1.3.3-20 In-pit dyke construction at the G.C.O.S. mine will allow tailings to be backfilled into the empty pits.

(Photo G.C.O.S.)

Figure 1.3.3-21 Tailings pond beach (in foreground); spigotting of slurry into pond (in background)

(Syncrude — Photo J.H.)

Figure 1.3.3-22 Tailings pipelines alongside a cell being constructed at the base of the tailings pond dyke.

(Syncrude — Photo J.H.)

Figure 1.3.3-23 The earth-filled starter dam of an out-of-pit tailings pond.

(G.C.O.S. — Photo J.H.)

Figure 1.3.3-24 An out-of-pit tailings pond approaching ultimate operational height.

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2.0 DEFINITION OF STUDY AREA

2.1 INTRODUCTION

The oil sands of Alberta are found in four major deposits: the Athabasca deposit, which is the largest and only deposit workable by surface mining methods, the Cold Lake deposit, the Wabasca deposit, and the Peace River deposit. The area chosen for this study is illustrated in Drawing No. B22910-01-00, 'Ore Body Locations Within Regional Mining Area', and covers 2,300 sq. km (25 townships from Township 94 to 98 between Ranges 7 and 11 west of the Fourth Meridian) in the Athabasca region north of Fort McMurray, Alberta. An appendix to section 2.0 defines the terms, factors and equations used in this section.

2.2 DRILLING INFORMATION

In connection with this study, Techman/RC obtained access to relevant oil sands drilling information on file at the Energy Resources Conservation Board (ERCB) offices in Calgary. These drill hole data were in the public domain as of November, 1977. Techman/RC was also granted access to confidential geophysical well log interpretations made by ERCB oil sands geologists from 1,862 drill holes in the study area (0.80 holes per sq. km). These log interpretations were in the form of drill hole data sheets, from which Techman/RC extracted the following borehole information:

- surveyed borehole location
- borehole collar elevation
- overburden thickness
- top reject thickness
- centre reject thickness
- plant feed thicknesses (+5% bitumen content)
- average bitumen saturation
- surface mineability factor

It should be noted that the log interpretations were based on a five weight percent minimum bitumen saturation cutoff.

2.3 TECHMAN/RC ISOPACH MAPS

Data from the ERCB log interpretations were plotted by computer, and a set of 1:50,000 scale isopach maps of the study area was generated which showed various formation thicknesses and oil sands quality factors. These isopach maps illustrated such factors as:

a. Overburden Thickness: total thickness of Pleistocene material, sands and clays of the Clearwater Formation and low-grade oil sands (top reject) above the uppermost pay zone.

b. Net Pay Zone Thickness: total thickness of +5% oil sands and associated centre reject material that would be mined as plant feed. For the purposes of this study, a 5% minimum bitumen saturation has been used. The ERCB considers this to be a minimum economic cutoff based on present-day economics and existing technology.

c. Mining Depth: total thickness of all plant feed grade and waste materials from ground surface to the top of the bottom reject zone.

d. GRAMT: a Techman/RC oil sands quality designation of average oil sands grade (GRA) multiplied by net pay zone thickness in metres (MT).

e. Surface Mineability Factor[1] (S.M.F.): an ERCB oil sands quality relationship calculated from the following equation:

$$S.M.F. = \frac{S}{1 + 0.9\ (Tw/To)}$$

where

S = average bitumen saturation in weight percent

Tw = total waste thickness

To = total plant feed thickness (net pay zone thickness)

Where the S.M.F. is less than a value of 5 for an oil sands deposit, the ERCB currently considers the deposit to be uneconomic, recognizing however, that the economic conditions will change with time and improving technology.

f. R-Factor : Syncrude Canada Ltd. economic factor[2] for oil sands recoverable by surface mining:

$$\text{R-Factor} = \frac{\text{Volume of bitumen in plant feed (barrels)}}{\text{Volume of total material moved (cu.yds.)}}$$

$$= \frac{d \times g \times 9.53}{D}$$

where:

d = thickness of oil sands (net pay zone thickness)

D = total depth from surface to bottom of oil sands formation (mining depth)

g = oil sands grade in weight percent

$$9.53 = \frac{\text{weight of one cubic yard of oil sands}}{\text{weight of one barrel of bitumen}}$$

The following descriptive terms pertaining to mineability were applied to the calculated ranges for this oil sands quality factor[3]:

Above 1.0 . . . . choice mineability
0.75 . . . . good mineability
0.50 . . . . fair mineability
0.25 . . . . marginal mineability
Below 0.25 . . . . unattractive mineability

Published geostatistical studies[4] of oil sands deposits within Bituminous Sands Lease 13 indicate that the drill hole sampling done to date on Lease 13 is sufficient for locating all major ore bodies and for accurate estimation of bitumen reserves. The statistical data show that variogram ranges (radii of influence) of net pay zone thickness and oil sands grade are 1,200 to 1,500 m. This is to say that, for adequate ore body location and definition, drill hole spacing should be less than 1,500 m.

A visual inspection of drill hole locations and spacings within the study area indicates that the drill hole density is probably adequate for major ore body location on the following Bituminous Sands Leases:

Lease No.	Lessee
13	Shell Canada Resources Limited/Shell Explorer Limited
18 (South half only)	Hudson's Bay Oil and Gas Co. Ltd.
24 (North half only)	BP Exploration Canada Limited
30	Home Oil Company Limited
34, 95 and 96	Petrofina Canada Ltd.
36	Mobil Oil Canada Ltd.
87 (North half only)	CDC Oil & Gas Limited/Tenneco Oil of Canada Ltd.
88	Amerada Minerals Corporation of Canada Ltd.
98	Sun Oil Company Limited

Unfortunately, these leases only cover about 40% of the study area and it might be argued that the Techman/RC isopach maps do not reflect the true economic geology within the study area. However, given the observed continuity of the McMurray oil-bearing formation over the entire Athabasca region, Techman/RC believe that the location and density of drill holes made available by the ERCB are adequate to delineate large areas of economically recoverable oil sands within the study area.

2.4 ORE BODY DELINEATION

Techman/RC assessed the relative merits of each of the six isopach maps with respect to its suitability for ore body delineation. It was decided that the two most useful isopach maps for this purpose were those showing "GRAMT" and "R-Factor", since a combination of these two maps took into account:

- overburden thickness
- net pay zone thickness
- centre reject thickness
- mining depth
- bitumen saturation
- tonnes of bitumen per cubic metre of total material moved

Techman/RC considered that the combination of the geologic criteria GRAMT and R-Factor adequately defined mineable oil sands areas for the purposes of this study. This combination provided an optimization of bitumen grade, pay zone, and waste material with GRAMT tending to favour bitumen content, and R-Factor providing an indication of mining economics.

Table 2.4-1 is a summary of nineteen ore bodies resulting from an application of 400 GRAMT and 0.60 R-Factor cutoff. Overall GRAMT and R-Factor were correlated with GRAMT and R-Factors calculated from oil sands quality information in the public domain pertaining to the G.C.O.S. operations on Lease 86[4], the Syncrude project on Lease 17[5], and the proposed Shell project on Lease 13[6,7] (prior to the formation of the Alsands Project Group). The average GRAMT was 456 and R-Factor 0.76. Specific ore body quality was based on information not allowing for dilution at reject/plant feed interfaces and for extra overburden removed at pit walls. Actual pit mining R-Factors will go down depending on the mining plan.

The relative mining economics of all three oil sands projects were assessed, and Techman/RC decided to select a quality limit that would delineate mineable ore bodies similar to those presently considered viable by the operators under present-day economics and existing technology. In general, for the purposes of this study, Techman/RC used a minimum economic cutoff of GRAMT greater than or equal to 400 and R-Factor greater than or equal to 0.600, and potential ore bodies would have to satisfy both conditions. The three ore bodies subjected to detailed design in this study averaged GRAMT=567 and R-Factor=0.74. Since this study was not restricted by Bituminous Sands Lease boundaries, it was possible to follow the economic geology to predict the most logical mining areas based upon the quality of the ore bodies.

The isopach maps of GRAMT and R-Factor for the study area were overlain and the boundaries of the ore bodies were traced. In this manner, a total of 19 ore bodies were delineated ranging in size from about 140 to 4,160 hectares. As well, the 0.3 R-Factor limit was shown to help assess the relative positioning of the ore bodies within the generally surface mineable region.

Drilling information from boreholes within each ore body was compiled and analyzed to determine overall average thicknesses and quantities of ore and waste, oil sands quality and in-place bitumen reserves. The Techman/RC mapping procedure did not take into account the topographic low of the Athabasca River valley. For example, ore bodies appear to be continuous across the valley, when in fact, the McMurray Formation has been eroded by the river. In effect, then, the mapping assumes a flat, pre-erosional landform.

A detailed breakdown of in-place bitumen reserves for each of the nineteen ore bodies mentioned in Section 2.4 is given in Table 2.4-1. The ultimate life expectancy for each ore body (based on GRAMT greater than or equal to 400 and R-Factor greater than or equal to 0.6) was calculated for the 60,000, 120,000 and 240,000 BPCD synthetic crude oil production cases.

2.5 SITE SELECTION CRITERIA

Major and minor considerations for ore body selection, not necessarily in order of importance, are listed as follows:

Major Considerations

a. Each ore body should have an optimum combination of thin overburden, thick pay zone, high bitumen saturation, and low fines content.

IN PLACE BITUMEN RESERVE SUMMARY

TABLE 2.4-1

PARAMETERS \ ORE BODIES	1	2	3A	3B	3C	3D	4	5	6	7	8	9A	9B	9C	10A	10B	10C	11A	11B
AREA (HECTARES)	4,161	1,967	242	221	393	217	1,502	1,103	3,307	727	412	666	291	140	904	170	323	455	470
DRILL HOLES	20	29	5	5	5	3	19	23	167	36	36	10	2	2	11	1	9	7	18
DENSITY (HECTARES/HOLE)	208	68	48	44	79	72	79	48	20	20	11	67	146	70	82	170	36	65	26
OVERBURDEN THICKNESS (METRES)	21.03	11 26	13.72	15.12	8.96	11.27	19.78	12.35	9.03	21.08	21.82	12 31	7.60	14.50	12.01	19.50	14 90	27.99	9.15
+5 TARSAND PAYZONE THICKNESS (METRES)	53.09	46 87	57.00	52.78	52.32	56.80	48.64	53.62	50.90	52.96	49.96	57.67	41.15	39.05	51.53	42.10	42 67	35 31	36.78
CENTRE REJECT THICKNESS (METRES)	10.15	8.95	13.16	11.34	3.94	28.53	9.43	10.51	8.20	9.56	8.30	21.40	15.25	20.95	5.96	5.20	11.59	0.80	4.96
TOTAL WASTE THICKNESS (METRES)	31.18	20.21	26.88	26.46	12.90	39.80	29.21	22.86	17.23	30 64	30.12	33.71	22.85	35.45	17.97	24.70	26.49	28 79	14.11
MINING DEPTH (METRES)	84.27	67.08	83.88	79.24	65.22	96.60	77.85	76.48	68.03	83.60	80.08	91.38	64 00	74.50	69.50	66.80	69.16	64.10	50.89
BITUMEN SATURATION (%)	12.22	11.99	9.84	11.15	10.60	11.58	11.88	12.04	11.79	12.05	11 98	12.09	12.13	12.34	10.61	11.84	11 34	12.85	12.50
FINES CONTENT (%)	11.12	12.04	20.64	15.48	17.60	13.68	12 48	11.84	12 84	11 80	12.08	11.64	11.48	10.64	17 56	12 64	14.64	8.60	10.00
WASTE/ORE RATIO	0.59	0.43	0.47	0.50	0.25	0.70	0.60	0.43	0 34	0 58	0.60	0 58	0.56	0 91	0.35	0.59	0 62	0 82	0.38
GRANT (GRADE-METRES)	649	562	561	588	555	658	578	646	599	638	599	697	499	482	547	498	484	454	460
R-FACTOR (BBLS/BCY)	0.73	0.80	0.64	0.71	0.81	0.65	0.71	0.80	0.84	0.73	0.71	0.73	0.74	0.62	0.75	0.71	0.67	0 67	0.86
TSMF	7.99	8.64	6.91	7.68	8.67	7.10	7.71	8.70	9.03	7.92	7.77	7 92	8.09	6.79	8.08	7.75	7 28	7 41	9.29
OVERBURDEN VOLUME (MILLION METRES^3)	875.0	221.5	33.2	33.4	35.2	24.5	297.1	136.2	298.6	153.3	89.9	82.0	22.1	20.3	108.6	33.2	48 1	127.3	43.0
+5% TARSAND VOLUME (MILLION METRES^3)	2,209.1	921.9	137.9	116.6	205.6	123.3	730.6	591.4	1,679.9	385.0	205.8	384.1	119.7	54.6	465.8	71.6	137.8	160.7	172.9
+5% TARSAND WEIGHT (MILLION TONNES)	4,423.2	1,845.9	276.2	233.6	411.7	246.8	1,462.8	1,184.2	3,363.8	770.9	412.1	768.0	239.8	109.5	932.7	143.3	275.9	321.7	346.1
CENTRE REJECT VOLUME (MILLION METRES^3)	422.3	176.0	31.8	25.1	15.5	61 9	141.6	115.9	271.2	69.5	34.2	142.5	44.4	29.3	53 9	8.8	37.4	3 6	23.3
IN SITU BITUMEN RESERVE (MILLION TONNES)	540.5	221.3	27.2	26.0	43.6	28.6	173.8	142.6	396.6	92.9	49.4	93.0	29.1	13.5	99.0	17.0	31.3	41.3	43.3
TONNES OF BITUMEN/HECTARE	129,897	112,506	112,397	117,647	110,941	131,797	115,712	129,284	119,927	127,785	119,903	139,640	100,000	96.429	109,513	100,000	96,904	90,769	92,128
ORE BODY LIFE* (YEARS) @ 60,000 BPCD	120.5	49.4	6.1	5.8	9.7	6.4	38.8	31.8	88.5	20.7	11.0	20.7	6.5	3.0	22.1	3.8	7.0	9.2	9.7
				28.0									30.2			32 9		18.9	
@ 120,000 BPCD	60.3	24.7	3.0	2.9	4.9	3.2	19.4	15.9	44.2	10.4	5.5	10.4	3.2	1.5	11.0	1.9	3.5	4.6	4.8
				14.0									15.1			16.4		9.4	
@ 240,000 BPCD	30.1	12.3	1.5	1.4	2.4	1.6	9.7	8.0	22.1	5.2	2.8	5.2	1.6	0.8	5.5	0.9	1.7	2.3	2.4
				6.9									7.6			8.1		4.7	

* CALCULATED ON THE BASIS OF:
MINING RECOVERY = 100%
EXTRACTION PLANT RECOVERY = 90.2%
SYNTHETIC CRUDE/BITUMEN RATIO = 0.868
PRODUCTION RATE = 100% CAPACITY

b. Adequate bitumen reserves should be contained within each ore body to support a synthetic crude oil facility for an approximate life span of at least 25 years.

c. The ore body should have a suitable plant site and tailings disposal area in close proximity. Potential tailings disposal sites should be located on barren or uneconomic oil sands areas, ie. thick overburden, high waste/ore ratio, etc.

d. The ultimate layout for the mine, plant, and the tailings disposal areas should not necessarily follow present day Bituminous Sands Lease boundaries.

e. The overall average maximum fines content of the plant feed should not exceed 15%. Extraction plant recovery efficiency is reduced as fines content of the plant feed increases.

f. Each ore body should be amenable to conventional dragline and bucket wheel excavator mining.

g. The density and distribution of drill holes within the ore bodies should be adequate and reasonable.

Minor Considerations

a. Because the site for the third oil sands plant in the Fort McMurray area very likely is to be located on the east side of the Athabasca River, Techman/RC determined that the study should be centred in oil sands deposits in that area.

b. Oil sands development should not take place too close to the McClelland Lake - Fort Hills area, since the area immediately west of McClelland Lake has some townsite development potential.

c. Oil sands development should be at least one mile away from the Athabasca River escarpment (environment corridor).

d. Techman/RC decided that the Great Canadian Oil Sands Ltd. (Lease 86) and Syncrude Canada Ltd. (Leases 17 and 22) project areas would be unsuitable for the purposes of this study, since little would be gained by commenting on or redesigning these existing oil sands operations.

e. Within Lease 13 of Shell Canada Resources Limited, two oil sands deposits designated by the Alsands Project Group as Shell No. 1 & No. 2 are thought to be prime development areas for the third oil sands plant in the Fort McMurray area. Techman/RC decided, therefore, that these deposits should be considered for the purposes of this study only if more suitable ore bodies could not be found.

f. Oil sands development should not take place on or in proximity to the Fort MacKay settlement or the Fort MacKay Indian Reservation No. 174.

2.6 PRELIMINARY ORE BODY EVALUATIONS

Each of the 19 major ore bodies delineated by Techman/RC was assessed and evaluated with respect to the major and minor site selection criteria discussed previously. Massive ore bodies were considered by themselves, while smaller ore bodies (in clusters) were grouped together to be serviced by a centroidally-located plant facility. The optimized ore body groupings, evaluations and the comments of Techman/RC are presented in Table 2.6-1.

On March 15, 1978, a meeting was held in Edmonton between Techman/RC and the Project Steering Committee. At that time, two choices were available to Techman/RC with respect to site selection. Techman/RC proposed development of mining, tailings disposal, and reclamation plans on three separate ore bodies for the three synthetic crude oil production cases as follows:

a. 60,000 BPCD case on Ore Body No. 4 or 5.

TABLE 2.6-1

TECHMAN/RC ORE BODY EVALUATION

Ore Bodies	Optimized Life Expectancy (Years)	Comments
1	30.1 at 240,000 BPCD	- under prime consideration for the 60,000, 120,000 and 240,000 BPCD production cases. - this ore body is of sufficient size to support all three plant sizes. - aquifer depressurization problems could be expected.
2	24.7 at 120,000 BPCD	- under prime consideration for the 120,000 BPCD case. - aquifer depressurization problems might be expected.
3A, 3B, 3C & 3D	28.0 at 60,000 BPCD	- these 4 ore bodies could be mined in one development with a centroidally-located plant site. - high fines content and low bitumen grade in 3A, 3B and 3C make this an unattractive proposition - No. 3D could be added to the reserves of Ore Body No. 1; however, the w/o ratio is fairly high.
4	38.8 at 60,000 BPCD	- under active consideration for the 60,000 BPCD case.
5	31.8 at 60,000 BPCD	- under prime consideration for the 60,000 BPCD case. - Ore Body No. 5 has higher grade and lower w/o ratio than No. 4.
6	22.1 at 240,000 BPCD	- this ore body would be considered for this study only as a last resort, as explained earlier. - this ore body encompasses the "Shell No. 1 and No. 2" tarsand ore bodies as well as the "Fina Daphne No. 3" ore body.
7	20.7 at 60,000 BPCD	- too small to be mined on its own. - could be mined in conjunction with Ore Body No. 6.
8	________	- not sufficient size to be mined by itself. - located too close to the Athabasca River.
9A, 9B & 9C	30.2 at 60,000 BPCD	- these 3 ore bodies could be mined in one development; however, 9B and 9C are located too close to the Athabasca River. - insufficient reserves to support 60,000 BPCD operation.
10A, 10B & 10C	32.9 at 60,000 BPCD	- could be mined in one development; however, a large quantity of reserves is located too close to the Athabasca River.
11A & 11B	________	- reserves not sufficient to support minimum production case.

b. 120,000 BPCD case on Ore Body No. 2.

c. 240,000 BPCD case on Ore Body No. 1.

Alternatively, Techman/RC was prepared to formulate mining, tailings disposal, and reclamation plans for all three production cases on Ore Body No. 1 alone, since it was large enough to support the three different plant sizes.

The merits of conducting the study on three ore bodies of different size were explained to the Steering Committee. Bearing in mind that the purpose of this study was to explore the interrelationship between oil sands mining activities and reclamation, Techman/RC favoured the option with three distinct ore bodies, since it was deemed to be the most cost-effective choice.

The Project Steering Committee, in response to the proposals and recommendations of Techman/RC, informed the Consultants by letter dated April 4, 1978 that the following ore bodies would be appropriate for further study:

a. No. 1 at a rate of 240,000 BPCD for 25 years.

b. No. 2 at a rate of 120,000 BPCD for 25 years.

c. No. 4 at a rate of 60,000 BPCD for 25 years.

In the case of Ore Body No. 4, the initial size was too large (by about 50%) to be considered as a whole for the 60,000 BPCD case. It was therefore necessary to discount fringe areas of the ore body in order to ultimately define an area of mineable oil sand reserves that could be exploited over a 25-year period.

2.7 FINAL ORE BODY SIZING PROCESS

Techman/RC determined that the final sizes of Ore Bodies No. 1, 2 and 4 were directly related to:

a. Mining recovery based on the mining method chosen. Techman/RC estimated the expected mining recovery for the three surface oil sands mining methods shown below:

(i) A one-bench dragline mining scheme;
(ii) A two-bench dragline mining scheme;
(iii) A three-bench bucket wheel excavator mining scheme.

b. Estimated extraction plant recovery based in part on the estimated plant feed grade and fines content. Techman/RC estimated the average extraction plant recovery to be 90.2%.

c. Bitumen upgrading recovery, i.e., conversion factor from volume of plant-recovered bitumen to volume of synthetic crude oil produced. Techman/RC estimated the average upgrading recovery to be 86.8%.

d. Derated mining and extraction plant production capacity extending from one to two years after start-up. It is doubtful that rated mine and plant capacity could be achieved immediately after commencement of synthetic crude production. Start-up problems have been anticipated and were taken into account with respect to final orebody sizing.

In the course of final ore body definition, it was necessary for Techman /RC to confirm and, in certain cases, to reevaluate the geophysical logs within or near the ore bodies. As a result of this reevaluation, changes were made to the ore body configurations originally presented to the Project Steering Committee on March 15, 1978.

A set of east-west-trending geological cross sections were prepared at three kilometre intervals through each ore body. These cross sections show the position and continuity of overburden, pay zone, and reject. As well, some indication of formations forming the pit floor are provided. These simplified cross-sections provide an overall impression of the geologic character of the ore bodies selected for detailed assessment in this study.

The final sizing of the ore body was done with the aid of specially prepared computer programs. For detailed reserve calculation and mine planning purposes, a 100 metre x 100 metre grid system was superimposed on each ore body and the surrounding low-grade areas immediately adjacent. This divided Ore Bodies No. 1, 2 and 4 into a number of contiguous blocks each one hectare in area. Values for thickness of glacial till, top reject, pay zone and centre reject, the average elevation of the topographic surface, and the top and bottom of the pay zone, as well as bitumen saturation, were determined for each hectare in the gridded areas. The effect of this digitizing process was to assign a set of these values to an imaginary drillhole at the centre of each hectare. With this digitized ore body data retained in a computer file, it was possible to evaluate ore body characteristics on an individual hectare basis.

Special attention was given to Ore Body No. 4. With an ore body quality cutoff of GRAMT = 400 and R-Factor = 0.6, the optimized life expectancy was estimated to be about 38.8 years at a synthetic crude oil production rate of 60,000 BPCD. Since this was larger than the 25 year reserve life specified for this study, Techman/RC found it necessary to reduce the size of Ore Body No. 4 by about 36%. This was accomplished by an iterative process whereby the cutoffs for GRAMT and R-Factor for each hectare block were increased while taking into account the derated plant start-up capacity and the estimated mining, extraction, and upgrading recovery efficiencies. With a cutoff of GRAMT = 500 and R-Factor = 0.6, it was found that sufficient bitumen reserves were contained in Ore Body No. 4 to support a 60,000 BPCD synthetic crude oil facility for at least 25 years.

The oil sands quality cutoffs used for Ore Bodies No. 1 and 2 were GRAMT = 400 and R-Factor = 0.6. These cutoffs were the same as those used in the original ore body delineation.

Ore body maps showing progressively less desirable ore were used to determine the direction in which changes to ore body limits should be made (see Ore Bodies No. 1 and No. 2 Under Changing Mining Cut-off, Drawing No. F22910-02-00, and Ore Body No. 4 Under Changing Mining Cut-off, Drawing No. E22910-03-00). It was found desirable to expand a mine boundary in the direction of sharply-dropping ore grades in order to reduce the loss of narrow strips of ore. It was also found desirable to adjust the boundaries to minimize the abandonment of ore strips during the mine life, since they would be expensive, if not impossible, to recover later. The possibility of ultimately expanding the mine into present- day submarginal ore was provided for, since such ore would likely become economically recoverable in the future. The final mineable ore bodies specified by the adjusted cutoff parameters of GRAMT and R-Factor were smoothed to produce the mine outlines displayed in Chapters 7.0, 8.0 and 9.0.

The ore body cut-off maps were also functional in determining the potential areas to serve as out-of-pit tailings pond sites.

Reserve calculations and pertinent mine planning data for Ore Bodies No. 1(Part A&B), 2 and 4, respectively, are summarized in Table 2.7-1, Ore Body No. 1A at GRAMT 400.00 and R-Factor 0.600; Table 2.7-2, Ore Body No. 1B at GRAMT 400.00 and R-Factor at 0.600; Table 2.7-3, Ore Body No. 2 at GRAMT 400.00 and R-Factor 0.600; and Table 2.7-4, Ore Body No. 4 at GRAMT 500.00 and R-Factor 0.600.

2.8 GENERAL COMMENTS REGARDING ORE BODY DELINEATION

Mineable oil sands reserves for each ore body were calculated on the assumption that intended mining operations are to be carried out in such a manner as to facilitate the recovery of all economic oil sands within a general area designated for mining. Consequently, the ore body limits may include islands of ore with a cut-off lower than that selected for the ore body. Since ore bodies are zoned in the vertical direction with alternating layers of ore grade and reject materials, the increase in incremental ratio in the vertical direction must also be considered. Based on economics alone, bands of ore grade oil sands may remain unmined because of excess centre reject overlying these bands. Incremental ratios could also be applied to bottom reject, reflecting the increase in extraction costs due to falling overall bitumen grades and in-

Table 2.7-1

ORE BODY NO. 1A (SOUTHERN SECTOR) AT GRAMT ≥ 400.00 AND R-FACTOR ≥ 0.600

Overburden - Bank Cubic Metres	379,360,000.00
Pe/CW Material - Bank Cubic	211,987,920.00
Top Reject - Bank Cubic Metres	167,372,094.00
Centre Reject - Bank Cubic Metres	126,257,929.00
Pay Zone - Bank Cubic Metres	1,024,733,712.00
Mean Grade - Weight Percent	11.66
Grade Standard Deviation - Weight Percent	0.90
Corresponding Area - Hectares	1,795.00
Mean Fines Content - Weight Percent	13.36
Fines Content Standard Deviation - Weight Percent	3.60
Corresponding Area - Hectares	1,795.00
Ore Body Area - Hectares	1,795.00
Mean Overburden Thickness - Metres	21.13
Overburden Standard Deviation - Metres	7.99
Corresponding Area - Hectares	1,795.00
Maximum Overburden Thickness - Metres	40.00
Minimum Overburden Thickness - Metres	3.00
Mean Pe/CW Material Thickness - Metres	11.81
Pe/CW Material Standard Deviation - Metres	7.84
Corresponding Area - Hectares	1,795.00
Maximum Pe/CW Material Thickness - Metres	37.00
Minimum Pe/CW Material Thickness - Metres	1.00
Mean Top Reject Thickness - Metres	9.56
Top Reject Standard Deviation - Metres	6.83
Corresponding Area - Hectares	1,750.00
Maximum Top Reject Thickness - Metres	31.39
Minimum Top Reject Thickness - Metres	0.00
Mean Centre Reject Thickness - Metres	9.12
Centre Reject Standard Deviation - Metres	5.05
Corresponding Area - Hectares	1,384.00
Maximum Centre Reject Thickness - Metres	31.70
Minimum Centre Reject Thickness - Metres	0.00
Mean Pay Zone Thickness - Metres	57.09
Pay Zone Standard Deviation - Metres	8.31
Corresponding Area - Hectares	1,795.00
Maximum Pay Zone Thickness - Metres	75.00
Minimum Pay Zone Thickness - Metres	31.55
Mean Mining Depth - Metres	85.26
Mining Depth Standard Deviation - Metres	10.94
Corresponding Area - Hectares	1,795.00
Maximum Mining Depth - Metres	115.00
Minimum Mining Depth - Metres	55.00
Average GRAMT	662.14
Average R-Factor	0.75
Waste-to-Ore Ratio	0.49
Synthetic Crude Yield at 90.2% Extraction and 86.8% Upgrading - Billion Barrels	1.17
Basic Mine Life at 120,000 BPCD - Years	26.68

NOTE: The data above are based on geology only, and do not include dilution, mining loss and extra overburden removed at pit walls. For data influenced by mining procedures see Table 10.6-1.

Table 2.7-2

ORE BODY NO. 1B (NORTHERN SECTOR) AT GRAMT $\geq$ 400.00 AND R-FACTOR $\geq$ 0.600

Overburden - Bank Cubic Metres	422,430,000.00
Pe/CW Material - Bank Cubic	345,602,288.00
Top Reject - Bank Cubic Metres	76,827,753.00
Centre Reject - Bank Cubic Metres	181,170,006.00
Pay Zone - Bank Cubic Metres	926,599,952.00
Mean Grade - Weight Percent	11.48
Grade Standard Deviation - Weight Percent	0.91
Corresponding Area - Hectares	1,834.00
Mean Fines Content - Weight Percent	14.09
Fines Content Standard Deviation - Weight Percent	3.66
Corresponding Area - Hectares	1,834.00
Ore Body Area - Hectares	1,834.00
Mean Overburden Thickness - Metres	23.03
Overburden Standard Deviation - Metres	8.62
Corresponding Area - Hectares	1,834.00
Maximum Overburden Thickness - Metres	64.00
Minimum Overburden Thickness - Metres	14.00
Mean Pe/CW Material Thickness - Metres	18.84
Pe/CW Material Standard Deviation - Metres	7.93
Corresponding Area - Hectares	1,834.00
Maximum Pe/CW Material Thickness - Metres	51.09
Minimum Pe/CW Material Thickness - Metres	9.28
Mean Top Reject Thickness - Metres	5.20
Top Reject Standard Deviation - Metres	3.74
Corresponding Area - Hectares	1,477.00
Maximum Top Reject Thickness - Metres	17.77
Minimum Top Reject Thickness - Metres	0.00
Mean Centre Reject Thickness - Metres	10.40
Centre Reject Standard Deviation - Metres	5.12
Corresponding Area - Hectares	1,742.00
Maximum Centre Reject Thickness - Metres	34.90
Minimum Centre Reject Thickness - Metres	0.00
Mean Pay Zone Thickness - Metres	50.52
Pay Zone Standard Deviation - Metres	7.58
Corresponding Area - Hectares	1,834.00
Maximum Pay Zone Thickness - Metres	67.00
Minimum Pay Zone Thickness - Metres	30.05
Mean Mining Depth - Metres	83.44
Mining Depth Standard Deviation - Metres	10.99
Corresponding Area - Hectares	1,834.00
Maximum Mining Depth - Metres	135.00
Minimum Mining Depth - Metres	62.00
Average GRAMT	581.87
Average R-Factor	0.66
Waste-to-Ore Ratio	0.65
Synthetic Crude Yield at 90.2% Extraction and 86.8% Upgrading - Billion Barrels	1,04
Basic Mine Life at 120,000 BPCD - Years	23.74

NOTE: The data above are based on geology only, and do not include dilution, mining loss and extra overburden removed at pit walls. For data influenced by mining procedures see Table 10.6-1.

Table 2.7-3

ORE BODY NO. 2 GRAMT ≥ 400.00 AND R-FACTOR ≥ 0.600

Overburden - Bank Cubic Metres	240,710,000.00
Pe/CW Material - Bank Cubic	181,139,696.00
Top Reject - Bank Cubic Metres	59,570,236.00
Centre Reject - Bank Cubic Metres	198,335,506.00
Pay Zone - Bank Cubic Metres	951,999,528.00
Mean Grade - Weight Percent	11.63
Grade Standard Deviation - Weight Percent	1.30
Corresponding Area - Hectares	2,132.00
Mean Fines Content - Weight Percent	13.49
Fines Content Standard Deviation - Weight Percent	5.21
Corresponding Area - Hectares	2,132.00
Ore Body Area - Hectares	2,132.00
Mean Overburden Thickness - Metres	11.35
Overburden Standard Deviation - Metres	7.08
Corresponding Area - Hectares	2,121.00
Maximum Overburden Thickness - Metres	36.00
Minimum Overburden Thickness - Metres	0.00
Mean Pe/CW Material Thickness - Metres	8.54
Pe/CW Material Standard Deviation - Metres	7.08
Corresponding Area - Hectares	2,121.00
Maximum Pe/CW Material Thickness - Metres	36.00
Minimum Pe/CW Material Thickness - Metres	0.00
Mean Top Reject Thickness - Metres	3.41
Top Reject Standard Deviation - Metres	2.66
Corresponding Area - Hectares	1,747.00
Maximum Top Reject Thickness - Metres	17.85
Minimum Top Reject Thickness - Metres	0.00
Mean Centre Reject Thickness - Metres	10.70
Centre Reject Standard Deviation - Metres	6.17
Corresponding Area - Hectares	1,853.00
Maximum Centre Reject Thickness - Metres	37.61
Minimum Centre Reject Thickness - Metres	0.00
Mean Pay Zone Thickness - Metres	44.65
Pay Zone Standard Deviation - Metres	8.40
Corresponding Area - Hectares	2,132.00
Maximum Pay Zone Thickness - Metres	69.50
Minimum Pay Zone Thickness - Metres	19.00
Mean Mining Depth - Metres	65.25
Mining Depth Standard Deviation - Metres	12.60
Corresponding Area - Hectares	2,132.00
Maximum Mining Depth - Metres	92.00
Minimum Mining Depth - Metres	27.00
Average GRAMT	518.23
Average R-Factor	0.77
Waste-to-Ore Ratio	0.46
Synthetic Crude Yield at 90.2% Extraction and 86.8% Upgrading - Billion Barrels	1.08
Basic Mine Life at 120,000 BPCD - Years	24.72

NOTE: The data above are based on geology only, and do not include dilution, mining loss and extra overburden removed at pit walls. For data influenced by mining procedures see Table 10.6-1.

Table 2.7-4

ORE BODY NO. 4 GRAMT $\geq$ 500.00 AND R-FACTOR $\geq$ 0.600

Overburden - Bank Cubic Metres	181,889,000.00
Pe/CW Material - Bank Cubic	136,926,830.00
Top Reject - Bank Cubic Metres	44,953,171.00
Centre Reject - Bank Cubic Metres	65,973,347.00
Pay Zone - Bank Cubic Metres	461,576,648.00
Mean Grade - Weight Percent	12.02
Grade Standard Deviation - Weight Percent	0.66
Corresponding Area - Hectares	989.00
Mean Fines Content - Weight Percent	11.91
Fines Content Standard Deviation - Weight Percent	2.65
Corresponding Area - Hectares	989.00
Ore Body Area - Hectares	989.00
Mean Overburden Thickness - Metres	18.39
Overburden Standard Deviation - Metres	7.10
Corresponding Area - Hectares	989.00
Maximum Overburden Thickness - Metres	33.00
Minimum Overburden Thickness - Metres	4.00
Mean Pe/CW Material Thickness - Metres	13.84
Pe/CW Material Standard Deviation - Metres	7.27
Corresponding Area - Hectares	989.00
Maximum Pe/CW Material Thickness - Metres	29.99
Minimum Pe/CW Material Thickness - Metres	1.00
Mean Top Reject Thickness - Metres	5.02
Top Reject Standard Deviation - Metres	3.85
Corresponding Area - Hectares	896.00
Maximum Top Reject Thickness - Metres	18.03
Minimum Top Reject Thickness - Metres	0.00
Mean Centre Reject Thickness - Metres	8.13
Centre Reject Standard Deviation - Metres	4.91
Corresponding Area - Hectares	811.00
Maximum Centre Reject Thickness - Metres	29.09
Minimum Centre Reject Thickness - Metres	0.00
Mean Pay Zone Thickness - Metres	46.67
Pay Zone Standard Deviation - Metres	4.55
Corresponding Area - Hectares	989.00
Maximum Pay Zone Thickness - Metres	59.00
Minimum Pay Zone Thickness - Metres	29.20
Mean Mining Depth - Metres	71.73
Mining Depth Standard Deviation - Metres	7.02
Corresponding Area - Hectares	989.00
Maximum Mining Depth - Metres	92.00
Minimum Mining Depth - Metres	47.00
Average GRAMT	560.84
Average R-Factor	0.75
Waste-to-Ore Ratio	0.54
Synthetic Crude Yield at 90.2% Extraction and 86.8% Upgrading - Billion Barrels	0.54
Basic Mine Life at 120,000 BPCD - Years	24.78

NOTE: The data above are based on geology only, and do not include dilution, mining loss and extra overburden removed at pit walls. For data influenced by mining procedures see Table 10.6-1.

creasing fines contents. If an ore body were defined using incremental criteria alone, significant oil sands reserves would remain in place and would likely be rendered unrecoverable.

The use of cut-off criteria provides a means of delineation which can serve as the first approximation in defining pit boundaries. On a regional map the factor used would represent a characteristic of the materials vertically underlying a specific point. Parameters established on this basis would be helpful in optimizing the ore bodies. Ore bodies in this study were delineated using primarily R-Factor and GRAMT. The average geologic R-Factors (indicators of mining cost) for Ore Bodies 1A, 1B, 2 and 4 are 0.75, 0.66, 0.77 and 0.75 respectively. The mining R-Factor which reflects the influences of the pit slope, pit perimeter and overburden depths of Ore Bodies No. 1, 2 and 4 are 0.60, 0.69, and 0.69 respectively. For Ore Body No. 2, only the ratio for the dragline mine plan is supplied, the dragline plan having been based on updated geology. The average geologic R-Factor for the three Ore Bodies is 10% higher than the mining R-Factor calculated after mine planning.

The validity of regional indicators in accurately delineating mining areas should be clearly understood by the user before these tools are used for ore body definition and regional planning. For example, use of GRAMT may not necessarily delineate the most desirable ore body for a single bench dragline operation. GRAMT is influenced by oil sand grade as well as pay zone thickness, and an ore body with high GRAMT may include areas of oil sands too deep to be mined by a single bench dragline operation.

When an ore body is delineated using an economic cutoff, ore along the boundaries may be rendered unrecoverable, especially if the mine is completely backfilled. To fairly treat this loss, a method of accounting must be developed that penalizes the value of the delineated ore. Such appraisals can be done effectively only by using a regional development planning approach designed to optimize the total or overall recovery of the Athabasca oil sands deposit.

References for Chapter 2.0

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Appendix to Section 2.0

I. GEOLOGICAL CROSS SECTIONS

The geological cross sections (under separate cover in Volume II) were drawn on a horizontal scale of 1:10,000 and a vertical scale of 1:1,000. The geological interpretation was based upon the following information:

- aerial photographic mosaic complete with 5 m contour interval topography, scale 1:25,000, compiled by Northwest Survey Corporation (Yukon) Ltd. and Surveys and Property Branch, Alberta Transportation, 1976.

- ERCB drill hole data interpreted by Techman/RC.

The geological interpretation was carried out on a straight-line basis. On the geological cross-sections the following abbreviations were used for the geological formations of the study area:

Geological Formation	Abbreviation
Pleistocene Material	Pe
Clearwater Formation	CW
Top Reject Zone	TR
Pay Zone	PZ
Centre Reject Zone	CR
Bottom Reject Zone	BR
Water Sands (Aquifer)	WS
Devonian Limestone	Dn

The drill hole numbers were shown at the bottom of the drill holes: the first two digits indicate the L.S.D. of a Section and the last two digits the Section of a Township.

II. DEFINITIONS OF TECHNICAL TERMS

1. Overburden (OB) - Pleistocene material (Pe), sands and clays of the Clearwater Formation (CW), and low-grade oil sands (top reject) above the uppermost pay zone.

2. Oil Sands Cut-off Grade - 5% bitumen saturation by weight.

3. Pay zone (PZ) - all oil sands zones defined as follows:
   (a) $\geq$ 5% bitumen and $\geq$ 5 feet zone thickness
   (b) < 5% bitumen and < 5 feet zone thickness, provided that condition (a) prevails both above and below this zone.

4. Net Pay Zone Thickness - accumulated thickness of pay zones.

5. Reject - all oil sands zones defined as follows:
   (a) < 5% bitumen and $\geq$ 5 feet zone thickness
   (b) $\geq$ 5% bitumen and < 5 feet zone thickness, provided that condition (a) prevails both above and below this zone.

6. Top Reject (TR) - lean oil sands (<5% bitumen) above the uppermost pay zone.

7. Centre Reject (CR) - lean oil sands (<5% bitumen) 5 feet or more in thickness between two pay zones.

8. Bottom Reject (BR) - lean oil sands (<5% bitumen) below the lowermost pay zone.

9. Mining Depth - total thickness of all ore grade and waste materials from ground surface to the top of the bottom reject zone.

10. GRAMT - an oil sands quality designation of average oil sands grade multiplied by the oil sands thickness, i.e. average bitumen saturation of pay zone in weight % (GRA) x pay zone thickness in meters (MT).

11. Surface Mineability Factor (S.M.F.) - an ERCB oil sands quality relationship calculated from the following equation:

$$S.M.F. = \frac{S}{1 + 0.9\ (Tw/To)}$$

where S = bitumen saturation in weight %

Tw = total waste thickness

To = total plant feed thickness (net pay zone thickness)

The ERCB suggests that S.M.F. relationship may be used to characterize the relative quality of potentially surface mineable oil sands deposits.

Techman/RC Surface Mineability Factor (T.S.M.F.)

$$T.S.M.F. = \frac{S}{1 + 0.9\ \frac{(OB + CR)}{PZ}}$$

12. R-Factor - A Syncrude-developed criterion for oil sands recoverable by mining:

$$\text{R-Factor} = \frac{\text{Volume of Bitumen in Plant Feed (bbls)}}{\text{Volume of Total Material Moved (cubic yards)}}$$

$$= \frac{d \times g \times 9.55}{D}$$

where d = thickness of oil sands

g = oil sands grade in weight %

D = mining depth

$$9.55 = \frac{\text{weight of cubic yard of oil sands}}{\text{weight of one barrel of bitumen}} = \frac{1.5309 \text{ tonnes}}{0.1603 \text{ tonnes}}$$

<u>R-Factor quality scale</u>*:

Above 1.0 - choice mineability
0.75 - good mineability
0.50 - fair mineability
0.25 - marginal mineability
Below 0.25 - unattractive mineability

* taken from "Industrial Development Study for Northeast Alberta Regional Plan", Hydrocarb Consultants Ltd., February, 1975.

13. <u>Waste-Ore Ratio</u> -

Waste-Ore Ratio =

$$\frac{\text{Volume Overburden (incl. Top Reject)} + \text{Volume Centre Reject}}{\text{Volume of Pay Zone}}$$

III. GENERAL MATERIAL CHARACTERISTICS

1. In situ Bulk Densities

- Pleistocene material (sands, gravels, silts, tills), Clearwater Formation (shales, mudstones, siltstones, sandstones), Oil Sands*: 125 lbs./b.c.f.

  125 lbs./b.c.f. = 3,375 lbs./b.c.y. - 1.6875 short tons/b.c.y.
  = 1.5309 tonnes/b.c.y. = 2.0023 tonnes/b.m^3

* Application to the Alberta Oil and Gas Conservation Board by Atlantic Richfield Company, Cities Service Athabasca, Inc., Imperial Oil Company, Limited, and Royalite Oil Company, Limited, May 1968.

- Crude Bitumen: 1,008 kg/b.m^3

  The density of crude bitumen is variable with typical values in the range of 986 kg/b.m^3 (12° A.P.I.) to 1,030 kg/b.m^3 (6° A.P.I.). The mean density of 1,008 kg/b.m^3 is generally accepted by the oil sands industry:

  1 barrel = 42 U.S. gallons = 5.6146 b.c.f. = 0.1590 b.m^3
  = 160.272 kg = 353.3 lbs. = 0.1767 short tons = 0.1603 tonnes

- Synthetic Crude Oil: 300 lbs./bbl. at 33.6° A.P.I.***

  1 barrel = 42 U.S. gallons = 5.6146 b.c.f. = 0.1590 b.m^3
  = 136 kg = 300 lbs. = 0.150 short tons = 0.1361 tonnes

*** Techman/RC

2. Swell and Load Factor****

	Swell	Load Factor
- Overburden (mostly Pleistocene)	20%	0.83
- Oil Sands in stockpiles, centre reject	25%	0.80
- Oil Sands on conveyors, in buckets	39%	0.72

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3. Bulk Densities of Excavated Materials

- Overburden: 103.75 lbs./l.c.f.

  103.75 lbs./l.c.f. = 2,801 lbs./l.c.y. = 1,400.5 short tons/l.c.y.
  = 1,270.5 tonnes/l.c.y. = 1.6618 tonnes/l.m^3

- Stockpiled Oil Sands: 100 lbs./l.c.f.

  100 lbs./l.c.f. - 2,700 lbs./l.c.y. = 1,350 short tons/l.c.y.
  = 1,225 tonnes/l.c.y. = 1.6022 tonnes/l.m^3

- Conveyed Oil Sands: 90 lbs./l.c.f.

  90 lbs./l.c.f. = 2,430 lbs./l.c.y. = 1.215 short tons/l.c.y.
  = 1.102 tonnes/l.c.y. = 1.4413 tonnes/l.m^3

4. Composition of In situ Oil Sands

The oil sands of the Athabasca region of Alberta have been described as a four-phase hydrocarbon solid composed of crude bitumen, water, solids (mineral component) and gas (usually methane). A typical weight percent analysis of in situ sands would be as follows:

bitumen:	12%
water:	4%
solids:	84%

The fines are defined as the solid material in the oil sands which will pass a 325 mesh sieve (-44 microns). Syncrude Canada Limited defines the "fines content" as the weight percent of -325 mesh material in the mineral component. For the purposes of this study the relationship between fines content and bitumen saturation for in situ oil sands is as follows:

$$\text{fines content} = (-4 \times \text{oil sands grade}) + 60\%$$

Example: for oil sands with an 11% grade, the fines content should be: $(-4 \times 11\%) + 60\% = 16\%$

IV. SAMPLE OF SIMPLIFIED RESERVE CALCULATION (BASED ON ORE BODY NO. 2)

For calculation purposes, the abbreviations OB, CR and PZ designate total formation thickness.

1. Area: 19,667,803 m^2 = 1967 Hectares

2. Drill Hole Density: $\frac{\text{1967 Hectares}}{\text{29 Holes}}$ = 68 Hectares/hole

3. Total Waste Thickness: OB + CR = 11.26 + 8.95 = 20.21 m

4. Mining Depth (MD): OB + PZ + CR = 11.26 + 46.87 + 8.95 = 67.08 m

5. Fines Content: (-4 x Grade) + 60% =
   (-4 x 11.99%) + 60% = 12.04%

6. Waste - Ore Ratio: $\frac{OB + CR}{PZ} = \frac{11.26\text{ m} + 8.95\text{ m}}{46.87\text{ m}} = 0.43$

7. GRAMT: Grade x PZ = 11.99 x 46.87 = 561.97

8. R-Factor: $\frac{PZ \times Grade \times 9.55}{MD} = \frac{46.87 \times 0.1199 \times 9.55}{67.08}$ = 0.80 bbls./b.c.y.

9. T.S.M.F.: $\frac{Grade}{1 + 0.9 \frac{(OB + CR)}{PZ}} = \frac{11.99}{1 + 0.9 \times 0.43} = \frac{11.99}{1.39} = 8.64$

10. Overburden Volume: Area x OB =
    19,667,803 m^2 x 11.26 m = 221.5 x 10^6 bm^3

11. +5% Oil Sands Volume: Area x PZ =
    19,667,803 m^2 x 46.87 m = 921.8 x 10^6 bm^3

12. Oil Sands Weight: Volume x 2.0023 tonnes/m^3

    921.8×10^6 bm^3 x 2.0023 = $1,845.8 \times 10^6$ tonnes

13. Center Reject Volume: Area x CR =

    19,667,803 m^2 x 8.95 m = 176.0×10^6 bm^3

14. In situ Bitumen Reserve: Oil Sand Weight x Grade

    1845.8×10^6 tonnes x 11.99%

    = 221.3×10^6 tonnes

15. Orebody Life at 60,000 BPCD

    Assumptions:

    - Mining Recovery = 100%, i.e. no mining losses or dilution
    - Extraction Plant Recovery = 90.2%
    - Synthetic crude/Bitumen ratio = 0.868
    - Plant Production = 100% capacity

$$\frac{221.3 \times 10^6 \text{ tonnes} \times 0.902 \times 0.868}{60{,}000 \text{ BPCD} \times 0.1603 \text{ Tonnes/bbl.} \times 365 \text{ c. days/year}} = 49.4 \text{ years}$$

16. Ore Body Life at 120,000 BPCD = $\frac{49.35}{2}$ = 24.7 years

17. Ore Body Life at 240,000 BPCD = $\frac{24.68}{2}$ = 12.3 years

3.0 REVIEW OF INFORMATION CONCERNING SELECTED MINING AREA

3.1 GEOMORPHOLOGY[1,2]

3.1.1 GLACIAL HISTORY

The sequence of glacial events which has occurred within the Athabasca oil sands region is represented (Table 3.1.1-1) by a series of glacial advances and retreats resulting in the deposition of the various stratigraphic units. The glaciers advanced and retreated an unknown number of times to deposit the undifferentiated till and stratified sediments which precede the deposition of an unnamed till. Subsequent glacial advances, probably from the east or east-north-east, deposited the unnamed till. A further advance from the northeast deposited the Firebag till and left the large kames present in the study area. The Fort Hills till was deposited by yet another subsequent advance of ice from the north. As the ice in the Clearwater River system melted, the glacial lakes formed by ice blockage receded and extensive outwash deposits were built up by streams flowing southward from the Fort Hills ice sheet over the glaciolacustrine sediments. A series of meltwater channels draining northward formed along the Athabasca Valley. Recent deposition and erosion following deglaciation resulted in further modification to the land surface.

3.1.2 GLACIAL STRATIGRAPHY

The stratigraphic units within the region consist of preglacial gravels and sands, tills, stratified deposits and recent sediments (see Table 3.1.1-1)

Little evidence of preglacial sediments is found in the study area. The presence of sand and gravel cappings on Muskeg Mountain is possible but uncertain due to lack of reliable drilling. The existence of undifferentiated till and stratified sediments is also uncertain; however, they are inferred to be present in areas of thick glacial drift on the slopes of Muskeg Mountain.

An unnamed till has been identified in surface exposures and drill holes to the south and west of the study area. The till is a silty to sandy loam and the coarse sand fraction is high in crystalline grains and low in carbonates. The till is dark grey when wet and unoxidized, and contains a moderate percentage of clasts.

The Firebag till outcrops extensively along the Firebag River to the northeast of the study area. It forms the surface till over most of the area south of the Fort Hills. The Firebag till is overlain by stratified sediments interpreted to be younger than the Fort Hills till; their texture is loam to sandy loam. The coarse sand fraction is relatively low in crystalline grains, high in quartz and contains appreciable carbonate grains, primarily dolostone. The till is dark grey to dark grey-brown when wet and unoxidized, and contains a relatively high percentage of clasts. It is commonly sandier when it directly overlies the McMurray Formation and becomes quite bituminous near the contact with oil sands.

The presence of lower stratified sediments between the Firebag and Fort Hills tills has not been clearly demonstrated. The kames east of the Fort Hills were deposited by the Firebag Glacier.

The Fort Hills till occurs to the north of the study area and may also occur south of the Fort Hills. This till is absent at higher elevations indicating that the suggested re-advance of the Fort Hills Glacier may have only been local. The till is dark grey or pinkish grey when wet and unoxidized, and commonly contains a relatively low percentage of clasts. Thin lenses of lacustrine silts and clays or glaciofluvial sands are common in the unit. The coarse-sand fraction of this till is lower in quartz grains and higher in carbonate grains than older tills; it is commonly less crystalline than the unnamed till.

TABLE 3.1.1-1

GLACIAL STRATIGRAPHY IN VICINITY OF STUDY AREA

ERA	PERIOD	DESCRIPTION OF UNITS	
CENOZOIC	Quaternary	Recent sediments:	includes lacustrine, alluvial and aeolian deposits and muskeg.
		Meltwater channel sediments	possibly early Athabasca River alluvium.
		Upper stratified sediments	outwash deposits glaciolacustrine deposits
			Fort Hills kame moraine deposits
		Fort Hills till	mixed glaciolacustrine deposits
		Lower stratified sediments (?)	glaciolacustrine deposits glaciofluvial deposits
		Firebag till	
		Unnamed till	
		Undifferentiated till and stratified sediments	
	?		
	Tertiary	Saskatchewan gravels and sands	primarily quartzite with minor chert, ironstone, coal
		Preglacial gravels and sands	fragments and clay lumps
MESOZOIC	Cretaceous	Sandstones, siltstones and shales.	

After McPherson and Kathol[1]

13-6-78R

Upper stratified sediments are associated with the melting of the Fort Hills ice sheet and include the Fort Hills kame moraine complex to the north east of the study area, mixed glaciolacustrine sediments and outwash sand. The kame moraine marks an ice frontal position of the Fort Hills ice advance. The deposits comprise stratified silts and fine-grained silty sands. Rock fragments coarser than sand are present but uncommon. Lenses and layers of pink-grey glaciolacustrine sediments are present below, within and above the kame. The matrix of the mixed glaciolacustrine sediments ranges from a heavy clay to a loam sand. The mixed sediments contain a high percentage of till-like materials and frequently show well-developed stratification. The outwash deposits which consist primarily of fine- to medium-grained sand overlie the glaciolacustrine sediments over much of the region. Sand and gravel occur within the outwash deposits as discontinuous bars and channels. Lenses of pink-grey glaciolacustrine sediments and till fragments are also found in the outwash. Between the Fort Hills and the Firebag River, the outwash exceeds 45 m in thickness.

Meltwater channel sediments typically occur along the sides of the Athabasca River valley below an elevation of 312 m.

3.1.3 SURFICIAL GEOLOGY

The study area forms part of a dissected highland or tableland underlain by nearly flat-lying shales, sandstones and limestones. Bedrock exposures are scarce due to the widespread cover of glacial and postglacial deposits. The glacial drift is generally of low relief, except where kame deposits occur.

The surficial materials within the study area are of glacial, glaciofluvial, glaciolacustrine, aeolian, lacustrine, alluvial, and organic origin. The location and areal extent of these deposits within the study area is illustrated on Drawing No. F22910-04-00, 'Surficial Geology, Hydrology and Topography, Ore Bodies No. 1 and 2' and Drawing No. D22910-05-00, 'Surficial Geology, Hydrology and Topography, Ore Body No. 4'.

a. Glacial Deposits:

Glacial deposits, in the form of till, are exposed predominantly on the flanks of major bedrock uplands, but are known to underlie most of the area. The tills consist of a heterogeneous mixture of silt, clay, sand, gravel and boulders; however, the till matrix (< 2 mm) is generally loam to sandy loam. The surface till has a local relief of less than 6 m. In the study area, the till forms a level to gently undulating plain on the flanks of the Muskeg Mountain Upland, and is characterized by small knobs and kettles with a few small till ridges, aligned knobs and fluting. Thickness of surficial materials on the slopes of Muskeg Mountain commonly ranges from 15 to 45 m, with a maximum depth of 140 m. There is evidence that the surface of the till has been modified by a glacial lake (or lakes).

b. Glaciofluvial Deposits:

Glaciofluvial deposits include sediments deposited by glacial meltwater streams. They are present in the study area as kames or kame moraines, and outwash sands.

- Kames and Kame Moraines: The kames and kame moraines are mound-like hills of stratified ice-contact drift. The three kame deposits within or immediately adjacent to the study area are: a kame moraine ridge situated along the northern lobe of Ore Body No. 2; a kame complex, which is partially eroded, to the south and west of Ore Body No. 4; and a major kame complex (the Fort Hills) to the northwest of the study area. The distribution of material comprising these kame deposits is not well known but appears to consist primarily of sand, silt and till. The tills associated with these deposits are commonly sandy. Lenses and layers of clay and gravel can also be expected in these glaciofluvial deposits. Commonly, a layer of sand and silt overlies till layers of variable thickness and extent. The till layers are often separated by layers of sand. It is uncertain whether the kame sediments extend only to the uppermost till or whether much of the till occurs as lenses and pockets within the kames.

The kame moraine near Kearl Lake covers an area of approximately 6.4 km^2 and has an elevation of up to 350 m or 20 m above adjacent terrain. The moraine once marked the shoreline of a glacial lake as evidenced by beaches developed along the northwest side of the ridge. The kame moraine comprises variable amounts of sand and till.

The Fort Hills kame deposit northwest of Ore Body No. 4 is up to 70 m thick and the top of kame moraine near Kearl Lake is up to 120 m thick.

- Outwash Sand: Outwash sand is the most common surface sediment in the study area and is deposited in the vicinity of the proposed tailings pond area for Ore Body No. 2, across the southern lobe of Ore Body No. 2 and north of Ore Body No. 4. The original depositional surface was flat with local relief generally less than 4.6 m resulting from bars, discontinuous terraces, complex "braided stream" patterns, possible buried ice blocks (which subsequently melted) and occasional sinkholes. The depositional surface has been further modified by the presence of discontinuous aeolian and muskeg deposits, and postglacial erosion. The outwash sand is generally very fine- to fine-grained, with coarser sand, gravel, till fragments, silt and clay being present in minor amounts. The sand also commonly contains re-worked bitumen. The thickness of the outwash sand is quite variable, but in this region ranges between 6 and 9 m.

c. Glaciolacustrine Deposits:

The glaciolacustrine deposits, which are fairly extensive in the study area, are classified as mixed deposits. The lithology of these deposits is quite variable, but they generally consist of stratified sand, silt, clay, and till or till-like material. These deposits are situated near Kearl Lake (overlying portion of Ore Bodies No. 1 and 2), and on the north-west flanks of Muskeg Mountain (extending partially across the southeast portion of Ore Body No. 1). The local relief is flat to gently undulating (generally less than 3 m or 10 ft.). The original surface of deposition is masked in some areas by a discontinuous cover of muskeg or recent lacustrine or aeolian deposits. The individual layers pinch out both horizontally and vertically over relatively short distances. Slump and flow structures are also common.

- Glacial Shorelines: Shoreline features representing remnants of glacial lakeshores are present on the flanks of Muskeg Mountain and the kame northeast of Kearl Lake. The beach ridges are commonly 30 m wide and range in length from 100 m to over 16 km. These features consist mainly of sand.

d. Recent Deposits and Features:

Recent deposits and features, formed subsequent to deglaciation of the region, are common in the study area. The deposits have formed as a result of subaerial geologic and climatic processes and are classified as aeolian, lacustrine and organic deposits. Erosional features are predominantly colluvial or alluvial in origin.

- Aeolian Deposits: Aeolian deposits occur where glaciofluvial sand constitutes the main surface material. These deposits form a discontinuous cover in dune and/or sheet form. Within the study area, they are most prevalent to the northeast of Ore Body No. 2 (in the vicinity of the proposed tailing pond). The deposit consists of well-sorted, fine- to medium-grained sand (mainly quartz).

- Lacustrine Deposits: Within the study area, recent lacustrine deposits are most prevalent around Kearl Lake, although smaller local depressions contain minor quantities of lacustrine sands, silts, and clays. The lacustrine deposits have a high organic content, and may contain muck and marl.

- Alluvial Deposits: Alluvial deposits within the study area consist of silt and clay deposited along recent streams to depths of less than 3 m. Minor lenses of sand and gravel within the silt and clay are common. Within the study area, these deposits are present in the lower reaches of the minor drainages originating on the flanks of Muskeg Mountain, and in the stream channels of the Muskeg River, the Firebag River, and Hartley Creek.

- Organic Deposits: Organic deposits or bogs (muskeg) are widespread and cover at least half the study area. Muskeg forms a fairly continuous cover in areas east and south of Kearl Lake (partially covering Ore Body No. 1), and in a wide band adjacent to the Muskeg River. Smaller areas of continuous muskeg occur adjacent to Hartley Creek, near the southern limit of Ore Body No. 2, and within and adjacent to Ore Body No. 4. Discontinuous but extensive muskeg areas overlying outwash sand and glaciolacustrine sediments are present throughout the study area. The muskegs are generally less than 3 m thick, and may extend to a depth of 9 m. The areal extent of a muskeg deposit does not necessarily reflect the depth of the deposit. Muskeg has been observed to be frozen at depth in late summer.

- Erosional Features: Eroded slopes, gullies, and stream valleys are present on the flanks of Muskeg Mountain. Slope materials include surficial or bedrock deposits overlain by a discontinuous cover of colluvium.

3.2 REGIONAL GEOLOGY[1,2,3,4]

The lowest stratigraphic sequence within the study area consists of metasedimentary and granitic rocks of Precambrian age (Figure 3.2-1). Approximately 150 m of Devonian carbonates and evaporites overlie the Precambrian basement. The Cretaceous oil-bearing McMurray Formation unconformably overlies the Devonian sequence. At the surface, a thin layer of glacial and younger materials overlies the Cretaceous rocks.

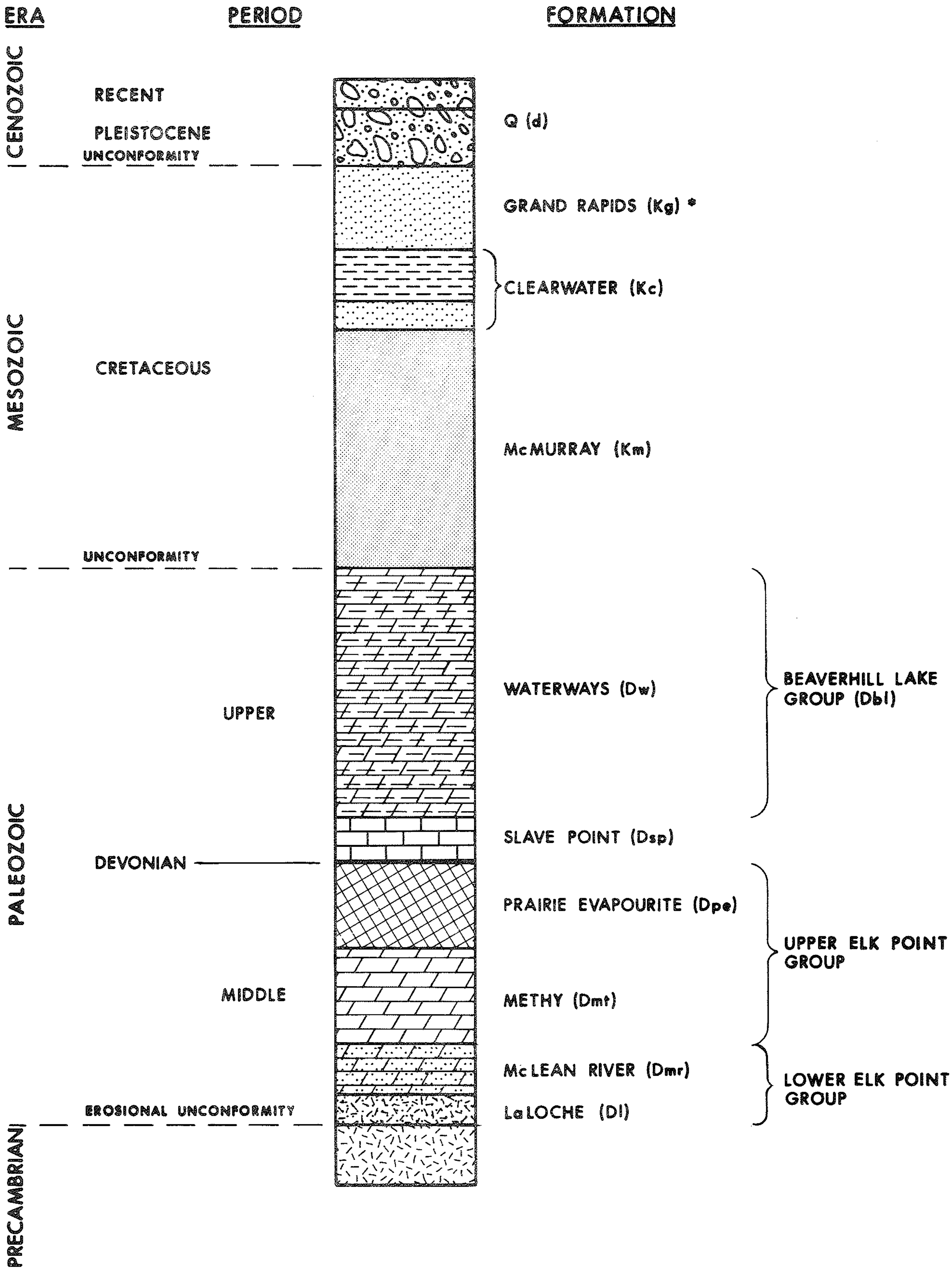
Oil sands of economic value are contained within the Cretaceous McMurray Formation. In the Athabasca region north of Fort McMurray, the McMurray Formation occurs very close to the surface, and over much of the study area subcrops directly below the Pleistocene surficial materials. The proximity of the oil sands to the ground surface makes these deposits amenable to extraction by mining methods. However, only a small fraction of the total oil sands reserves are surface mineable.

3.2.1 PRECAMBRIAN

The Precambrian rocks, consisting of granitic plutonic rocks with some granite gneiss and metasedimentary rocks, form the impermeable basement within the study area. The Precambrian surface has considerable local relief, and a southwesterly regional slope of approximately 5.7 m per km, as shown in Figure 3.2.1-1.

3.2.2 PALEOZOIC

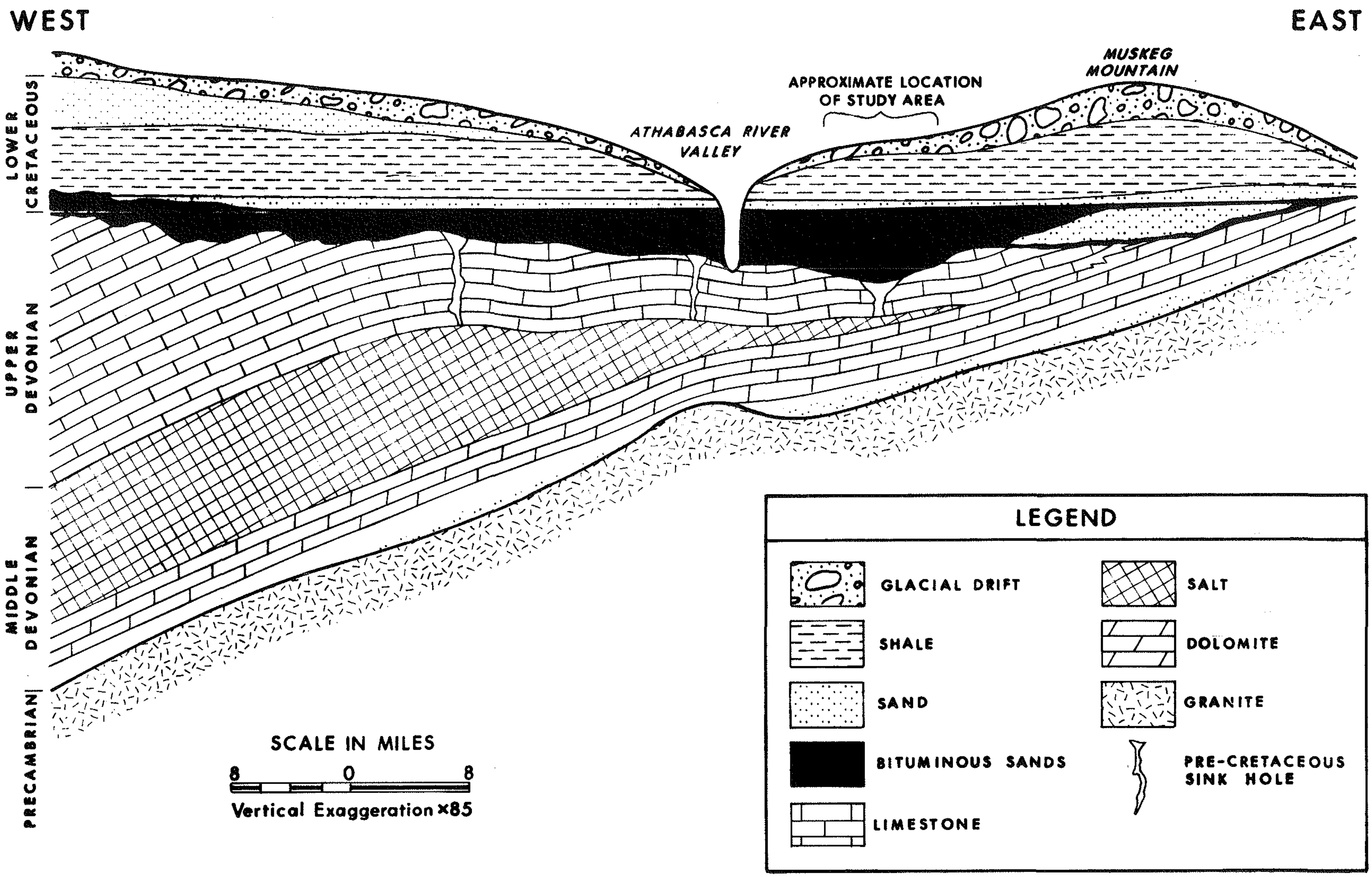
The Paleozoic stratigraphic section is represented by a carbonate-shale-evaporite sequence of Middle and Upper Devonian age. The Paleozoic rocks form a southwest-dipping monoclinal structure with a north-northwest strike. Superimposed upon the monoclinal structure are local anticlines, synclines, terraces, and some faulting related to basement features or to later events.



* GRAND RAPIDS FORMATION IS NOT PRESENT WITHIN THE LIMITS OF THE ORE BODIES

GENERALIZED STRATIGRAPHIC COLUMN FOR STUDY AREA

FIGURE 3.2-1



SIMPLIFIED GEOLOGICAL CROSS SECTION THROUGH THE ATHABASCA OIL SANDS DEPOSIT

After Carrigy[3]

FIGURE 3.2.1 - 1

Middle Devonian

The Middle Devonian rocks within the study area are contained within the Elk Point Group, consisting of the LaLoche, McLean River, Methy, and Prairie Evaporite Formations.

The LaLoche Formation is equivalent to a "granite wash" and consists mainly of feldspathic and gritty sandstones which unconformably overlie the Precambrian surface. Its thickness varies from zero to 40 m.

The McLean River Formation is predominantly a dolomitic sandstone with interbeds of silty and sandy shale, mudstone, and thin beds of anhydrite and gypsum. Its thickness varies from 18 to 49 m.

The Methy Formation is composed mainly of dolomites and exhibits biothermal reef buildups to as much as 82 m in thickness.

The Prairie Evaporite Formation is predominantly halite, but significant amounts of anhydrite, gypsum, and silty shale are also present. The formation varies in thickness (within the region) from zero in the eastern portion of the Athabasca oil sands to approximately 245 m in the western portion. Formation thinning in the east is a result of both depositional thinning and post-depositional dissolution of the salt by groundwater (mainly from within the underlying Methy Formation). Collapse of overlying units has resulted where dissolution and removal of salts have occurred. Continued subsidence and generation of highly saline groundwater present significant geotechnical problems to potential oil sands operations.

Upper Devonian

The Upper Devonian rock units consist of the Waterways Formation (Beaverhill Lake Group) and the underlying Slave Point Formation.

The Slave Point Formation is relatively thin (less than 15 m) and consists of limestone and dolomitic limestone.

The Waterways Formation, which consists primarily of shale and shaley limestone, is subdivided into five members (in ascending order: the Firebag, Calumet, Christina, Moberly and Mildred Members). Its thickness varies from zero (along its eastern subcrop edge near the Canadian Shield) to 215 m (in the west). This formation underlies the entire study area and forms the bedrock surface along the Athabasca lowland, south of Fort MacKay and to the south and north of the study area. The formation crops out locally along the lower reaches of the Muskeg and Firebag Rivers.

The Firebag Member consists of approximately 52 m of shales and argillaceous limestone which paraconformably overlie the Slave Point Formation and conformably underlie the Calumet Member.

The Calumet Member is about 30 m of resistant, fine-grained and clastic limestone, sharply bounded above and below by shales.

The Christina Member is defined as approximately 27 m of argillaceous limestone and shale lying above the Calumet Member. In one locality in the Athabasca oil sands, the Calumet Member is unconformably overlain by the McMurray Formation.

The Moberly Member is dominantly clastic limestone approximately 60 m thick. Outcrops of this member are numerous along the Athabasca River and along lower reaches of the Muskeg and MacKay Rivers. The member becomes more shaley towards the top, which represents an erosion surface throughout most of the Clearwater-Athabasca River area. It is unconformably overlain by the McMurray Formation, with progressively younger beds overlapped from east to west.

The Mildred Member is about 43 m thick and consists of greenish-grey calcareous shale, argillaceous limestone, and some pale-brown clastic limestone. This member does not outcrop within or adjacent the study area.

The Waterways (Devonian) surface exhibits classic karst topography with blind valleys, internal drainage sinks, pinnacles and sharp ridges. The complex configuration of the Waterways surface had a strong influence on the deposition of the overlying McMurray Formation. The lower contact of the oil sands is highly irregular, and requires extensive drilling to closely define the oil sands reserves.

3.2.3 MESOZOIC

The Mesozoic rocks in the study area are represented by the McMurray and Clearwater Formations (and Grand Rapids Formation outside of ore body limits) of Cretaceous age which unconformably overlie the Waterways Formation. The profound influence of the pre-Cretaceous topography on Mesozoic sedimentation is lessened, since the landscape was buried, and the influence is not readily noticeable in the strata above the overlying Middle Clearwater Formation.

The McMurray Formation, consisting of bitumen-impregnated sand, subcrops under all of the study area except the lower slopes of Muskeg Mountain to the southeast. The McMurray Formation has been divided into four informal stratigraphic units: pre-McMurray beds, lower, middle, and upper units. The McMurray Formation is believed by some authorities to be of deltaic origin with beds of the middle unit being primarily foreset beds and beds of the upper unit large topset beds.

The pre-McMurray beds represent remnants of a coarse-grained quartzose sandstone, cemented by silica and goethite which appear to unconformably underlie the McMurray Formation.

The lower unit of the McMurray Formation is comprised of lenticular beds of trough cross-stratified conglomerates, sand, shale, and silt which occupy the deeper depressions on the pre-Cretaceous erosion surface. Basal strata consist of residual clays derived from weathering of the Waterways limestone.

The clays (mineral portion of these strata) consist of illite, kaolinite, vermiculite, and mixed-layer clays. The clays, which have been noted in thicknesses up to 30 m, are overlain by conglomerate, sand, and some shale and silt. The coarse-grained sands often contain large wood fragments, well-rounded quartz grains, numerous feldspar cleavage fragments and minor amounts of mica. In some places, the sand contains only fresh water (basal water sands); elsewhere it is impregnated with bitumen (oil sands).

The middle unit of the McMurray Formation generally lies between 229 and 286 m above mean sea level. The unit consists mainly of oil-impregnated, fine-grained, well-sorted quartz sand. Lenticular beds of silts, shales and clays are interbedded within the sands. Cross-stratification of sand beds (decimetres to metres thick) dips at 8 to 12° on the average, but angles approaching 40° have been observed.

The upper unit of the McMurray Formation consists of generally horizontally-bedded fine-grained sands, sandy silts, silts and intraformational silty clay beds. Kaolinite and illite are the primary clay minerals. Large shallow channels or scours filled with silts and siderite-cemented siltstones are also present.

The McMurray surface is quite high in the southeast half of the study area and exceeds elevation 335 m east and southeast of the study area. The surface drops off irregularly to form a trough-shaped low along the Athabasca River to the west. The highs in the McMurray surface are coincident with highs in the underlying Waterways surface. The slopes of the McMurray surface overall are generally quite low, being considerably less steep than most slopes on the underlying Waterways surface.

In general, the overburden (all sediments overlying the McMurray Formation) is thick where the formation is overlain by both younger Cretaceous sediments and surficial deposits. On Muskeg Mountain, southeast of the study area, the overburden, including the Clearwater and Grand Rapids Formations, attains a thickness of up to 200 m. The overburden is generally the thinnest in the vicinity of the ore bodies, where the McMurray Formation subcrops directly below the surficial materials. The overburden is thick where large kame moraines occur, such as immediately west of Kearl Lake between Ore Bodies No. 1 and 2.

The Clearwater Formation, which overlies the McMurray Formation, is dominantly grey to grey-black shale, and commonly contains varying amounts of sand, silt, indurated siltstone, ironstone layers, and gypsum crystals. The Wabiskaw Member, found at the base of the Clearwater Formation, consists of a thin bed of glauconitic sandstone of fine to medium grain size, and contains varying proportions of silt and clay. In some localities, the Wabiskaw Member is oil-bearing. The edge of the Clearwater subcrop trends northeasterly-southwesterly, intersecting the southeastern edge of Ore Body No. 1. Erosional remnants of the Clearwater Formation may be present elsewhere in the study area.

In the study area, the Grand Rapids Formation is present only on the flanks of Muskeg Mountain. The dominant lithology is a "salt and pepper" sandstone; however, siltstone and shale layers are common. This unit is of little or no concern to this study.

3.3 HYDROGEOLOGY[2,4,5,6,7,8]

The following section of the report presents a hydrogeologic model of the regional groundwater system within the Athabasca Oil Sands region. Each stratigraphic unit from the Precambrian up to and including the surficial deposits is discussed relative to its hydrogeological significance, with special reference to the study area. Detailed geological descriptions of the stratigraphic units have been presented in Section 3.2. The subject of mine dewatering as it affects the development and subsequent reclamation of an oil sands mine is discussed in Section 5.3, Tailings Disposal Techniques.

3.3.1 PRECAMBRIAN

The Precambrian rocks are assumed to form an impermeable basement throughout the study area, and are not of immediate concern to this study.

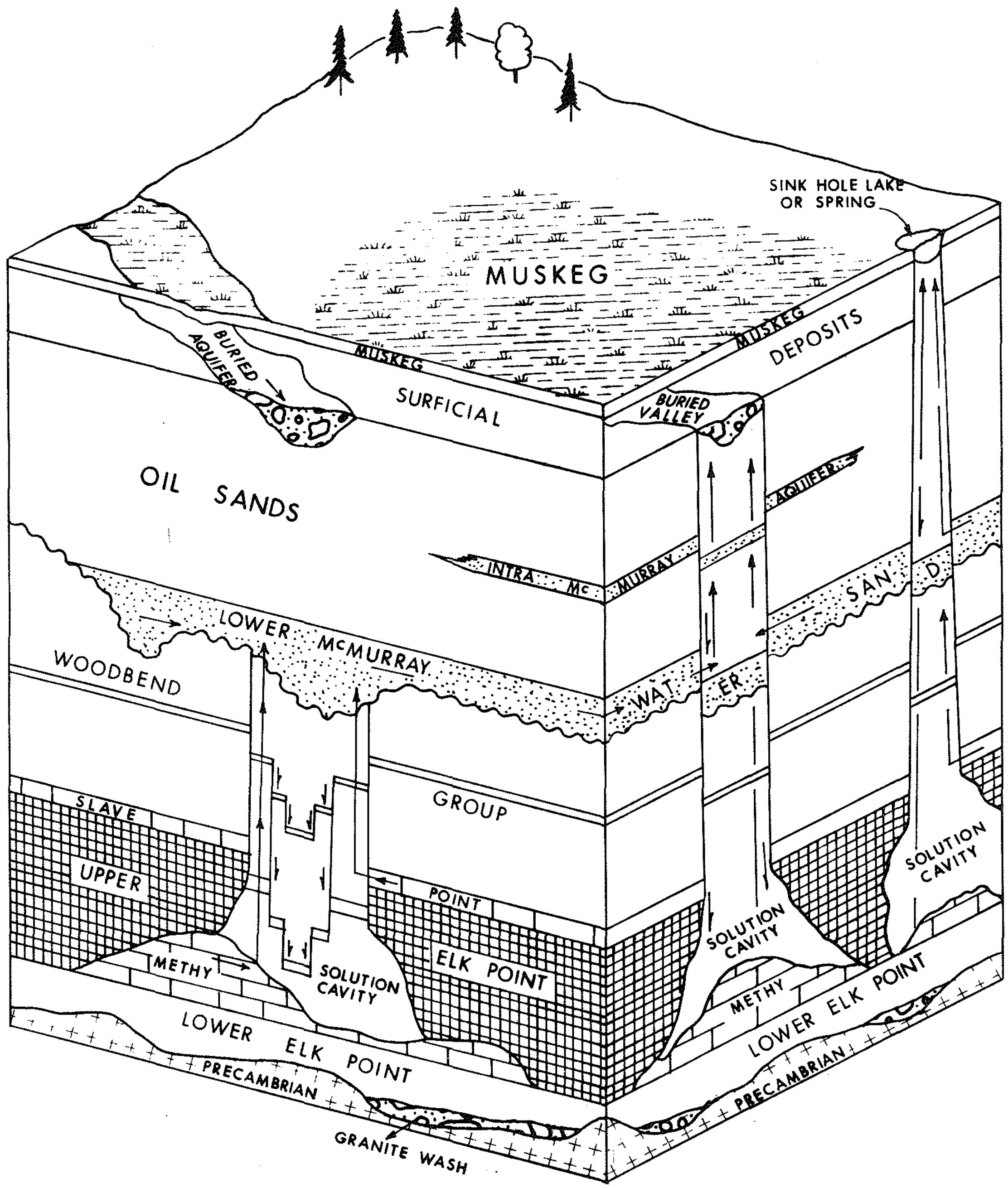
3.3.2 PALEOZOIC

The Paleozoic rocks are generally divided into two hydrostratigraphic units: the Middle Devonian represented by the Prairie Evaporite, Methy, McLean River and LaLoche Formations (as shown in Figure 3.3.2-1); and the Upper Devonian, which includes the Beaverhill Lake Group (Waterways Formation) and the Slave Point Formation.

Middle Devonian Hydrostratigraphic Unit

The LaLoche Formation is known to be permeable and porous, and contains salt in at least one location in the Athabasca Oil Sands. The formation is considered to be an aquifer of sporadic distribution and is not of direct concern to this study since it is located far below any zone of influence important to surface mining operations.

The McLean River Formation is relatively impermeable and acts as a cap rock or confining bed for any groundwater movement in the underlying LaLoche Formation.



SKETCH SHOWING POSSIBLE EFFECTS OF SALT SOLUTION ON GROUNDWATER SYSTEMS

(not to scale)

After Gorrell[4]

FIGURE 3.3.2-1

The Methy Formation is very porous and permeable where reefal build-ups are present. The formation waters are usually highly saline and under high hydraulic head. The reefal facies appear to be more prevalent toward the eastern portion of the Athabasca Oil Sands deposit (i.e. the vicinity of the study area). Although not likely, there is a possibility that the highly saline waters could flow upward into higher porous units and eventually reach surface oil-sands mining operations (Figure 3.3.2-1). Limited deep-well waste disposal might be considered in this unit because of its high porosity and permeability. However, the likelihood of a hydrogeological connection with overlying units should be determined in advance. It is possible that the Methy Formation subcrops beneath beds of the Lower McMurray water sands in the study area, or possibly on the slopes of Muskeg Mountain. Farther east, the Methy Formation may subcrop directly below the surficial glacial deposits. To the east of the study area, the Methy Formation may be recharged directly from the surface. Information in the vicinity of the study area (east of the Athabasca River) indicates that piezometric heads in the McMurray Formation are only 20 m higher than those in the Methy Formation. Dewatering of the McMurray Formation and subsequent surface mine development would reduce McMurray heads below those in the Methy Formation. Thus stresses set up by the removal of water in the overlying McMurray Formation could result in floor heave in the pit bottom and may cause a subsequent breakthrough. The thinning of the protective overburden over the high-pressure reefal Methy Formation in the study area further increases the likelihood that uplift pressures from groundwater in the Methy reefs could be strong enough to cause development of fractures and subsequent release of saline water into mine openings and hence to the surface.

The Prairie Evaporite Formation is significant to the regional movement of groundwater, as major collapse structures resulting from dissolution of salt beds have affected the stratigraphy and structure of the overlying beds. Salt dissolution within the Prairie Evaporite Formation is believed to have commenced during Devonian times and continued to the present day. Within the study area, the Prairie

Evaporite Formation has thinned partly as a result of salt dissolution, but also because of a depositional thinning in the eastern portion of the Athabasca oil sands region. The salt beds of the Prairie Evaporite Formation surround the porous and permeable dolomites of the Methy Formation which contain water under high pressure; when undisturbed the salt formations act as confining beds. In cases where the salt has been removed by dissolution, disturbance of the overlying beds could result in a hydraulic connection between upper and lower aquifers, allowing release of highly saline water upwards from the Methy Formation.

The hydrogeological regime of the Middle Devonian hydrostatic unit is different on the west and east sides of the Athabasca River. To the west, this unit is characterized by confined aquifers, hydrostatic heads at or above ground level, and total dissolved solids concentrations of 200,000 to 300,000 mg/l. In the study area east of the Athabasca River, the absence of a thick Prairie Evaporite Formation results in a different hydrogeological regime. Here the conditions are characterized by much lower expected dissolved solids concentrations (50,000 to 100,000 mg/l) and westward lateral flows. In the study area, flows within these hydrostatic units are expected to be small because of the low hydraulic conductivity (10^{-5} to 10^{-8} cm/sec); hydraulic heads are just above 245 m elevation.

<u>Upper Devonian Hydrostratigraphic Unit</u>

The Upper Devonian hydrostratigraphic unit consists of all Devonian strata above the evaporite deposits. The Slave Point Formation is the lowest stratigraphic unit of the Upper Devonian. It is not generally porous or permeable in the vicinity of the study area, but thin porous water-bearing beds have been noted west of the Athabasca River. This formation could possess a high porosity and permeability, and it may contain a significant aquifer under artesian pressures. Where the Slave Point Formation is not permeable or disturbed, it forms part of the protective cover over the Methy Formation. Correlation of this unit in the study area is difficult.

The Beaverhill Lake Group (Waterways Formation) which overlies the Slave Point Formation, is considered to represent one hydrostratigraphic unit, as the lithologic characteristics of the members are similar and all members are generally low in permeability (hydraulic conductivities of 10^{-5} to 10^{-7} cm/sec). The thickness of this unit approaches zero to the east of the study area. The Beaverhill Lake Group forms part of the protective cover that confines the waters of the Methy Formation, where salt dissolution has not affected its permeability.

The Upper Devonian hydrostratigraphic unit is characterized by dominantly horizontal direction of flow, an impermeable basement formed by the evaporite rocks (more prevalent west of the Athabasca River), and low hydraulic heads along the Sewatakun Fault (which intersects the Athabasca River at Fort McMurray and Fort MacKay and parallels the river 10 km west of Tar Island). The rocks in this unit act as a low potential drain under most of Alberta because of widespread and highly permeable reef complexes. The unit is not reef-bearing in the study area, however. The relationship of primary hydraulic conductivity to topography is not adequate to account for the extensive lowering of fluid potentials in this unit. It is hypothesized that the low heads are caused by a major fault (Sewatakun Fault), and by collapsed areas resulting from dissolution of salt. Groundwater flows upward along this fault, enhancing the dissolution of salt and the subsequent collapse of the Beaverhill Lake Group. Collapse increases the macroscopic permeability of the overlying units in the vicinity of the Athabasca River. This increased overall permeability has contributed to the spread of low fluid-potential zones in the Upper Devonian hydrostratigraphic unit. The hydraulic head of the Upper Devonian hydrostratigraphic unit in the vicinity of the three ore bodies is near 290 m elevation.

3.3.3 MESOZOIC AND CENOZOIC

The Mesozoic and Cenozoic rocks in the study area are of Cretaceous, Pleistocene and Recent ages.

Cretaceous Hydrostratigraphic Unit

The Cretaceous hydrostratigraphic unit consists of the unconsolidated surficial deposits (Pleistocene and Recent), and all Cretaceous rock units including the McMurray, Clearwater, and Grand Rapids Formations.

The McMurray Formation contains at least two types of aquifers: the lower basal water sands, and intra-ore body aquifers. Both are of hydrogeological significance to the study.

The lower basal water sands (assorted layers of sands, silts, and clays) have transmissibilities of 87,000 to 225,000 Lpd/m in the Athabasca Oil Sands area. Holes to the immediate north of Ore Body No. 4 indicate the Lower McMurray water sand is separated from an overlying water-bearing bed by a thick shale-siltstone zone. Pumping tests indicate that the water sands are confined, and that there is a lack of vertical hydraulic continuity in this area. In some local areas, lateral continuity is good, but this is variable. Continuity is strongly affected by the configuration of the Devonian surface and the effects of collapse features. Thick sections of the Lower McMurray basal water sands probably have good lateral continuity. In areas where underlying salt dissolution has caused collapse of overlying units, direct vertical communication with highly saline Devonian waters is possible.

The regional structure of the Paleozoic (Devonian) rocks is of a southwesterly-dipping monocline with a north-northwest strike. During the hiatus between deposition of the Devonian rocks and the overlying Cretaceous strata, the Devonian rocks were probably subjected to several periods of subaerial erosion which resulted in a complex configuration of its surface. Pre-Cretaceous tilting of the Devonian rocks is believed to have resulted in a regional slope of 2.8 m per km. Post-Cretaceous tilting resulted in an additional 1.5 to 2.1 m per km of dip of the Cretaceous rocks. Subsidence of the entire area has resulted from gradual dissolution of the underlying salt beds (Prairie Evaporite Formation). The resultant Devonian surface has considerable relief with some slopes as steep as 68 m per km. Many of the ridges trend north-northwest and correspond to the strike of the erosion-resistant carbonates.

The Devonian (Waterways) surface is high to the northeast, east and southeast of the study area with elevations generally exceeding 275 to 290 m above mean sea level. From these topographic highs the surface drops off in an irregular trough shape into the Athabasca River valley where elevations are generally less than 236 m above mean sea level. The irregular Waterways surface is characterized by numerous elongated ridges and channels which trend in various directions, and often end abruptly and irregularly in circular topographic highs and lows of various sizes. These features could be a result of subaerial erosion or of structural flexures caused by subsidence. A conspicuous broad topographic low in the Devonian surface, encompassing approximately 50 km^2, occurs below the southwest portion of the Fort Hills adjacent to the study area. Numerous channels trend into this low, suggesting that an internal drainage system existed at one time. One of the best-developed channels trends northeast along the Muskeg River and then swings north to enter the broad low under the Fort Hills. As well, there are many smaller, irregularly-shaped basins that appear to have developed internal drainage systems.

This detailed knowledge of the Devonian surface configuration is necessary for an understanding of groundwater movements. Individual sink holes could be acting as vertical conduits for groundwater (see Figure 3.3.2-1). Many low features in the Devonian surface are coincident with thick zones of basal water sands. However, lows developed on the Devonian surface during post-McMurray time (a result of salt dissolution in the underlying Prairie Evaporite Formation) may not show the same thickening. Water-bearing sands with high hydraulic levels have been noted overlying highs in the Devonian surface. In addition, evidence indicates that the weathered surface of the Devonian may form local aquifers.

Intra-ore body aquifers are generally thin and of limited extent. Flow rates are generally low and total dissolved solids tend to increase with depth to about 12,000 ppm. These aquifers may be under pressure.

The Clearwater Formation is low in permeability and acts as an aquitard. The lower Wabiskaw Member, although permeable, is not an important aquifer.

The Grand Rapids Formation is porous, permeable, and water-bearing, but does not appear to be present within the study area except on the upper slopes of Muskeg Mountain.

Detailed information on aquifers within the glacial drift overlying bedrock is sparse. It is believed that aquifers within the surficial materials may be of considerable importance from a mining standpoint. The study area is covered extensively by relatively coarse-grained glacial deposits including outwash sands and ice-contact deposits. These deposits can be quite permeable and may contain large amounts of water. Interlensing of finer-grained beds may result in locally-perched water tables. Meltwater channels containing sands and gravels are probably the most important source of groundwater in the surficial deposits within the study area. However, the location and extent of these features in the study area are not well known. Organic deposits, in the form of continuous or discontinuous muskeg cover, are prevalent through the study area. These muskegs are supersaturated and may present considerable handling problems during pre-stripping operations.

It is apparent that very little practical information is available concerning the hydrogeology of the study area. Consequently the severity of mine dewatering problems and the environmental impact of disposing of groundwater cannot be estimated reliably.

It is believed that mining of the Athabasca Oil Sands requires a regional rather than site-specific approach in areal dewatering prior to the introduction of mining. Sufficient data, gathered by an appropriate government agency in cooperation with the oil sands industry, could lead to the development of an accurate picture of regional groundwater regimes before, during, and after mining. Current information is spotty and difficult to adapt to the regional scale at which dewatering problems must be addressed.

3.4 METEOROLOGY AND CLIMATOLOGY

3.4.1 STATISTICS CHARACTERIZING CLIMATE[9]

Climatic parameters relevant to the identification of major impacts of oil sands mining, and to the development of reclamation plans are summarized below.

a. January is the coldest month, with a mean daily temperature of -21.3°C; July is the warmest month, with a mean daily temperature of 16.4°C (Fort McMurray Airport).

b. Last spring frost normally occurs between June 1 and 15; first autumn frost usually occurs between September 1 and 15. The average frost-free period is between 80 and 100 days, depending on topographic location.

c. Number of degree days over 5.6°C (approximate minimum for plant growth) ranges between 1800 and 2100.

d. Mean annual precipitation ranges between 400 and 450 mm on the Clearwater Lowland, and between 450 and 500 mm on the Muskeg Mountain Upland and Firebag Plain.

e. Of these totals, 230 and 280 mm falls as rain during the growing season. Between 1300 and 1500 mm of snow falls; upland areas typically receive more snow.

f. Precipitation is normally recorded on 120 to 130 days during the year.

g. Maximum precipitation may be expected in the period July 16 - 31. Precipitation during July ranges between 60 and 75 mm, whereas 50 to 65 mm may be expected in June, and less than 40 mm will probably fall in May.

h. Most thunderstorms are expected during July (nearly 30 hours). June and August normally have about 10 and 15 hours of thunderstorms, respectively.

i. The prediction of a maximum one day rainfall based on a 5 year return is 40 to 50 mm; when based on a 25 year return, this total rises to 65 to 75 mm.

j. Annual climatic variability is $\pm 35\%$. Variability during the growing season is about 30%.

k. The area usually receives between 1800 and 1900 hours of sunshine per year.

l. Average potential evapotranspiration is 450 to 500 mm. Actual evapotranspiration ranges between 400 and 450 mm.

m. Prevailing winds are from the west, with southwesterlies predominating in summer and northwesterlies in the fall and winter.

Four main areas of climatology that influence, or are influenced by, development of mines and associated facilities at the study Ore Bodies include:

- Dispersion of stack emissions from the plant;
- Low-level emissions from various sources;
- Ice fog and low-temperature water fog from condensation of airborne moisture;
- Secondary effects of the interaction of fog and emissions.

3.4.2 DISPERSION OF PLANT EMISSIONS

A number of emissions emanate from the plant including SO_2, nitrogen oxides, H_2S, CO, and particulates. Of these, only SO_2 (which combines to form H_2SO_4) and particulates are of special importance. SO_2 (H_2SO_4) may cause acid rain affecting soils, water systems, and vegetation; particulates contribute to fog density, under certain conditions, by creating nuclei for fog formation.

Although the problem is of both local and regional significance, proper pollution control can prevent deleterious effects. Ground fumigation in the vicinity of the mine from stack emissions is largely prevented by maximizing stack height. However, the probability of ground fumigation is greatly increased during inversion breakup in the winter; the tendency toward ground fumigation is also increased because of the prevailing westerly, southwesterly (summer) and northwesterly (fall) winds which drive air toward the Muskeg Mountain Upland (to the south and east of Ore Bodies No. 1 and 2). Ground fumigation along this upland could be more frequent; however, no major effects on soils or vegetation are expected. Little chance of fumigation is expected at Ore Body No. 4, due to the relatively flat surrounding topography. The cumulative effects of emissions from simultaneous operations in the areas of Ore Bodies No. 1 and No. 2 may cause SO_2 values to exceed current regulations.

3.4.3 LOW-LEVEL EMISSIONS

These emissions come from a variety of sources. Internal combustion engines are a major source of CO, CO_2, and water vapor, which under low-temperature conditions form a dense, sometimes noxious ice fog. Other sources of water vapor include flue gases, incinerators, human respiration and perspiration, and losses from buildings. Nuclei (for ice fog formation) are usually molecular-sized (gases) or slightly larger and originate from combustion by-products, dust, and all sources of gaseous emissions.

These emissions usually do not represent problems in themselves, but in combination with low temperatures and thermal inversions, surface fogs are created and intensified. These effects may be minimized by limiting the use of diesel/gasoline-powered equipment in favor of electrical types and by reducing water vapor emissions. Amounts of ground-level emission decrease proportionately with mine size (because less equipment is used) but likely remain nearly constant per cubic metre of overburden and oil sands removed.

3.4.4 ICE FOG AND LOW TEMPERATURE WATER FOG

During periods of extreme low temperature (-30°C), ice fog can be expected to form over hot tailings discharge points and the immediately-adjacent open pond surface and over plant sites and active mining areas. Ice fog is the result of the condensation of water vapor (most of which is artificially generated since ambient relative humidity is extremely low) around molecular-sized nuclei (mostly artificially generated from gaseous emissions). The presence of these airborne particles increases the temperature at which fogs form, thereby increasing their frequency and duration. Ice fogs also intensify ground-level pollution by taking up gaseous and particulate substances and holding them in a stagnant air mass characteristic of inversion conditions. Gravitational-drainage air flows may carry these fog incidents over pit areas during extended periods of extreme cold and no wind. Otherwise, westerly winds may carry fog offsite to the east.

This is a significant local and regional problem, with the possibility of slowing or halting mining activities (due to poor visibility), and of contributing to offsite fog occurrences. The impediment or prevention of mining activities is only threatened during extended periods of extreme cold (otherwise fogs are quickly broken up and dissipated), and where mining is taking place in proximity to open wet ponds and plant sites where water vapor is emitted. Local topography, prevailing wind direction, and mine site layout indicate that the probability of serious ice fog formation is greatest at the onset of pond storage of heated tailings, during dyke construction and in proximity to plant sites. The ratio of water in the high temperature tailings stream to cool water contained within the pond is critical when considering pond-originated fog. When the ratio is high, the hot water has relatively little effect on overall temperature and the pond remains frozen over.

Mixing of ice fogs and stack emissions is not likely to occur since the inversion conditions that promote ice fog formation ensure that stack emissions are held aloft. Some degree of mixing may occur when inversions break up (but no implied danger), and when fogs and emissions encounter topographic highs (Muskeg Mountain Upland) and turbulence results (possibility of noxious fogs in this case).

Since the most important factors contributing to ice fog formation are the high temperature of the wet tailings stream, the size of the open portion of the active disposal area within the various tailings ponds and the amount of water vapor released at the plant site, mitigative measures may include limiting any or all of these. Water vapor over ponds may be reduced by artificially cooling tailings prior to deposition (for which several methods are available) or better, by discharging in such a way as to retain a frozen pond surface, as is done now after the first stage of pond initiation. A problem with no ready solution is limitation of water vapor emission. Construction of in-pit dykes for tailings or sludge disposal between ponds and pits may help keep fog out of the pits. Some progress has also been made in fog dissipation techniques, though these are still generally unsatisfactory.

3.4.5 EMISSION - FOG INTERACTIONS

Industrial activities and their associated ground-level pollutants (combustion products, gaseous substances, etc.) greatly increase the number of suitably-sized nuclei available for ice-fog particle formation. An increase in the number of the nuclei, combined with the presence of large amounts of water vapor causes increased fog density. The presence of nuclei also results in fog formation at higher temperatures, thus increasing fog frequency and duration. Airborne ice particles and accompanying stagnant air conditions act so as to trap ground-level pollutants, and cause air pollution indices to increase by a factor of 2 or 3. These include SO_2, nitrogen oxides, CO, CO_2, and particulates. It is unlikely that stack emissions will mix with fogs in the mine site area; however, mixing and resultant H_2SO_4 formation may occur along the slope of the Muskeg Mountain Upland.

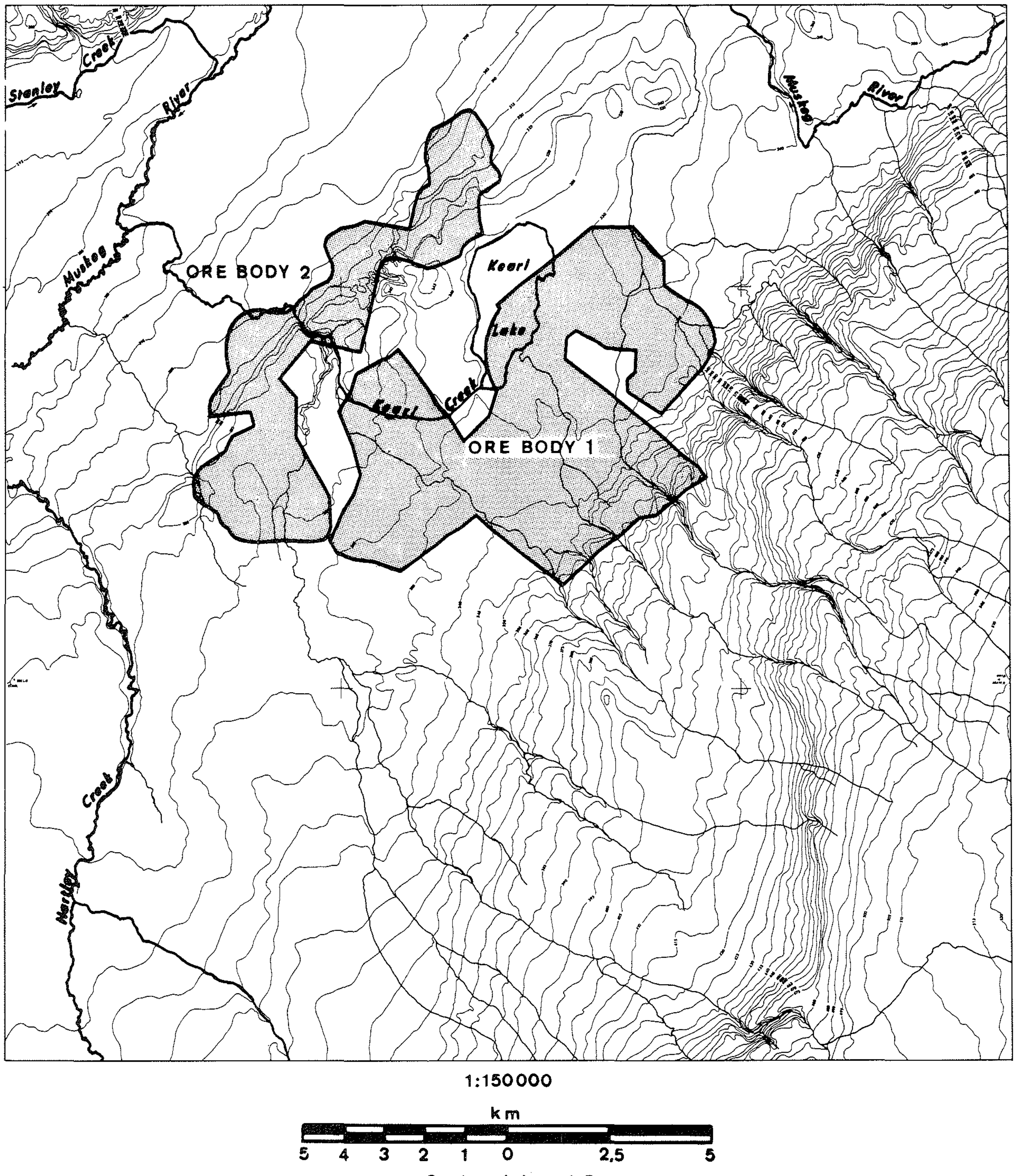
The concentrating effect of ice fog on ground-level pollution sources will probably not contribute to health hazards unless stack emissions mix with ground-level fogs on the slope of the Muskeg Mountain Upland. Mitigative measures include maximizing stack height, minimizing ground fogs and reducing ground-level pollution sources. The Ore Body No. 1 mine, plant, and tailings pond are located at the foot of the ridge forming the Muskeg Mountain Upland. Of the three ore bodies studied, this ore body is the most likely to be affected by emission-fog interactions.

3.5 HYDROLOGY[10]

The study area is situated within the Muskeg River watershed, which drains directly into the Athabasca River, and thence northward into the Mackenzie River. The Muskeg River generally flows from northeast to southwest. Upper portions of the basin lie in an area of relatively heavy timber. This area is known as Muskeg Mountain and is fairly well-drained. The central portion of the basin, adjacent the river itself, is a broad, flat, poorly-drained plain underlain by outwash sands. This area is almost entirely mantled by marshes and treed bogs (muskeg deposits). Hartley Creek, a major tributary of the Muskeg River, drains northward off Muskeg Mountain, and is reported to contain the best aquatic habitat of the study area (see Section 3.8, Aquatic Habitats). Kearl Lake, located approximately midway between Ore Bodies No. 1 and 2, is the only major lake in this watershed. Together with the waterladen bogs that surround it on three sides, Kearl Lake contributes a considerable amount of water to the system. Other waters are contributed by runoff from the southern escarpment of the Fort Hills, to the north of the river.

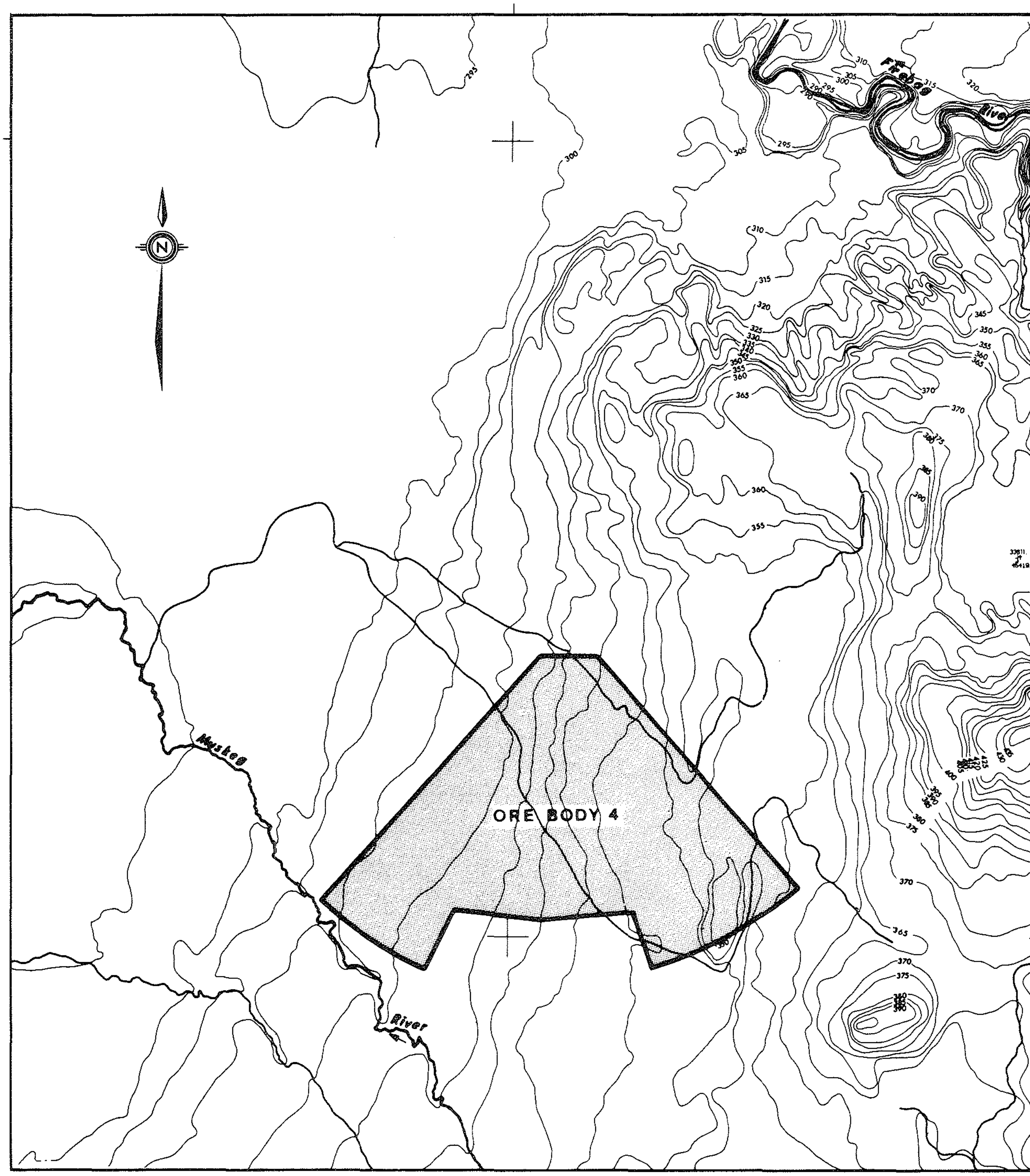
Major hydrologic features of the study areas are shown in Figures 3.5-1 and 3.5-2.

Due to the types of surficial materials found over most of the Muskeg River watershed, much of the precipitation falling as heavy rain is stored as groundwater in surficial aquifers. The generally low drainage density of this basin (except for the slopes of Muskeg Mountain) is a result of this rapid percolation of precipitation into underlying surficial outwash sand layers. Although much of the basin is flat and thus poorly drained, and covered with stunted bog or marsh type vegetation, the steeper areas along the southern, eastern, and northern portions of the basin are well-drained and vegetated with relatively heavy forest that acts to stabilize these slopes. Kearl Lake and the numerous marshes, bogs, and ponds of the Muskeg River watershed act to control runoff and lower flood peaks. They also act to store large volumes of water, and drainage of these lakes and bogs may not only result in sig-



REGIONAL HYDROLOGIC FEATURES ORE BODIES No. 1 AND No. 2

FIGURE 3.5-1



1:75000

km

1 0,5 0 2 4

Contour Interval: 5m

REGIONAL HYDROLOGIC FEATURES ORE BODY No. 4

FIGURE 3.5-2

Prepared for Alberta Department of The Environment

nificantly higher average flows in the Muskeg River, but may also cause greater flow fluctuation and higher flood peaks.

Mean daily discharge of the Muskeg River (recorded at its mouth) is about 7 m^3/sec. Peak flows for the lower part of the river reach 42 - 56 m^3/sec during April and May; heavy August rains have been responsible for flows of 56 m^3/sec resulting in flooding of surrounding areas. Low flows are about 0.3 - 0.4 m^3/sec during February and March. Annual runoff for the entire basin is about 1524 m^3/ha. For Hartley Creek, a major tributary, peak flows on lower portions are about 11.3 m^3/sec, while low winter flows are 0-0.005 m^3/sec.

3.6 SOILS

The soils of the study area have only recently been studied in detail. Early exploratory survey work[11] roughly described soils (in the areas overlying the three development models) as being in the following classes:

Grey Wooded	Mostly developed directly on sandy glacial tills
Minimal Podzol	Developed on outwash and lacustrine sands and gravels mostly overlying glacial tills
Acid Brown Wooded	Developed on lacustrine sands
Organic	Sedge and moss (muskeg) deposits over 30 cm deep overlying various materials.

Surficial materials from which all mineral soils are derived are of a sandy nature. Even the glacial tills are sandy, since these result from sandstone bedrock.

A more modern treatment of the soils in the AOSERP Study Area is in progress, and an interim report with maps is available[12]. This inventory uses the biophysical approach to land mapping. The area is divided into Sub-regions, each of which is further divided into Districts, and Systems. The land unit of significance in this study is the Land System. It is similar in concept to the soil association, and comprises certain defined proportions of various soil types. Where the proportions of soil type vary substantially within a system, two different map unit designations are used (eg., MIL 1 and MIL 2). Combinations of mapping units are used where soil, landform or slope conditions are complex. Table 3.6-1 provides descriptions of physiography, parent material and soil type of each land system and mapping unit in the three development areas considered in this study.

TABLE 3.6-1

LAND SYSTEMS AND SOIL TYPES OF RELEVANCE TO DEVELOPMENT OF THE SELECTED OREBODIES (EXCERPTED FROM TURCHENEK AND LINDSAY, 1978)

				SYSTEM COMPONENTS		
PHYSIOGRAPHY	SYSTEM	UNIT	PARENT MATERIAL	DOMINANT SOILS	SIGNIFICANT SOILS	MINOR INCLUSIONS
Undulating ground moraine and hummocky moraine with some glaciofluvial inclusions; elevation 450 - 500 m.	Kenzie	KNZ 1	Undecomposed to moderately well decomposed moss peat (Bog).	Typic Mesisol	Terric Mesisol Typic Fibrisol	Sedge peat Peaty Gleysols
	Kenzie	KNZ 2	As above	Terric Mesisol	Typic Mesisol Terric Fibrisol	Peaty Gleysols Sedge peat
	Eaglesham	EGL 1	Undecomposed to moderately well decomposed sedge peat (Fen).	Typic Mesisol	Terric Mesisol	Moss peat Peaty Gleysols
Undulating ground moraine with some inclusions of glacio-fluvial and ice-contact deposits; elevation 350 - 650 m.	Kinosis	KIN 1	Moderately coarse to medium textured light brown to brown glacial till.	Orthic Gray Luvisol	Gleyed Gray Luvisol Peaty Gleysols	Organics
	Steepbank	STP 1	As above	Peaty Gleysols	Gleyed Gray Luvisol Orthic Gray Luvisol	Organics
Undulating and duned aeolian sand plains; elevation 400 - 500 m.	Heart	HRT 1	Coarse textured, light yellowish brown to grayish brown, well sorted and loose or poorly compacted aeolian deposits.	Eluviated Eutric Brunisol		Gleyed Eluviated Eutric Brunisol Peaty Gleysols
Undulating glaciofluvial plains; mainly outwash deposits; includes eroded till and meltwater channels along Athabasca and Clearwater Rivers; elevation 250 - 350 m.	Mildred	MIL 1	Coarse textured, brown to grayish brown outwash deposits; generally thick; locally thin over till or bedrock	Eluviated Eutric Brunisol		Gleyed Eluviated Eutric Brunisol Peaty Gleysols
		MIL 2	As above	Eluviated Eutric Brunisol	Gleyed Eluviated Eutric Brunisol Peaty Gleysols	Organics
	Bitumount	BMT 1	As above	Peaty Gleysols	Gleyed Eluviated Eutric Brunisols Eluviated Eutric Brunisol	Organics
Hummocky and rolling, highly dissected kame and kame moraine deposits with some glacial till inclusions; elevation 300 - 350 m.	Firebag	FIR 1	Mainly coarse textured, often gravelly and bouldery brownish to grayish brown ice-contact materials.	Eluviated Eutric Brunisol		Gleyed Eluviated Eutric Brunisol Peaty Gleysols
		FIR 2	As above	Eluviated Eutric Brunisol	Gleyed Eluviated Eutric Brunisol Peaty Gleysols	Organics

Site 1

Based on generalized, exploratory soil survey work[11] and recent mapping[12], Ore Body No. 1 comprises a complex of units which include Orthic Grey Luvisols, developed on glacial till deposits with Organic bog soils (various mesisols and fibrisols) predominating in poorly drained locations. Typical profiles can be found in Turchenek and Lindsay[12]. Complete descriptions of mine facilities developed on the various soil units are provided in Table 3.6-2.

The major portion of this ore body and the areas nearby is mantled by muskeg deposits. The depth of these deposits is not accurately known, and although regionally, deposits usually do not exceed about 1 m, local topography (fairly flat), drainage patterns, and current vegetation types indicate that here deposits may be fairly deep. The area retains large amounts of surface and subsurface water, and controls the local hydrological regime.

Relatively good sources of the sandy textured till are available for reclamation after mining. Abundant muskeg is also available. Sandy clay loam topsoils (Orthic Grey Luvisols) may also be worth salvaging for reclamation purposes where they occur in conjunction with fine-textured surficial materials, or where salvage operations are technically possible.

Replacement of the inherent water storage capacity of the area presently dominated by muskeg may be very difficult, if not impossible. This water storage capacity is critical, however, to maintaining a comparable hydrological regime during and after mining; the maintenance of a delayed flood stage is in turn important from an aquatic fauna viewpoint (see Section 3.8, Aquatic Habitats). Re-establishment of this regime in reclamation schemes (possibly by an inter-connected holding pond system) should be given consideration.

Site 2

Part of the ore body is mantled by Eluviated Eutric Brunisols and peaty Gleysols, with sporadic but substantial muskeg development. These mineral soils are developed on sandy clay loam till which is sometimes overlain by slope wash sands. Example profiles of the dominant soils are presented in Turchenek and Lindsay[12].

For the most part, these soils are poorly to imperfectly drained.

The northern and eastern portions of the ore body, together with the proposed tailings area oversize reject pile, overburden dumps and plant site, display soils developed directly on outwash sands and gravels. The surface soils are also Eluviated Eutric Brunisols, peaty Gleysols and, in depressional areas (such as the plant site and portions of the tailings pond), muskeg of undetermined (but likely shallow) depth.

Although it is less certain than at Site 1, it is probable that adequate amounts of finer-textured surficial materials and muskeg are available for reclamation. Salvage of mineral topsoils is probably not technically feasible for reclamation programs, however, because of their shallow depth and relatively sandy texture.

Site 4
The eastern half of this ore body exhibits similar soil conditions to those in the Fort Hills District, which is typified by coarse-textured, gravelly soils in well-drained sites. Soils are predominantly Eluviated Eutric Brunisols rapidly developed on glacial tills. Although isolated pockets of muskeg occur, they are likely infrequent, and very shallow deposits are most common. The western portion of the ore body has a higher proportion of imperfectly- to poorly-drained peaty Gleysols in combination with the brunisols, developed on outwash sands. The tailings pond and plant site areas are located on soils of similar character and origin (outwash sands and gravels).

Muskeg deposits are sporadic and infrequent over the ore body, and soils tend to be very coarse-textured. Peaty Gleysols may prove to be the best reclamation material available, although localized finer-textured tills may also be useful. Strict soils, till and muskeg salvage practices must be employed when mining this ore body, as reclamation materials are limited. In the event that offsite borrowing of suitable materials is necessary, extensive muskegs are to be found in the area between the ore body and the string bog, to the west of the mine and plant site.

TABLE 3.6-2

MAPPING UNITS AND COMPLEXES FOUND ON VARIOUS OREBODIES AND FACILITIES
(BASED ON MAPPING BY TURCHENEK AND LINDSAY, 1978)

UNIT OR COMPLEX	FACILITY* SITE 1**	SITE 2	SITE 4
KNZ 1	Mine	--	--
KNZ 1 - EGL 1	Mine	Mine (m)	Tailings Pond (m)
KNZ 1 - BMT 1	--	Tailings Pond	--
	--	Mine	--
KNZ 2	Tailings Pond	O.B. Dump	--
	Mine	Plantsite (m)	--
KNZ 2 - STP 1	Mine	Mine (m)	--
	Tailings Pond	O.B. Dump	--
	Plantsite	Plantsite (m)	--
KNZ 2 - HRT 1	--	Tailings Pond (m)	--
KNZ 2 - MIL 2	--	--	Mine
	--	--	Plantsite (m)
	--	--	Tailings Pond (m)
	--	--	O.B. Dump
KNZ 2 - BMT 1	Mine (m)	Tailings Pond	--
	--	Mine	--
	--	O.B. Dump	--
EGL 1	Mine (m)	Mine (m)	--
EGL 1 - KNZ 1	--	Tailings Pond	Tailings Pond (m)
EGL 1 - BMT 1	--	--	O.B. Dump (m)
KIN 1	Tailings Pond	--	--
	Plantsite	--	--
	Mine	--	--
KIN 1 - KNZ 2	Tailings Pond	--	--
KIN 1 - MIL 2	--	Mine (m)	--
	--	O.B. Dump (m)	--
STP 1 - KNZ 2	Tailings Pond	Plantsite	--
	Mine	--	--
MIL 1	--	Mine (m)	--
	--	Tailings Pond	--
MIL 1 - KNZ 2	--	--	Mine
FIR 1	--	Mine	Tailings Pond (m)
	--	Oversize Reject Pile	O.B. Dump (m)
FIR 2	--	--	Mine
	--	--	Plantsite
	--	--	O.B. Dump (m)
	--	--	Tailings Pond
FIR 2 - KNZ 2	--	--	Tailings Pond (m)

* Those facilities located on this soil unit in the vicinity of the orebody development indicated.

** Those units which are only minor (<10%) contributors to the soils overlying a facility are suffixed by (m).

Those units or components underlined are predominantly organic in nature (for details see Table 3.6-1).

General Remarks

A few statements can be made concerning the general agricultural and forestry capability of soils in this region as compared to those encountered elsewhere in the province. Forest development on organic soils tends to be unproductive and stagnant conditions result in stunted stands of poor quality timber. Attempts are being made, however, to improve forest productivity through improvements in local drainage regimes. The primary commercial uses of organic soils in Canada are as soil amendments, as support for special agricultural crops (blueberries, wild rice, etc.) and for limited supplementary grazing[13]. Although organic soils occur throughout the province, their occurrence is much more extensive in the cooler northern parts of Alberta.

The Eluviated Eutric Brunisols belong to the Brunisolic order of soils, and are usually developed on basic or calcareous parent materials of medium to coarse texture (glaciofluvial outwash, morrainal, and aeolian). These soils underlie much productive forestland in Canada, but are infrequently cultivated; this is related to the geographic and climatic regions in which they are found (usually alpine or sub-arctic). Where they support nonproductive forests, limitations usually result from climatic severity, rough or steep topography, shallowness, and stoniness. Rapid drainage caused by coarse texture may result in local moisture deficiency problems.

The peaty Gleysols are members of the Gleysolic order. Although these have not been differentiated on soils mapping of the study area, three sub-groups are represented, namely Orthic Luvic Gleysols, Orthic Gleysols, and Rego Gleysols, all with surface organic layers (up to about 40 cm thickness). These intergrade in appearance with Terric Mesisols with which they are associated. These soils generally support wet forests, shrub-marshes, and grass-sedge meadows. Forests developed on gleysols are generally unproductive due to poor drainage, which results from factors related to typical climate and geography (short growing season, cool climate). When drained, productivity may increase substantially. Deep tilling of the overlying peat into the general profile might also increase productivity. Without proper management these soils rarely support commercial forests or cultivation.

Orthic Grey Luvisols belong to the Luvisolic order. In northern Alberta, these soils often support productive commercial-quality forests, but under more favorable climatic conditions, they are frequently cultivated or used for grazing. In addition to climatic restrictions, agriculture on these soils is limited by low organic matter, poor physical structure and low initial fertility. Addition of significant quantities of nitrogen, phosphorus, and sulphur is usually necessary.

Soils in Alberta with fair to good agricultural capability and productivity range from those of the Solonetzic to the Chernozemic orders. These are associated with the open prairie and aspen parkland regions of the province. Solonetzic soils are characterized by problems of sodium accumulation, and require intensive land management in order to obtain and sustain satisfactory crop yields. Chernozemic soils are the most fertile soils in Alberta, and require only a moderate amount of management.

3.7 VEGETATION

A typical hydrological distribution of forest stands at the three study sites is:

Dry Conditions: (upland; often sandy or gravelly; often southerly facing; typically little organic matter accumulation; Podzols, Dystric Brunisol soils predominate) Jack Pine (*Pinus banksiana*) and Aspen Poplar (*Populus tremuloides*) stands.

Mesic Conditions: (imperfectly drained; flat to slightly undulating; soil types include Organic, Grey Luvisols and medium textured types) White Spruce (*Picea glauca*), White Birch (*Betula papyrifera*) and Balsam Poplar (*Populus balsamifera*) stands.

Wet Conditions: (poor drainage; depressional area, sometimes with standing water; mostly Organic soils, sometimes Gleysols) Black Spruce (*Picea mariana*), Tamarack (*Larix laricina*) and sedge or shrub marshes.

3.7.1 VEGETATION CLASSIFICATION

A number of vegetation classification systems have been devised for the oil sands region, based both on field studies and photo interpretation. The various systems have been developed to suit individual purposes and are variously restricted in scope. Renewable Resources Consulting Services Ltd.[14,15] outlined a simple classification system based primarily on tree species composition. More recent surveys have concentrated on collection of site information in developing floristic classifications. These have included Stringer[16], who sampled stands throughout the AOSERP Study Area, Peterson and Levinson[17], who concentrated their efforts on the Syncrude Lease 17, and R.M. Hardy and Associates Ltd.[18], who surveyed the G.C.O.S. Lease 86. Airphoto-based mapping projects that incorporated these floristic data to a greater or lesser extent include Shell Canada Limited[19], Turchenek and Lindsay[12], Thompson *et al*[20], and Alsands Project Group[21].

The following descriptions and discussions are based on Thompson *et al*[20] habitat mapping project for the AOSERP Study Area. This system seems to incorporate the results of detailed floristic analysis (Stringer3ul6) with FCIR photograph interpretive characteristics. It also has incorporated edaphic and wildlife components in a comprehensive way.

a. Wetland Communities

These are of four major types, plus an undifferentiated, complex unit which is difficult to map on a large scale.

(i) Fen

- Poorly drained, on level to slightly sloping sites; often part of slow-moving surface or near surface drainage systems.
- Development corresponds with areas where drainage or flow of water occurs.
- Found on organic soils of decomposing sedges (mesisols).
- Composed of sedges (*Carex spp.*) and a near-continuous low shrub community, often swamp birch (*Betula pumila*); willow (*Salix spp.*) and alder (*Alnus sp.*) sometimes also occur.

(ii) Black Spruce Bog

- Poorly drained, on level to depressional sites; generally high water tables.
- Drainage and internal circulation of water is poor, resulting in stagnant conditions.

- Generally found on organic soils developed from *Sphagnum* or peat moss fibrisols.

- Sites dominated by dense stand of black spruce (*Picea mariana*) (up to 10 m height).

- Tamarack (*Larix laricina*) are sometimes present; low shrub layer is usually of Labrador tea (*Ledum groenlandicum*), blueberry (*Vaccinium spp.*) and cranberry (*Rubus spp.* and *Oxycoccus sp.*); grasses and sedges (*Carex brunnescens*) are sometimes present.

(iii) Semi-open Black Spruce Bog

- Similar in hydrology and soils to (ii), but slightly more moist.

- Soils are dominantly fibrisols.

- Black spruce stands are less dense than (ii) and tend to include more shrubs such as bog birch (*Betula glandulosa*), and willows; tamarack is more often present; groundcover is also similar to (ii), but often with a more diverse herb flora.

(iv) Lightly Forested Tamarack and Open Muskeg

- Drainage is poor; sites are wetter than both (ii) and (iii).

- Soils are typically acidic fibrisols.

- As with other bog forests, the nutrient status is low.

- Tree growth is sparse and of low height (often below 2 m); shrub cover is extensive and similar to (iii); understory is also similar to (iii).

b. Bottomland and Riparian Communities

(i) Deciduous Shrub

- Occupies a zone between rivers and streams, and riparian forest.

- Also found on river bars and along minor drainage courses throughout the area.

- Depending on local drainage characteristics, soils comprise coarse-textured fluvial types with mesisols.

- Shrubs dominate, notably willow, alder (5 - 8 m), white birch (Betula papyrifera) and immature poplar (Populus spp.); dogwood (Cornus stolonifera) may also be present in a lower layer.

- Trees are essentially absent and the herb layer is sparse.

(ii) Bottomland and Riparian Forest

- Found on floodplains of major rivers and tributaries.

- Soils are often of alluvial origin; nutrient status is good.

- Mature forests of balsam poplar (Populus balsamifera) and/or white spruce (Picea glauca); stands may be pure or mixed, (up to 30 m).

- Aspen poplar (Populus tremuloides) and white birch may also be present.

- A dense tall shrub layer of willow and alder is usually present, and is accompanied by less dense low shrubs and herb layers.

c. Upland Communities

(i) White Spruce - Aspen Forest

- Usually occurs on well- to moderately well-drained sites, but is sometimes found on wetter sites depending on soil type and nutrient status.

- Soils are variable.

- Aspen poplar and white spruce are usually the dominant trees, but significant occurrences of white birch, jack pine (*Pinus banksiana*), balsam fir (*Abies balsamea*), balsam poplar and black spruce may sometimes be found.

- Differences in relative abundance of the principal trees is a reflection of site-specific successional patterns and stages.

- Understories are generally diverse, particularly in deciduous-dominated stands; pure stands of white spruce have less abundant shrub and herb floras.

(ii) Mixed Coniferous

- Variable drainage, depending on site conditions and dominant conifer present.

- Most often medium to poor drainage on level or slightly sloping ground with a discontinuous veneer of peat.

- Forest cover is dominated most frequently by black spruce, but jack pine and white spruce may also be present; stands are dense and usually tall (over 6 m).

- Understory is relatively diverse, particularly the low shrub and herb layers (including feather mosses).

(iii) Jack Pine

- Dry, sandy sites often on aeolian deposits.

- Jack Pine is the only species of tall trees in this upland community (up to 15 m), but aspen and black spruce may also occur, especially if the landform pattern comprises sandy hills interspersed with poorly drained upland sites.

- Understory tends to be very sparse due to dryness; no tall or medium height shrub layer; low shrubs include blueberry (Vaccinium myrtilloides), and bearberry (Arctostaphylos uva-ursi); herb layer includes grasses and fruticose lichens.

(iv) Upland Open

- Open grassy meadows occur as openings in the aspen forest; soils and hydrology the same as the surrounding forest (usually pure aspen stands).

d. Miscellaneous Communities

These include recent burns, highly disturbed sites, non-vegetated sites (often as a result of recent slides or slumps), and aquatic communities.

3.7.2 VEGETATION CLASSIFICATION OF STUDY SITES

A reasonable amount of variability exists within these vegetative types, and intergrading and seral stages are common. With this in mind, only an approximate vegetative characterization is possible. Vegetation habitat mapping is available from AOSERP[20]. Timber management plans and data on Timber Quota holders is available from the Alberta Forestry Service.

Site 1

Throughout the organic soil zone to the east and south of Kearl Lake, and adjacent to the lake outflow, the Wetland Communities and the Deciduous Shrub unit of the Bottomland and Riparian Communities predominate, depending on the height of the local water table. Inundation and stagnation of surface water tend to keep most of this area in a low productivity class with no potential for commercial (forestry) use under present conditions. Under these conditions, very deep muskeg deposits can sometimes be expected, and at least locally, permafrost may be encountered (at 50-90 cm).

To the south and east of the organic soil zone, the land slopes upward onto the Muskeg Mountain Upland, and glacial till deposits underlie Grey Luvisols. Mature Upland Communities are bisected by Deciduous Shrub development along channels draining the Upland. Because soils are characteristically fairly coarse-textured and drainage is considerably improved, site productivity is good. Stands of commercial quality timber are present.

The proposed tailing area and plant site are located in this well-drained, productive forest zone. Timber harvesting and salvage will be necessary. The development of the mine at this location, together with its associated tailings area and plant site, will also have major visible impacts, but these will be of relatively minor importance. The major direct impact will be the loss of several square miles of commercial timber of moderate value. Reclamation to vegetation of equivalent or greater value (regionally) is possible, especially at the Improved and Enhanced Levels.

Site 2

The majority of this ore body is covered in immature poplar- and white spruce-dominated Upland Communities, with sporadic occurrences of Wetland and Deciduous Shrub Communities, where local drainage conditions warrant. These minor vegetation types are most clearly developed in the southwestern muskeg portion of this ore body. The predominantly slope wash and glaciofluvial outwash sand parent material of the soils over this ore body indicates that muskeg deposits underlying Wetland Communities are probably fairly shallow, except for occasional deep deposits in some depressional areas.

The tailings pond and plant site are partly located on similar Wetland Community types, but because of the proximity of the tailings pond to the Muskeg River, peat deposits are expected to be somewhat deeper. Although no commercial quality timber is known to be located over the ore body or any of the associated facilities, the ridge which runs southwest to northeast along the length of the ore body is moderately productive and offers good wildlife habitat potential. Impacts to vegetation resources are considered of moderately low importance.

Site 4

This is a considerably drier site than those previously discussed, and for the most part, has proportionately more productive vegetation communities (since they are not limited by stress associated with flooding). Communities developed include Mixed Coniferous (mine area) and White Spruce-Aspen Forest. Some stands located on the tailings pond site are considered to be of commercial value. The central part of the ore body has some muskeg development, but this probably reflects only shallow peat deposits.

Site productivity indicates that moderate wildlife habitat potential exists, and that development of the tailings pond would involve the loss of possible commercial timber stands.

As at Site 1, prior timber harvesting and salvage will be necessary. Impacts to vegetation are considered minor compared to those in other resource areas.

3.7.3 GENERAL REMARKS

Generally, regional vegetation is of low productivity due to adverse climate (short growing season, etc.), poor soil conditions (excessive sandiness or organic matter content), and prolonged inundation by surface water tables. This implies three things:

1. Properly planned and conducted reclamation operations can improve both the productivity and the utilization potential of vegetative communities.

2. Although absolute values of vegetation resources are low compared to province-wide norms (and so, too, therefore are absolute impacts), the rarity of commercial grade timber regionally implies somewhat higher relative impact values.

3. The loss of fairly productive wildlife habitat is also a notable impact from a regional point of view.

3.8 AQUATIC HABITS

The following represents a consideration of aquatic habitats for major streams in the study area, together with a list of recommendations for mine design and environmental protection.

Muskeg River

The lower Muskeg River is considered to be moderately important as fish habitat and has slight limitations for sport fish production[22,23,24]. Mature Arctic grayling (Thymallus arcticus), northern pike (Esox lucius), and yellow walleye (Stizostedion vitreum) have been collected in this area, and large numbers of longnose sucker (Catostomus catostomus) and white suckers (Catostomus commersoni) are known to use the lower reaches for spawning. The ability of the river to overwinter sport fish is presently unknown.

The middle sections of the Muskeg River have less importance as fish habitat. A low gradient is characteristic of this zone, and extensive areas of laminar flow have been identified. Some high-quality pools associated with riffle areas have been noted. Fish species collected in this section include northern pike and Arctic grayling.

The upper sections of the Muskeg River are considered to have poor fish habitat[24] because of restricted channel size, low gradient, and low winter flows. Numerous beaver dams also restrict fish movement from the lower Muskeg River. The upper reaches of the Muskeg River have no measurable winter discharge[25].

Hartley Creek

Hartley Creek is the main tributary of the Muskeg River, and has relatively good water quality, high sustained flows, and good spawning habitat for salmonid fishes[19]. Arctic grayling is the dominant species in the upper section of Hartley Creek[22], while a few mountain whitefish are also present. Excellent spawning habitat has been observed throughout the area. Lower sections are generally inferior to the upper section.

All three sites examined during the present study involve the Muskeg River. The synergistic effects of developing these sites in combination with the Shell development must be considered when evaluating potential impacts. An example of this relates to disposal of saline water. The summer discharge of the Muskeg River may be sufficient for surface disposal of saline water from one site but not for 2 or more. Alternate methods of saline water disposal are required for sites in the Muskeg watershed.

Firebag River

The Firebag River provides some of the best fish habitat in the northeast region of Alberta[24]. This stream supports substantial populations of yellow walleye, northern pike, and Arctic grayling. Sections of the stream adjacent to Site 4 are characterized by moderate gradients and pool-riffle ratios of 1:1. Large deep pools often 300 m or more in length provide potentially excellent habitat for northern pike and yellow walleye. These pools could also be utilized as wintering areas for resident fish populations.
The regional importance of the Firebag River as fish habitat requires that no effluent or runoff from disturbed sites should enter this stream. The total watershed area disturbed must be kept to a minimum.

With regard to impact on aquatic habitats, the following general recommendations apply:

a. Protective measures should be sufficient to maintain water quality in downstream areas.

b. Because of the relative importance of Hartley Creek as fish habitat in the Muskeg drainage, diversions to Hartley Creek should be avoided.

c. Highway routing should avoid proximity to or paralleling of streams. Crossings should be made as far upstream from the mouths of tributaries as possible.

d. Low winter flows in the Muskeg drainage are not sufficient for adequate dilution of saline water. Summer flows will not support saline water disposal by more than one plant. Alternate disposal methods should be considered (i.e., tailings pond or deep well injection).

e. An undisturbed zone adjacent to all major streams should be incorporated in mine plans. Plant and tailings ponds should not be within flood zones.

f. Plant and tailings ponds should be located in such a way as to prevent formation of narrow corridors along the Muskeg River.

g. Muskeg River crossings should be kept to a minimum. All crossings should be designed to allow fish movement.

h. No gravel should be removed from the Muskeg River or its tributaries.

i. Reclamation of disturbed areas of the lower Muskeg River and Hartley Creek should include the replacement of suitable spawning and overwintering areas.

j. Streamside vegetation is a critical factor for maintaining suitable water quality and providing protective cover. Reclamation planning should include the immediate development of suitable soils and streamside vegetation along all diversions, major or permanent drainage systems, and along disturbed streams.

k. Whenever possible the same corridor should be used for roads and utilities.

l. Stream channel reclamation should include measures for increasing habitat diversity through the use of drop structures, rock islands, and wing deflectors.

m. Construction activities near streams should be scheduled before and after spawning or migration periods.

n. Stripping of large areas of muskeg, stream diversions, and stream channelization will cause reduced upstream stages and increased peak flows in downstream areas. Settling ponds and storage areas are required to reduce potential flooding in downstream areas.

Characteristics of aquatic habitats and potential impacts of development at each site are presented in the following tables:

Table 3.8-1	Site 1 - Characteristics
Table 3.8-2	Site 1 - Impacts
Table 3.8-3	Site 2 - Characteristics
Table 3.8-4	Site 2 - Impacts
Table 3.8-5	Site 4 - Characteristics
Table 3.8-6	Site 4 - Impacts

Table 3.8-1

SITE 1 - CHARACTERISTICS

Aquatic Habitats	Characteristics	Scale of Importance	Recommendations
General	Development of Site 1 involves the headwaters of Muskeg River, Kearl Lake, a large area of muskeg, and portions of an upland groundwater discharge area. Approximately 9 percent of the total Muskeg River drainage will be affected by development of Site 1 (runoff of approximately 19.7 x 10^6 m^3 per year).	Significant impact on the entire drainage.	Protective measures should be sufficient to maintain water quality in downstream areas. Methods for controlling runoff must be incorporated in mine design.
Upper Muskeg River Tributaries	No information on fish populations. Low stream gradient in muskeg areas, silty bottom, occlusion by macrophytes and low habitat diversity reduce fish habitat potential. Headwater areas along slopes of Muskeg Mountain have higher gradient and more diversity but are not expected to have fish populations. The upland area may be a groundwater discharge area for aquifers in the Grand Rapids formation.	Considered to be of low importance as fish habitat.	
"Kearl Creek"	No information on fish populations. The relatively large drainage area probably has a fairly consistent discharge and may provide some overwintering areas in lower reaches. Seasonal use by Muskeg River populations is also possible. Upper reaches have numerous beaver dams which probably limit fish access to Kearl Lake area.	Considered to have low fish habitat potential.	
Kearl Lake	Kearl Lake is a shallow lake with a maximum depth of 1.8 m and a surface area of 5.4 km^2. Its shallow depth and heavy macrophyte populations limit fish habitat potential. No fish populations are known in the area. The lake drains an area of 67 km^2 (26 mi^2) and forms part of a significant surface water storage area.	Low fish habitat potential. Important surface water storage area.	Possible effects Site 1 development may have on surface runoff require settling ponds and storage facilities to delay runoff.

Table 3.8-2

SITE 1 - IMPACTS[26]

Activity	Probable Effects on Aquatic Systems in Area	Scale of Effect	Recommendations
Diversion Channels	Protection of the mine site will require streams draining the upland areas of Muskeg Mountain to be diverted. Diversion to the north will be to the upper Muskeg River. Diversions to the south-west will be to Hartley Creek. The fish habitat of Hartley Creek should be protected if possible.	Major diversions.	Diversions to Hartley Creek should be avoided.
Road Construction	Road construction will result in surface erosion and higher sediment loads in Muskeg River.	Local.	Highway routing should avoid close proximity or paralleling along stream.
Surface Drainage	Kearl Lake and a large area of muskeg will require drainage. Discharge in receiving tributaries and Muskeg River will be increased and result in increased erosion, untable channels, and higher sediment loads.	Affects full length of Muskeg River and some tributaries.	
Sub Surface Drainage (using wells)	Dewatering operations will draw down watertable and affect groundwater flow in area. Lower hydrostatic pressures may allow saline groundwater to enter freshwater aquifers. Some local subsidence possible. Water will probably be of poor quality.	Local.	Water should be disposed of in tailings pond.
Saline Water Disposal	Large mining area may result in the need to dispose of large volume of saline water. Disposal in surface waters may cause serious damage to downstream fish habitat. Winter flows are not sufficient to allow surface disposal year-round.	Could have significant effect on entire Muskeg River.	Saline water should not be released to streams during winter months. Alternate disposal methods should be considered (i.e., tailings pond or deep well injection).
Vegetation Removal	Lower retention capability of area will result in earlier response, faster rates of runoff, higher volumes, and higher peak flows. Combined effect would be unstable channels, increased soil erosion, and high sediment loads in Muskeg River. Large area of disturbed land at Site 1 may result in higher frequency of flooding in downstream areas of Muskeg River.	Significant effect on entire drainage.	Settling ponds and storage facilities to delay runoff are required.
Overburden Removal	Large area (approximately 93 km^2) will be involved in overburden removal. Infiltration of water will be lower, resulting in more surface runoff, high sediment loads, etc.	Significant effect on Muskeg River.	Settling ponds and storage facilities to delay runoff are required.
Pit Excavation	Pit water will probably be of poor quality (containing bitumen, etc.) and require disposal in a pond.	Significant effect on Muskeg River.	Pit water should be disposed of in tailings pond.
Tailings Deposits	Source of supply for contaminants (sodium hydroxide, bitumen, etc.) which by infiltration or erosion could reach surface runoff.	Local but important.	Requires erosion protection and methods for returning contaminated surface runoff.
Tailings Pond	Contamination of surface and groundwaters may result from percolation, seepage or local failures. Hydrostatic head of pond will create man-made recharge area.	Local but important.	Probability of percolation, seepage and failure must be minimized.
Water Requirements	Large amount of make-up water will be required for plant operation (approximately 1.7 m^3/sec). This will require collection, transport and storage facilities. Utility corridor will probably be different from road access.	Utility corridors may affect large part of Muskeg Basin.	Wherever possible the same corridor should be used for road and utilities.

Table 3.8-3

SITE 2 - CHARACTERISTICS

Aquatic Habitats	Characteristics	Scale of Importance	Recommendations
General	Development of Site 2 involves the middle and lower sections of the Muskeg River and two major tributaries: Hartley Creek, "Shell tributary E", and an unnamed tributary draining Kearl Lake (referred to hereafter as "Kearl Creek"). Local flooding is known to occur along the Muskeg River. Parts of Site 2 plant and ponds may be within a flood zone.		An undisturbed zone adjacent to all major streams should be incorporated in mine plans. Plant and pond should not be within flood zone.
Muskeg River (Middle)	Muskeg River, upstream from Hartley Creek, is a low gradient stream with extensive areas of laminer flow. Riffle areas are sometimes associated with high quality pools. Mature northern pike and Arctic grayling have been collected in this area. The proposed Shell project would require diversion of a section of the stream near the confluence with Hartley Creek.	Considered to be of moderate to low importance as fish habitat.	Plant and tailings pond must be located to prevent formation of narrow corridor between Shell mine and Site 2 development.
Muskeg River (Lower)	In general the lower reaches of the Muskeg River (downstream from Hartley Creek) present no severe habitat limitations for most fish species and are considered to have good fish habitat.	Considered to be of moderate to low importance in overall Athabasca drainage.	Upstream sites affect this region Protective measures should be sufficient to maintain overall water quality in downstream areas. Combined effects of 2 or more plants should be considered.
Lower "Kearl Creek"	No information on fish populations. The relatively large drainage area probably has a fairly consistent discharge and may provide some overwintering areas in the lower reaches. Seasonal use by Muskeg River populations is also possible. This stream bisects the tailings pond area and water quality will be difficult to maintain once development proceeds.	Considered to have low fish habitat potential.	Protective measures should be sufficient to maintain overall water quality in downstream areas.
Shell Tributary "E"	No information on fish populations. Lower section may be used seasonally by Muskeg River populations. Development of Site 2 plant area may eliminate portions of this stream.	Small drainage area makes this stream of low importance.	
Hartley Creek	The water quality of Hartley Creek, its relatively high sustained flows and the presence of good spawning areas make this the best fish habitat in the Muskeg drainage. Upper sections are generally superior to lower sections, Arctic grayling is the dominant species.	Excellent fish habitat.	Undisturbed areas adjacent to Hartley Creek are required. No diversions into this stream should be allowed.

Table 3.8-4

SITE 2 - IMPACTS[26]

Activites	Probable Effects on Aquatic Systems in Area	Scale of Effect	Recommendations
Diversion Channels	Major diversion may be required for "Kearl Creek" in mining area. Stream flow will be reduced or eliminated in lower sections of Kearl Creek. Waters diverted south to Hartley Creek could have major impact on fish habitat with increased discharge, resulting in lateral or vertical instability, and higher sediment loads in Hartley Creek. Large lake may form if diversion dams are required. The level of Kearl Lake may be raised.	Local effect to "Kearl Creek" may have significant impact on Hartley Creek.	Diversion of Kearl Creek should avoid Hartley Creek (perhaps by diverting north-west directly to Muskeg River).
Road Construction	Extension of roads from Shell area will produce only minor impacts on aquatic habitats. Road construction will result in some additional surface erosion and higher sediment load.	Moderate to low.	Muskeg River crossings should be kept to a minimum. All crossings should be designed to allow fish movement. Roads paralleling Hartley Creek should be avoided.
Surface Drainage	Extensive areas of muskeg will require drainage in the plant and tailings pond areas. Will result in increased discharge in Muskeg River with some reduction in water quality. Lag time between precipitation events and subsequent high runoff would be shortened. High sediment loads are expected in receiving streams.	Will have considerable local effects in receiving streams.	Surface drainage waters should avoid Hartley Creek. Settling ponds and storage facilities to delay runoff are required.
Sub Surface Drainage (using wells)	Dewatering operations in mine area will draw down watertable and affect small waterbodies in area. Water will probably be of poor quality.	Local.	Water should be disposed of in tailings pond.
Saline Water Disposal	Surface disposal of saline water will result in degredation of Muskeg River water quality. Combined effects of two or more plants could result in serious damage to downstream fish habitat. Winter flows in Muskeg River are not sufficient to allow surface disposal year-round.	Significant effect on lower half of Muskeg River.	Saline water should not be released during winter months. Consideration must be given to saline water disposal by other possible plants in area.
Vegetation Removal	Lower retention capability of area will result in earlier response, faster rates of runoff, higher volumes, and higher peak flows. Combined effect would be unstable channels, increased soil erosion, and high sediment loads in Muskeg River.	Significant effect on lower half of Muskeg River.	Settling ponds and storage facilities to delay runoff are required.
Overburden Removal	Infiltration amounts would be lower, resulting in more surface runoff, higher sediment loads, etc.	Significant effect on lower half of Muskeg River.	Settling ponds and storage facilities to delay runoff are required.
Pit Excavation	Pit water will probably be of poor quality (containing bitumen, etc.) and require disposal in a pond.	Local.	Pit water should be disposed of in a tailings pond.
Tailings Deposits	Source of supply for contaminants (sodium hydroxide, bitumen, etc.) which by infiltration or erosion could reach surface runoff.	Local but important.	Requires erosion protection and methods for returning contaminated surface runoff. Deposits should be kept some distance from Muskeg River and Hartley Creek and outside of flood zone.
Tailings Ponds	Close proximity of Site 2 pond with Hartley Creek, Muskeg River, and Kearl Creek may result in contamination through percolation, seepage or local failures. Hydrostatic head of pond will create man-made recharge area.	Local but important.	Extensive muskeg areas must to removed to insure dyke footing. Probability of percolation, seepage, and failure must be minimized.

Table 3.8-5

SITE 4 - CHARACTERISTICS

Aquatic Habitats	Characteristics	Scale of Importance	Recommendations
General	Site 4 involves two major drainage systems: The Firebag and Muskeg Rivers. Environmental protection (maintenance of water quality, prevention of siltation, monitoring, etc.) is made much more complicated by involving both systems.	Significant impact, since Site 4 can affect two major drainage systems.	Attempts should be made to restrict the impact to one drainage or to prevent any effluent from entering the Firebag Drinage system.
Firebag River	The section of the Firebag River adjacent to Site 4 has been identified as one of the higher quality stream in N.E. Alberta. Watershed management in the Firebag basin is essential in order to ensure long-term fish habitat quality and avoid any impact which may reduce habitat potential.	One of the most important streams in the oil sands area.	No effluent or runoff from disturbed sites should enter this stream. Total area disturbed must be kept to minimum to protect watershed.
Upper Muskeg River	No information exists on fish distribution although restricted channel size, high gradient, and low flow limit possible use by fish. Numerous beaver dams are located in the stream adjacent to Site 4.	Low fish habitat potential.	Environmental protection measures should be sufficient to maintain overall water quality which would protect fish habitats in downstream areas.
Lower Muskeg River	In general the lower reaches of the Muskeg River present no severe habitat limitations for most fish species and are considered to have good fish habitat.	Considered of moderate to low importance in overall Athabasca drainage.	Synergistic effects of several plants must be considered in developing mine plan.

Table 3.8-6

SITE 4 - IMPACTS[26]

Activities	Probable Effects on Aquatic Systems in Area	Scale of Effect	Recommendations
Diversion Channels	No major diversions are expected. Minor diversion may produce higher sediment loads but will not have direct effect on other water quality parameters. No fish habitats will be affected by the required diversion.	Local	
Road Construction	Assuming access is a continuation from the Sheel lease area the access road will parallel Muskeg River for most of its length. Construction will result in increased surface erosion and higher sediment loads in Muskeg River and Tributaries crossed by road. Removal of stream gravel may seriously damage spawning habitat.	Affects full length of Muskeg River.	Highway routing should avoid close proximity or paralleling of Muskeg River. Crossings should be made as far upstream from the mouth of tributaries as possible. No gravel should be removed from the Muskeg River or tributaries.
Surface Drainage	Areas of muskeg at mine site will require drainage. Upper reaches of Muskeg River and its tributaries will receive additional flows. Plant and tailings pond may require some surface drainage.	Local	Wherever possible surface drainage waters should be directed to the Muskeg River.
Sub Surface Drainage (using wells)	Dewatering operations will draw down watertable and affect small waterbodies in area. Water will probably be of poor quality.	Local	Water should be disposed of in a tailings pond.
Saline Water Disposal	Surface disposal of saline water will result in degradation of Muskeg River water quality. Combined effects of Site 4 and other plants disposing of saline water could result in serious damage to downstream fish habitat. Winter flows in Muskeg River are not sufficient to allow surface disposal year-round.	Affects full length of Muskeg River.	Saline water should not be released during winter months. Consideration must be given to saline water disposal by other plants in area.
Vegetation Removal	Lower retention capability of areas will result in earlier response, faster rates of runoff, higher volumes of runoff and higher peak flows. Combined effect would be unstable channels, increased soil erosion, and high sediment loads in watercourses. Both Muskeg and Firebag watersheds may be involved.	Affects full length of Muskeg River and part of Firebag River.	Settling ponds and storage facilities to delay runoff are required. Where possible, surface runoff to Firebag River should be prevented or treated (in settling ponds, etc.).
Overburden Removal	Infiltration amounts would be lower, resulting in more surface runoff, high sediment loads, etc.	Full length of Muskeg River and part of Firebag River.	(Precautions similar to vegetation removal above).
Pit Excavation	Pit water will probably be of poor (containing bitumen, etc.) quality and require disposal.	Local	Pit water should be disposed of in a tailings pond and be used as plant process water.
Tailings Deposits	Source of supply for contaminants (sodium hydroxide, bitumen, etc.) which by infiltration or erosion could reach surface runoff.	Local	Requires erosion protection and methods for returning contaminated surface runoff.
Tailings Ponds	Close proximity of Site 4 pond to Firebag River may result in contamination through percolation, seepage or local failures. Hydrostatic head of pond will create man-made recharge area.	Local but important.	Percolation, seepage, and failure probability must be minimized. Permeability of pond area must be determined and the possibility of sink holes in the area must be eliminated.

3.9 TERRESTRIAL FAUNA

The following list outlines those species or groups of species which occur in the study region, their relative abundance and distributions, and their regional importance. Site-specific information is discussed subsequently.

Moose

The oil sands region is only moderately productive for ungulates, moose (Alces alces) being clearly the most important of these. Canada Land Inventory (CLI) rates most of the area as classes 4 and 5 for moose, but specific critical ranges of considerably greater value for production and wintering exist. Although seasonal habitat preferences vary, tall shrub and deciduous habitats are preferentially occupied, and coniferous and disturbed habitats are avoided. River valleys appear to be favoured, particularly for wintering. In fact, the area immediately adjacent to the Firebag River is rated class 3W, and supports wintering populations from nearby areas (probably including our study areas). A 1972 survey[27] showed similar densities of moose in the present 64,750 ha study area as in the overall oil sands area (0.19 per km^2).

Other Ungulates

Deer (Odocoileus hemionus and O. virginianus), woodland caribou (Rangifer caribou) and buffalo (Bison bison) may occur on the study area from time to time, but regionally, their occurrence is uncommon. No special comments regarding habitat or distribution seem warranted.

Beavers

Regionally, beavers (Castor canadensis) are both abundant and economically important. They are strongly associated with small streams bordered by aspen or balsam poplars. Willows are also common habitat components. Beavers are the most important fur species in this region, and provincially. However, their overall abundance and their under-utilization by trappers[28] indicates that minor reductions in numbers will not be regionally important. Economic impacts to specific trappers may be significant, however.

Other Furbearers

Other furbearers which make important contributions to trapper income (as opposed to absolute numbers trapped) include lynx (Lynx canadensis), muskrat (Ondatra zibethica) and mink (Mustela vison). The lynx tends to be associated with snowshoe hares which frequent tall willow and spruce-willow areas. Muskrats are found only around fairly shallow lakes, and are only rarely trapped on traplines which cross the study area. Mink are associated with muskrats, and with riparian stream habitats.

Waterfowl

Generally, the oil sands area is only moderately productive of waterfowl (CLI classes 5 and 6); however, a few waterbodies are quite important regionally since the oil sands (and the nearby Peace-Athabasca Delta) are located on four migratory flyways. McClelland Lake is especially important as a fall staging lake but it also supports a reasonable number of breeding pairs in summer[29]. Kearl Lake is less important for fall staging, but supports a somewhat higher density of breeders than does McClelland Lake. The string bog southeast of McClelland Lake also supports some fall staging waterfowl, but its main importance is the high breeding density it shows, and its value for spring migrants. Presumably, its shallowness ensures that it is one of the first aquatic areas in the region with open water in spring, thus attracting large numbers of early arrivals.

The most common species utilizing the area are lesser scaup (Aythya affinis), mallard (Abas platyrhynchos), American widgeon (Mareca americana), blue-winged teal (Anas discors), common goldeneye (Bucephala clangula) and white-winged scoter (Melanitta deglandi).

Factors limiting the quality of aquatic habitat for waterfowl include low soil and water fertility, and sparse vegetation growth (and thus poor availability of food, nesting cover and brood cover).

Site 1

Ungulate habitat, particularly for moose, is very limited. Beavers, and possibly mink, utilize various of the streams transecting the ore body, tailings disposal area, and plant site. Waterfowl use Kearl Lake for breeding and for fall staging.

The mining and tailings disposal scheme will have little effect on big game populations, but beavers and waterfowl will be completely eliminated through stream diversion and the draining of Kearl Lake. Waterfowl are likely to be attracted to the large tailings pond, especially during fall migration. Tailings ponds which remain partially open in winter are attractive to spring migrators as well, since little open water is available at that time of year. Waterfowl must be deterred from landing on bitumen-contaminated ponds, particularly in early spring.

Site 2

This area is somewhat better for ungulates (moose) than Site 1. The poplar forests provide better food and cover, and the habitat adjacent to the Muskeg River and Hartley Creek may be used by moose in winter. The area is very good for beaver, and may also support mink and muskrats. Most of the tributaries to the Muskeg River are used by beaver. Waterfowl use is restricted to Kearl Lake, which probably will not be drained, and to the area around the confluence of the Kearl Lake outflow and the Muskeg River. The same comments regarding tailings pond hazards to waterfowl apply here. Offsite habitat mitigation for wildlife should not be necessary.

Site 4

Although the ore body has only moderate ungulate habitat capability, the Firebag River provides good moose wintering habitat. This riverside habitat should be protected from disturbance, as it may also serve as a movement corridor for local populations. The area is not important for furbearers. Waterfowl apparently use the string bog (adjacent to the western mine limit) for spring staging and breeding. This area should be protected from disturbance. Waterfowl deterrent devices must be utilized, particularly because of the proximity of tailings areas to the string bog and the Firebag River.

With respect to impacts on wildlife, the following should be considered:

a. Since waterfowl, especially during migration, are both the most important and the most sensitive wildlife resource, strict protective measures must be utilized. Reduction of the total surface area of tailings ponds, coupled with elimination of any hydrocarbon slick, are key priorities. The area of open water must be reduced during the time of spring (April-May) and fall (September-October) migration, and deterrent devices should be used. Natural and manmade reservoirs used for fresh water, etc., must either be kept completely free of bitumen and toxic chemicals, or be equipped with deterrent devices.

b. Reclamation plans should give high priority to the creation of suitable staging and breeding ponds.

c. Contingency planning must be undertaken to prevent pipeline (and other sources) spills from entering tributaries to the Athabasca River, thereby being transported into the Peace-Athabasca waterfowl breeding areas.

d. Beaver, the mainstay of the local trapping industry, should be protected where possible by minimizing stream diversions, by preventing extreme peak flows, and by conserving natural aspen and willow habitat near streams.

e. Reclamation should encourage the reestablishment of beaver populations by re-introductions, as well as tree species selection.

f. Critical moose habitat (Firebag River valley), and preferred habitat (Muskeg River, Hartley Creek) should be conserved where possible. Habitat adjacent the Firebag River is considered to have especially high regional value.

g. Reclamation plans should encourage moose re-invasion by maximizing small forest openings, by planting to favoured deciduous tree and shrub species, and by creating more diverse terrain conditions.

h. Resource impacts to other furbearers, other ungulates, and smaller mammal species are considered low to moderate, and no special mitigation is recommended.

i. It is not thought that mining will threaten rare or endangered species, but where raptors are encountered, special protective measures should be utilized.

j. Relative impact to upland birds is considered fairly low. No special measures are recommended.

k. Access and hunting should be strictly controlled to minimize human predation.

l. Human waste disposal areas (primarily garbage dumps) should be located far from working or living areas to minimize bear-human interactions.

3.10 LAND USE

Land use capabilities are considered high in the areas of recreation (water-related activities predominantly), wildlife utilization (waterfowl hunting, viewing), and fur trapping. Capabilities for forestry are presently low to moderate, although appropriate permits for harvesting are in place. Capabilities for energy resource utilization (synthetic oil production) rank highest of all. Fishing, big game hunting, urban development, industrial development (except synthetic oil), and agriculture rank low.

Local resources are generally underutilized. Only about 10% of the exploitable increment of furbearers is harvested annually[28]. No active lumbering operations harvest existing commercial timber. There is, however, a Timber Quota A7-Q3) in existence, granting Swanson Lumber Co. Ltd. 9.5 Mfbm annual allowable cut on 120 year rotation. A fuelwood permit also allows 160 Mfbm annual cut in the Fort McKay area, outside of the study area. Limited populations and access account for the nonexistence of the recreation industry. Hunting and fishing activity is also very low.

The process of assigning future land uses prior to reclamation planning must take into consideration land use capabilities and utilization (briefly listed above), but it must also project into the future what manner of demand and utilization can be anticipated. Since utilization of all resources is presently low, there are few demand statistics to provide indications. For these reasons, the following suggestions must be considered somewhat speculative.

It can be supposed that any new community located in the oil sands area (possibly located in the Fort Hills) and intended to serve the oil sands industry will display most of the sociological characteristics of the existing town of Fort McMurray. Its citizens can be expected to show fairly standard patterns of demand for recreation that are typical of other predominantly resource extraction-based communities. These include:

- moderate to high demand for consumptive recreation such as hunting and fishing;
- high demand for motorized forms of recreation such as snowmobiling and use of all-terrain vehicles, 4-wheel drive trucks, and power boats;
- low demand for nature study, hiking, wilderness camping, cross-country skiing, and other non-consumptive wilderness activities;
- moderate to high demand for formal facilities such as campsites, boat launches, cabins, and roadways.

Since it will be mostly local residents using recreation facilities created through reclamation, planning should attempt to satisfy the predicted type of demand. The construction of access roads, formal campsites, and water-oriented recreation facilities presents certain resource use conflicts, however, particularly with regard to protection of sensitive wildlife areas.

Bearing in mind the type of recreation demand anticipated, management of reclaimed land for consumable (i.e., hunted or fished) wildlife species is also of high priority. Creation or enhancement of moose and waterfowl habitats is particularly important.

Although furtrapping is presently a major land use, its underutilization tends to indicate that sufficient demand is not present to give this land use high priority. However, furbearers are essential ecological components, and contribute significantly to aesthetic appreciation. Beavers, particularly, are critical to the maintenance of certain hydrological regimes that in turn make other land uses possible (wetlands for waterfowl, ungulates, and other furbearers). Furbearer habitat should be maintained for this reason.

Commercial forestry is a very high priority, since it may be a key in the future to providing for a balanced, diverse, and stable regional economy. It is doubtful, however, whether the creation of small commercial forests would be of value. It will probably be necessary, when considering creation of commercial-grade timber stands, to attempt management of large tracts of land.

Land use recommendations for reclamation of each of the three development sites being considered in this study follow:

Site 1

- replacement of a water storage area is highly recommended because the loss of Kearl Lake and the often deep muskeg deposits near it will significantly alter regional hydrology. A large post-mining lake formed by the last mined-out pit would be attractive.

- the slope of Muskeg Mountain (including the site of the proposed tailings pond) has sufficient grade to ensure its future superior drainage characteristics, and replacement of this area to commercial forest stands (Pinus sp. or Picea glauca) seems reasonable. The idea has good credibility since it would replace existing commercial stands on the site and would become part of a fairly contiguous commercial timber area (Muskeg Mountain), thus making harvest more economically attractive.

- since recreation demand for hunting and fishing will be high, water-based recreation associated with newly-created wetlands will be possible.

Site 2

- forestry use is only ruled out where local soils, materials handling problems or other technical considerations render it technically impracticable.

- the creation of clear water ponds is a part of all plans. The reduction of slopes and creation of suitable lake margins around clear water ponds will assist regional waterfowl populations.

- local topography favors establishment of upland wildlife habitats. The proximity of the ore body and the tailings pond (which in some schemes is backfilled) to the Muskeg River valley and other good wildlife habitats tends to improve opportunities for regional wildlife habitat improvement. Species to be encouraged would be moose (which already use the Muskeg River valley), beaver, muskrat, and later snowshoe hare, lynx, and mink. Native trapping opportunities would be enhanced, as would consumptive recreation (i.e., big game hunting).

- the proximity of this site to streams with some sport-fishing capability (.e.g, Hartley Creek), indicates that recreation facilities such as campsites, boat launching and rentals, cabins, and the like would be favoured by recreationists. This area seems the best of those being examined in the study area for a recreation "node".

Site 4

Site 4 contains small amounts of commercial-grade timber, is in proximity to ungulate wintering areas and movement corridors (Firebag and Muskeg River valleys) and is near waterfowl staging and breeding areas (McClelland Lake and the String Bog to its southeast). These facts suggest the following:

- commercial forest planting is an acceptable land use for the Improved and Enhanced Levels, when adequate soil and land surfaces are available.

- the area will be frequented by both ungulates (mostly moose) and migrating and breeding waterfowl. To further enhance the amount of habitat available to these species will probably result in good utilization and increased regional populations.

- the establishment of wildlife resources would improve the recreation capability of the area, especially since it is situated alongside the already rather attractive ungulate wintering area, namely the Firebag River valley.

3.11 RECLAMATION EXPERIENCES TO DATE

Practical reclamation experience in the Athabasca oil sands includes revegetation at the Great Canadian Oil Sands mine, and involves primarily tailings dyke stabilization by revegetation. In addition, some reclamation of associated disturbances (diversion channels, powerline rights-of-way, borrow pits, etc.) has been carried out by both G.C.O.S. Ltd. and Syncrude Canada Ltd.; these attempts are mostly experimental, but comprise primarily surface stabilization measures and experimentation. The greater bulk of oil sands reclamation experiences comprise growth chamber and field plot experiments; observations from these have been used by Techman/RC to suggest the success of various reclamation options.

The following consideration attempts first to outline the key problems complicating efforts to revegetate tailings sands and, to a lesser extent, overburden. This is accompanied by a discussion of experimental and analytical work aimed at solving these problems. Finally, concerns specific to reclamation materials and revegetation practices are outlined.

A list of factors limiting or complicating reclamation success includes the following[30,31]:

a. Adverse climate:

- Short growing season;
- Little precipitation;
- Irregular distribution of summer precipitation.

b. Adverse chemical and physical properties of tailings sands:

- Initially high soluble sodium values (toxic) which are soon reduced to non-toxic levels by leaching;
- Unfavorable physical characteristics including low available moisture-holding capacity (although opinions differ depending

on definition and sampling particulars), extremely low cation-exchange capacity (CEC), high erosion potential, and high bulk density;

- Absence of microbiological activity;
- Absence of organic matter;
- Deficient or low values of calcium, potassium, nitrogen, available phosphorus, and zinc; marginally adequate amounts of copper, manganese, and sulphur.

c. Adverse chemical and physical properties of randomly selected (but presumably sandy) overburden materials:

- Bitumen present resulting in toxicity;
- Sometimes high salinity and alkalinity;
- Unfavorable physical properties attributed to spent tailings sands apply here to a varying, but lesser extent due to better moisture retention ability and the presence of minor quantities of clay and plant nutrients;
- Very low organic matter;
- Low nutrient levels.

d. Limited availability of potential soil supplements over some parts of the region, particularly clay and silt sources.

e. General inadequacy of currently used application and tillage equipment for certain reclamation activities.

In combatting the effects of adverse climate, an approach is often taken to select species genetically suited to local climatic conditions. This practice has many merits and should always comprise part of a reclamation program. Climate is not usually a problem in itself however, but lack of precipitation over much of the summer period acts to intensify the effects of inherently low moisture-holding capacity (tailings sands) resulting in drought. Tailings sand in its pure state contains about 1% available moisture[30], or enough to support a grass/legume crop for 3 or 4 days. Results of tests at Syncrude indicate that increasing available moisture to 4% may extend the period to drought to as much as 15 days. To achieve this, 10 cm of 50% clay till, and an unspecified amount of peat is added to the 25 cm tailings sand layer being ameliorated[30]. This requires an estimated 1000 cubic metres of 50% clay till per hectare. This results in an ameliorated or "prepared soil" layer about 0.4 m thick.

Efforts to overcome other problems associated with the poor physical and chemical properties of tailings sands have taken the same approach: amend the qualities of tailings sands to an "acceptable" level through addition of minimum necessary amounts of peat (muskeg) and an overburden material (clay till when available). Although tailings sands are deficient or low in most major nutrients, the problem is more that the cation-exchange capacity is low, and many of the nutrients added are quickly leached beyond the rooting zone. Addition of high clay-content overburden and/or peat are effective in increasing CEC. The CEC of peat ranges from less than 100 meq/100 g to about 200 meq/100 g. Overburden has generally lower values, and these closely follow the clay content of the material. The CEC of till on the Syncrude lease ranges from 3 to 40 meq/100 g, with clay contents of between 10 and 60%. A 1:1 mixture of 50% clay till and tailings sands results in a material with a CEC of 10 meq/100 g. The CEC can be quickly although possibly temporarily increased further by modest additions of peat. For example, 19 kg of peat added to 1 cubic metre of tailings sands results in a mixture with a CEC of 3 meq/100 g (assuming an average CEC of peat is 150 meq/100 g), and an organic matter content of 2%.

Peat is able to add organic matter, increase cation-exchange capacity and available moisture-holding ability, lower bulk density, and maintain pH near neutral. Clay till is apparently the most desirable mineral amendment. It can increase cation-exchange capacity and available moisture-holding ability, reduce free drainage, and improve certain nutrient deficiencies (Ca, K, Zn, Cu, Mn and SO_4-S). Nitrogen and available phosphorus are still inadequate for plant growth and soil materials require fertilization. Considerable variability exists between the chemical and physical qualities of the various till types, and between these and other overburden materials. Details are presented in Table 3.11-1.

The following statements and conclusions from an address to the Canadian Land Reclamation Association[31], although specific to a single set of experiments, represent a good summary of research findings over the last 5 years.

- Properties of low moisture retention, low available plant nutrients, and high erosion potential make the tailings sand a very poor surface to revegetate directly without amendment;

- Although overburden materials are also sandy in nature, the presence of some clay makes them less erodable than tailings sand. In comparison to tailings material, overburdens are very well buffered around their neutral to mildly alkaline pH;

- Peats are generally low in available plant nutrients, especially nitrogen and phosphorus, although levels do vary with the source of the peat;

- Results showed that, as long as optimum moisture and nutrient conditions could be maintained, tailings sand, peat, overburden, lean tar sands and various mixes could all support plant growth;

- The main advantages of peats in reclamation are high water holding characteristics and good cation exchange capability;

- The inoculation of the revegetation surface with micro-organisms from the peat probably represents one of the more important long term effects of soil surface amendments to the tailings and slope;

- The results indicated that although large increases in plant productivity could be induced through addition of nutrients, the area could not be considered self-sustaining if fertilization were discontinued;

- Frequent small additions of fertilizer were more beneficial than larger yearly or biennial applications;

- Electrical conductivity was highest where overburden was added in addition to peat;

- This inability to get roots to penetrate significantly into the tailings sand layers below was considered a serious deficiency;

- Although erosion on test plots was more severe in the overburden-amended plots, in no case could the effect be considered serious where a plant cover had been produced;

- All the different surfaces involved (i.e. tailings sand, peat, overburden, and their mixes) have high water infiltration characteristics (21.6 to 30.5 cm/hr). As a consequence of this high absorptive capacity only between 0.79% and 2.5% of the intercepted rainfall was collected as runoff. The amounts of nutrients lost (in this way) were also small;

- Leaching losses were greater than runoff losses. Between 15.7% and 28.9% of the intercepted rainfall leached below the 30 cm depth (of soil amendment). Nutrient losses were consistently higher in the overburden-amended treatments.

Summaries of results of other revegetation studies follow:

a. Greenhouse experiments indicated that growth of barley on media comprised of tailings sands, overburden and peat, was primarily governed by fertilizer addition but peat and overburden addition to tailings sands also resulted in growth increase[33].

Table 3.11-1

SUMMARY OF SELECTED CHEMICAL AND PHYSICAL PROPERTIES OF SOILS
AND OTHER RELATED MATERIALS FROM SYNCRUDE AND G.C.O.S. LEASES
DATA INDICATE RANGES OF VALUES OBTAINED FROM A LITERATURE REVIEW[31, 32]

Materials	pH	Cond. mmho per cm	NH_4^+ ppm	NO_3^- ppm	Org. C %	Organic Matter %	P ppm	K ppm	Na ppm	Ca ppm	$CaCO_3$ equiv. %	Total N %	Sulfate S ppm	Exchange complex, me/100 g: CEC	K	Ca	Na	Mg	Particle size distribution, %: Sand	Silt	Clay	% moisture at field capacity (1/3 bar.)
Tailings Sands	6.1-9.7	0.55	1.06	0.5-1.3	0.20	-	1.3-3.5	4.0-12.5	75	75	-	0.005	11.4	2.9	0.26	0.25	0.43	0.48	96.6	1.0	2.4	14
Syncrude Overburden	6.2-7.4	2.46	0.18	0 -1.8	1.24	-	2.3-18	32.5-47	65	1870	4.48	0.019	278.4	3.9	0.64	12.00	0.43	1.70	64.8	18.4	16.8	18
GCOS Overburden	6.3-7.8	1.51	trace	0 -4.8	1.89	-	0.6-9.8	33 -205	75	1555	3.53	0.024	232.0	3.9	1.30	13.25	0.43-2.8	2.80	59.9	21.0	19.1	18
Agricultural Soil (Malmo)	6.0-6.1	0.52	3.37	15.96	6.37	10.96	11.5	150.0	85	4950	-	0.450	9.5	37.8	3.80	31.75	0.43-0.53	4.80	38.4	45.2	16.4	30
Heavy Oil Sands (GCOS mine site)	5.70	0.19	trace	trace	10.70	-	2.5	12.5	80	90	-	0.049	48.0	2.9	0.64	0.25	0.43	0.48	68.0	30.4	1.6	14*
Lean Oil Sands (air-dried)	6.4-6.8	0.15	trace	0 -1.8	6.12	-	2.0-11	7.5-7.8	45	145	-	0.037	6.3	2.9	0.26	5.00	0.22	0.35	96.2	0.6	3.2	14
Peat (GCOS, drained)	4.1-6.7	0.16	trace	0 -53.8	56.75	97.60	0 -4.0	34-298	230	3250	-	0.863	460.0	127.1	3.80	33.25	0.65	9.04	-	-	-	250**
Peat (Syncrude)	4.7-6.3	-	-	3.5	-	-	0.6	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Soil (Ae, Bf and C horizons from aeolian ridge)	6.0-6.6	-	-	1.5	-	-	3.5-16	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-

* 5% field moist
** on an air dry basis

b. Growth chamber tests compared the early development of twenty-five species of grasses and legumes on five different soil types (organic peat, clay-loam, sand, oil sands and tailings sands). Clear interpretations of study results were not provided. However, considerable variation in plant growth showed the importance of plant selection for revegetation[34].

c. Growth chamber experiments compared plant growth on overburden materials, tailings sands, peat and heavy and lean oil sands materials collected from the Syncrude and G.C.O.S. leases[32]. The following conclusions were provided:

- as compared with a good agricultural soil (Malmo), growth of plants on the oil sand-bearing materials was poor.
- dry matter yields on tailings sands:overburden (1:1) mixes and tailings sands:peat:overburden (2:1:1) mixes were generally better than on tailings sands alone and tailings sands:peat (1:1) mixes for various levels of nutrient application.
- growth on thoroughly mixed tailings sands and peat appeared to be better than growth where the peat was placed on the tailings without being mixed.

d. In recent revegetation experiments on tailings sands slopes[31], the amounts of fertilizer added in the first year varied between 80 kg-N, 35 kg-P, 75 kg-K, and 20 kg-S per hectare, and 300 kg-N, 80 kg-P, 300 kg-K, and 40 kg-S per hectare. Preliminary results indicate that amounts added should be at a minimum 80 kg-N, 20 kg-P and 80 kg-K per hectare per year to produce adequate results.

e. A recent study[35] examined revegetation on borrow areas developed for the extraction of granular materials and disposal areas created for discarding earthen spoils material. These enabled a comparison of growth capability of "clay" and "gravel" tills, as well as peat, although study areas were generally heterogenous 'soil' mixes. Heavy applications of fertilizer were provided.

Plant cover and standing crop were highest on level peat areas and poorest on gravel slopes. Plant productivity was superior on level clay sites compared to steep clay slopes.

The following conclusions regarding soil reclamation have been drawn by the Consultants, and are reflected in the reclamation plans developed by this study:

- All materials available in the Muskeg River area that might be used as components of prepared soil have some major physical and chemical qualities that would act as limitations to good plant growth. The objective, which could only be definitively met at the operational level, is to combine the materials in the most advantageous proportions.

- Tailings sands as described in this study have no qualities that would act to improve either the physical or the chemical characteristics of the future growth medium. Only where amendments are dense and impermeable could tailings sands act to improve texture. Simply stated, tailings sand would appear to dilute the positive qualities of whatever other soil amendments were used.

- Present experience shows that a relatively thin layer of muskeg (15 cm) with intensive fertilization and maintenance appears adequate for short-term erosion control. Long-term erosion control providing greater choice in plant species for revegetation will require the addition of suitable overburden materials.

- The fact that the majority of rooting stops at the tailings sand and amended tailings sand interface indicates that only shallow rooting plants can be grown on such a thin soil veneer. Deeper-rooting grasses and trees must be provided with deeper soil (prepared soil) to achieve optimum growth and long term stability. Specific depths of soil to be used must ultimatey depend on site specific criteria for end land use, physical and chemical properties, productivity and erosion control expectations and cost;

- A relatively deep prepared soil is necessary to prevent leaching of water and nutrients below the amended layer and thus below the depth of root penetration.

A number of additional reclamation problems with lesser relevance to the cost generation objective of this project have not been addressed in this review:

- Predation by rodents (mostly microtines) on tree seedlings, particularly in areas previously seeded to grasses (for erosion control);

- Seepage from tailings dykes largely caused by high phreatic surfaces that causes both erosion on the dykes and increased area salinity;

- Low survival of tree and shrub seedlings due to competition from dense grass mats, a consequence of inconsistent reclamation objectives;

- Definition of a period in time when fertilization is no longer required to sustain vegetation. It is worth emphasizing that it has not yet been demonstrated that maintenance of oil sands reclamation (particularly dyke slopes) can be discontinued without loss of the erosion control mat, at least localized dyke failure (sloughing) and/or severe loss of productivity. The problem is complex, however, and much detailed work is presently underway.

These conclusions, and suggestions made in subsequent chapters, have been made by Techman Ltd. and Rheinbraun Consulting - GmbH staff after careful consideration of operational problems and limited experimental results available at the time of writing.

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4.0 DEFINITION OF LEVELS OF RECLAMATION

4.1 GENERAL OBJECTIVES OF OIL SANDS RECLAMATION

The reclamation of the mined-out areas of the Athabasca oil sands region challenges present and potential mine operators because of a combination of adverse climatological and meteorological conditions, limited natural soil depths, and the complexity, overall, of the mining and tailings disposal operations (See: Section 3.1, Meteorology and Climatology; Section 3.6, Soils; Section 3.11, Reclamation Experiences to Date; Section 5.3, Tailings Disposal Techniques; Section 5.4, Reclamation Techniques, as well as mine plans described in Sections 7.1, 8.1 and 9.1, General Design Concepts).

Mining of oil sands deposits for the production of synthetic crude oil involves a large-scale disturbance of the landscape. Immediate and progressive reclamation of the disturbed area avoids both the long-term destruction of the typical landscape and the high cost of reclaiming an area years after its disturbance. Therefore, one objective of mine planning as well as reclamation planning should be to minimize the out-of-pit area disturbance, and to maximize the use of in-pit areas for the deposition of overburden and tailings. Mining and reclamation activities should be carried out simultaneously in separate areas of the mine, coordinated by a predetermined schedule of development activities.

It is believed that the costs incurred by the mine in achieving selected standards of reclamation may be much greater than the actual value of the restored landscape. Nonetheless, successful and permanent reclamation of the mined-out areas should be highly valued since it will determine the course of longterm development of the McMurray region.

There are many possible degrees of reclamation of an oil sands mine or region. The terms of reference of the present study required that the possibilities be ranked "minimum", "improved" and "enhanced". The detailed definition of each of the three levels of reclamation is determined by the following set of influencing factors: time, overburden, tailings pond, landform, land use, and tailings characteristics. As a result of detailed investigations regarding mine and reclamation planning of three potential oil sands mining sites, the relative significance of these factors became progressively more evident; consequently, the definition of the levels of reclamation was evolved rather than arbitrarily predetermined. The practical elements of materials handling in terms of reclamation potentials are given prime consideration.

The terms "minimum", "improved" and "enhanced" are not to be confused with the absolute quality or desirability of the applied reclamation activity. Rather, the terms generally reflect advances in tailings disposal and mine operating techniques as well as extraction technology which will lead to a progressive improvement in reclamation potential of any given mine site. The advantages realized by this classification result from sophistication in the applied materials handling schemes, or are the direct result of improvements made feasible by advanced extraction technology. Table 4.1-1, Definition of Levels of Reclamation, provides a concise summary of the attributes of each level of reclamation.

4.2 MINIMUM LEVEL OF RECLAMATION

The Minimum Level of Reclamation is defined as follows:

Time - The time schedule of the reclamation activities is determined by the site-specific conditions found at the mine site. The development of an efficient mine plan is paramount, the type of reclamation generally being the result of the requirements of the mine plan rather than a major objective.

Prepared Soil - A prepared soil (muskeg-overburden mixture) of 0.4 m thickness is spread over "dry" reclaimable surfaces. The actual depth ultimately relates to the quality of the soil, the quality being sufficient for the minimal vegetation demands featured at this level. A prepared soil layer is created on the reclamation site by trucking and spreading 0.2 m muskeg previously removed from

Table 4.1-1

DEFINITION OF LEVELS OF RECLAMATION

	MINIMUM	IMPROVED	ENHANCED
TIME ELEMENT	Reclamation work done at rates which make overall mining most economical.	Reclamation work will have direct influence on mine design and operating costs.	The reclamation objectives are dominant in mine design and determination of operating cost.
MUSKEG	Muskeg for prepared soil will be selected from the routine muskeg stripping operation.	Muskeg for prepared soil is obtained by selection from the routine muskeg stripping operation.	Muskeg for prepared soil is obtained from especially selected sources known as "muskeg mines".
OVERBURDEN	Selected with minimal effort as material becomes available in the course of over-burden removal.	Selective salvage of greater quantities of overburden. Larger quantities may result in some rescheduling of overburden stripping operation.	Selected to produce optimal soils by blending. Selection capability highest and least disruptive to general over-burden removal.
PREPARED SOIL	Availability a by-product of mining, least selective, uniformity of minimal importance, 0.6 m average depth.	Selected with consi-deration to economics of selection, coarsely blended, 1.0 m average depth.	Carefully selected for final land use and uniformly blended, 1.0 m average depth.
PONDS	Ponds reduced to minimum area by physical dimensioning.	Ponds reduced to mini-mum area by physical dimensioning as well as treatment of sludge.	No ponds remaining.
DRAINAGE	Minimum drainage for environmental protection.	Optimal drainage for partial land use.	Drainage for optimal regional land use.
STABILIZATION	All slopes geotech-nically stable and erosion control cover only.	All slopes reduced to assist revegetation.	All slopes biologically stable for all planned land use.
LANDFORM	No attempt at making compatible landform. Erosion control considerations only.	Solutions compatible with planned land use otherwise erosion control only.	Final landscape largely determined by land use.
LAND USE	Will provide environ-mental protection to surrounding areas.	Most economical designated land use.	Create maximum diversity, desirable recreation and economic land uses, may improve surrounding environment.
TAILINGS	Wet	Modified (dewatered)	Dry

the mining area and stored alongside the pits, and 0.2 m overburden also purposely stockpiled near the pits or in controlled areas of waste dumps. Following spreading and mixing by dozer, the mixture is cultivated to a depth of 0.6 m to incorporate an additional 0.2 m of tailings sand. The final ratio of muskeg to overburden and sand should be 1:2. This should permit the establishment of noncommercial forest. On waste dumps, the 0.2 m sand layer is replaced by 0.2 m overburden for a total of 0.4 m overburden. Therefore prepared soil on waste dumps requires only trucking of muskeg.

<u>Tailings Pond</u> - The sizes and shapes of the tailings ponds depend both on the tailings disposal technology employed, and on the materials handling schedule of the mine. Rehandling of sludge from pond to pond to reduce final wet surface area is desirable. Reclamation of dyke slopes and near-level sanded-in ponds is done as material becomes available from the mine's overburden stripping operation.

<u>Landform</u> - The final landform reflects the necessity of efficient materials handling in the mine. Slope stability and controlled surface drainage are achieved. The final remaining pit is rapidly filled with water (pumping from the Athabasca River, if necessary) to minimize the deleterious effects of groundwater seepage on water quality of the remaining lake.

<u>Land Use</u> - To adapt the reclaimed area to the existing land use, mainly noncommercial forestry, reclamation is deemed desirable. Grass-legume covering of the area is permissible only in areas where it is necessary to avoid erosion where forest reclamation is not possible for various reasons, or where wildlife-browsing areas are

desired. Even though the grass-legume planting of the area is not used as standard for the Minimum Level, it could be used over a considerable area where it forms part of the land use plan.

Tailings - Wet tailings limit reclamation to the slopes and sanded-in areas of the tailings ponds. Areally large and currently unreclaimable wet pond surfaces remain, although the ponds are dimensioned so as to cover a minimum area.

4.3 IMPROVED LEVEL OF RECLAMATION

The Improved Level of Reclamation is defined as follows:

Time - The requirements for a more demanding type of reclamation compared to that at the Minimum Level are considered during mine planning. The availability of sufficient reclamation soil (overburden and muskeg) may affect the materials handling schedule of the mine.

Prepared Soil - Prepared soil meets the requirements of self-sustaining noncommercial forestry reclamation, combined with a versatile choice of trees. The thickness of the surface layer is 1 m and in some cases more, depending on the composition of the soil and the underlying materials. Blending considerations are important and, combined with quantity requirements, begin to affect the equipment selection and mining method. Initial blending of suitable overburden material and muskeg is achieved by building strategically-located layered stockpiles. Such stockpiles are layered with 1 m muskeg followed by 2 m overburden. A dozer operating technique employing ripping, cross-cutting, and blading further mixes the soil prior to loading with front-end loaders onto trucks. The materials placed into these stockpiles are somewhat selected, i.e., consideration to quality is given. The stockpiles are constructed during the winter months, overburden being acquired either at the face or at the spreader, and muskeg being selected from the prestripping operation. The stockpiles are mined as reclamation material is required. A muskeg-to-overburden ratio of 1:2 is achieved. Transporting to the reclamation site is done using large off-highway trucks. Spreading and mixing is done by dozer, followed by cultivation with deep penetrating plowing wherever possible.

Tailings Pond - The sizes and shapes of the tailings ponds depend on the site-specific conditions at the ore body, tailings disposal technology employed, and mining method. A further reduction of wet tailings pond surface area is made by dewatering of sludge in conjunction with bitumen recovery. The opportunity to chemically treat all or partial quantities of sludge is afforded, creating a potential for reclamation of the final sludge pond.

Landform - Disturbed surfaces are shaped to form a variety of gently graded areas. A gently sloping surface is achieved on the sanded-in ponds. Drainage is sufficient to allow for dry-ground tree species, and surface runoff is controlled to even out the water retention capabilities as well as contribute to runoff clarification. The final remaining pit is rapidly filled with surface water (pumping from the Athabasca River, if necessary) to minimize the deleterious effects of groundwater seepage on the water quality of the remaining lake.

Land Use - In order to adapt the resulting landscape to its present surroundings, mainly forest reclamation is required. Many different types of trees are planted. Areas that are not suitable for deep-rooting trees because of the insufficient thickness and quality of the prepared soil are planted with a grass-legume erosion control mat and shrubs.

Tailings - Tailings sludge pumped from the ponds to the bitumen-recovery plant is partially dewatered by chemical and mechanical processes. This simultaneously reduces the overall disturbed area and increases the dry areas available for reclamation purposes.

4.4 ENHANCED LEVEL OF RECLAMATION

The Enhanced Level of Reclamation is defined as follows:

Time - When a dry tailings product is generated, the dumping conditions are similar to those of mines where granular overburden is handled. At this level, there is maximum potential for the beneficial placement of dry tailings. Reclamation may be a key consideration for the mine. In this case, the equipment is more advantageously and efficiently employed to produce an optimum landform and so maximize prepared soil utilization, as compared with the two previous reclamation levels. In addition, the requirement for only limited containment structures allows greater flexibility in mine scheduling.

Prepared Soil - Thickness and quality of the surface layer of soil is varied according to the potential utilization of the reclaimed area. As a rule, improving the quality makes a difference to the utilization potential of the reclaimed areas. The target depth of prepared soil is 1 m with a ratio of 1 part muskeg to 2 parts overburden. The source of muskeg is either in the area of future mining development or off the mine site itself. Hydraulically-mined muskeg is dewatered and blended with carefully selected overburden from the mine. A stacker forms alternating layers of muskeg and overburden. The components are mixed into prepared soil by the buckets of a small bucket wheel reclaimer and transported annually via conveyor to predetermined temporary field storage sites. From these sites, trucks transport the material to the reclamation site for spreading. Additional field blending and deep cultivation are required only where special conditions exist that have resulted in segregation, over-compaction, or similar difficulties.

Tailings Pond - Tailings disposal resembles normal outside and inside dumps, which are readily made compatible with the landscape. Few, if any, ponds are needed.

Landform - The relief of the areas used for mining purposes (outside dumps, inside dumps, and a minimum of tailings ponds) is modelled according to the surrounding landscape. Transitions from the natural to the artificial landscape are gradual. The final remaining pit is rapidly filled with surface water (pumped from the Athabasca River, if necessary) to minimize the deleterious effects of groundwater seepage on the water quality of the remaining lake.

Land Use - The proportion of good forestry soils is increased compared to the original natural conditions. Forestry reclamation results in the creation of a forest with commercial capability. Recreational land uses are also incorporated where desirable.

Tailings - The dry tailings provide the opportunity for reclaiming the entire mining area. Special treatment of the tailings area does not appear to be necessary, as long as sufficient capping of overburden and prepared soil is provided.

5.0 MINE DEVELOPMENT CRITERIA

5.1 GEOLOGICAL AND ENGINEERING CRITERIA FOR MINE DESIGN

5.1.1 MINING GEOLOGY

The regional geology of the Athabasca oil sands area has been discussed in Section 3.2, Regional Geology. In the course of this study, a set of east-west trending geological cross sections was prepared at 3 - km intervals through Ore Bodies No. 1 and 2. A single cross section was drawn through Ore Body No. 4. The cross-sections are presented under separate cover (Volume II - Drawings) and are numbered as follows:

Ore Body No. 1, Cross Section	B_1 - B_1', Dwg. No. BR 22915-06-00
	C_1 - C_1', Dwg. No. BR 22915-07-00
	D_1 - D_1', Dwg. No. BR 22915-08-00
	E_1 - E_1', Dwg. No. BR 22915-09-00
Ore Body No. 2, Cross Section	B_2 - B_2', Dwg. No. BR 22915-10-00
	C_2 - C_2', Dwg. No. BR 22915-11-00
	D_2 - D_2', Dwg. No. BR 22915-12-00
	E_2 - E_2', Dwg. No. BR 22915-13-00
Ore Body No. 4, Cross Section	B_4 - B_4', Dwg. No. BR 22915-14-00

For the location of these cross sections, consult Drawing No. B2290-01-00, Ore Body Locations Within Regional Mining Area.

The cross sections show the position and continuity of overburden, pay zone, and reject. As well, some indication of formations forming the pit floor are provided. These simplified cross sections provide an overall impression of the geologic character of the ore bodies selected for this study.

The characteristics of Ore Body No. 1 are as follows (for reference, see Table 2.8-1, Ore Body No. 1A at GRAMT $\geq$ 400.00 and R-Factor $\geq$ 0.600), and Table 2.8-2, Ore Body No. 1B at GRAMT $\geq$ 400.00 and R-Factor $\geq$ 0.600):

a. The overburden consists of glacial drift of Pleistocene age, silts and clays of the Clearwater Formation, and low-grade oil sands (top reject) above the uppermost pay zone. The overburden thickness for Orebody No. 1A averages about 21.1 m, but ranges from 3 to 40 m. For Ore Body No. 1B, average overburden thickness is about 23.0 m, and ranges from 14 to 64 m.

b. The mean thickness of economic oil sands (net pay zone thickness) is about 57.1 m for Ore Body No. 1A, but ranges from 31.6 to 75 m. For Ore Body No. 1B, the mean thickness is about 50.5 m and ranges from 30.1 to 67 m.

c. For Ore Body No. 1A, the in situ bitumen grade averages 11.66% (11.48% for Ore Body No. 1B). The oil impregnation within the pay zones is not homogeneous, but varies in grade from 9.86% to about 13.46% (9.66% to 13.30% for No. 1B). The fines content of the pay zone averages about 13.36% (14.09% for 1B), but varies from about 6.16% to about 20.56% (6.77 to 21.41% for No. 1B).

d. In Orebody No. 1B, a predominant band of centre reject is apparent. On the average, at least one band of centre reject may be expected on any particular mining face. Top reject thickens to the southeast and bottom reject material is intermittent or discontinuous.

e. The Devonian paleotopographic surface shows undulations. A water-bearing aquifer sand underlies the entire ore body with thickening in the paleotopographic lows.

The characteristics of Ore Body No. 2 are as follows (see Table 2.8-3, Ore Body No. 2 at GRAMT $\geq$ 400.00 and R-Factor $\geq$ 0.600):

a. The overburden material which overlies Ore Body No. 2 is similar to that described for No. 1. The mean overburden thickness is about 11.3 m, but ranges from 0 to 36 m.

b. The net pay zone thickness (+5% oil sands) averages about 44.6 m, but varies from 19.0 to 69.5 m.

c. The in situ bitumen saturation for Ore Body No. 2 averages 11.62%. The oil impregnation within the economic zones is not homogeneous, and varies in grade from 9.02% to about 14.22%. The mean fines content of the pay zone is 13.52%, but ranges from about 3.12% to 23.92%.

d. On the north end of the ore body, there is usually one continuous band of centre reject present. Statistically, at least one band of centre reject may be expected on a mining face.

e. The paleotopographic surface of the Devonian limestone formation is undulating. A thick water-bearing aquifer sand underlies the entire ore body.

The characteristics of Ore Body No. 4 are as follows (see Table 2.8-4, Ore Body No. 4 at GRAMT $\geq$ 500.00 and R-Factor $\geq$ 0.600):

a. The overburden material which overlies the +5% oil sands of Ore Body No. 4 is similar to that described for Ore Body No. 1. The average thickness of overburden is about 18.4 m, but varies from 4 to 33 m. The overburden thickens to the east.

b. The net pay zone thickness (+5% oil sands) averages about 46.7 m, but ranges from 29.2 to 59 m.

c. The average in situ bitumen grade for Ore Body No. 4 is 12.02%. The hydrocarbon impregnation within the pay zone is not homogeneous, and varies in grade from 10.70% to about 13.34%. The fines content of the pay zones averages 11.91%, but varies from about 6.61% to about 17.21%.

d. Intermittent bands of centre reject appear between pay zones of Ore Body No. 4. Statistically, at least one band of centre reject may be expected on any particular mining face. Top reject is intermittent. Bottom reject is apparent under the eastern half of the ore body, but disappears towards the west.

e. The paleotopographic surface of the Devonian Waterways Formation is fairly flat. A very thick, massive aquifer sand underlies the entire ore body.

5.1.2 STATISTICAL ANALYSIS OF CENTRE REJECT

The quantity and grade of mineable oil sands have been estimated by Techman/RC for Ore Bodies No. 1, 2 and 4 using selective mining criteria based on a cutoff grade of 5% bitumen content by weight. Material with 5% bitumen or more (+5% material) would be mined and transported to the extraction plant, while material containing less than 5% bitumen would generally be rejected. The criteria applied are:

Ore:

All oil sands zones $\geq$ 5% bitumen and $\geq$ 1.52 m zone thickness (+5% material and $\leq$ 5% bitumen and < 1.52 m zone thickness (-5% material)

Reject:

All oil sands zones < 5% bitumen and $\geq$ 1.52 m zone thickness (-5% material) and > 5% bitumen and < 1.52 m zone thickness (+5% material)

The ERCB uses these criteria to delineate mineable oil sands reserves. In general, the -5% material above the mineable oil sands feed section of the McMurray Formation (top reject) would be rejected with the overburden, while the -5% material below the feed section (bottom reject) would be left in place. ERCB geologists are of the opinion that lean oil sands contained within the plant feed section could be rejected by selective mining whenever this lean zone is 1.52 m or more in thickness. Lean oil sands less than 1.52 m in thickness would be included in the plant feed.

The mining methods examined by Techman/RC in this study required that consideration be given to the selective digging capability of the prime excavators, especially with respect to the handling of centre reject. The draglines are not able to achieve the 1.52 m cut-off used in the reserve calculation. Instead, a 3.66 m cut-off was assumed to be attainable, and the quantities of plant feed and waste were adjusted accordingly. Such adjustments required a statistical analysis on centre reject data to:

- predict the number of occurrences of centre reject that may be expected in each ore body.
- determine a thickness frequency distribution of centre reject that may be encountered during mining operations.
- determine the overall average thickness of centre reject for each ore body.

The 3.66 m cut-off is considered to be the minimum practical thickness that can be excavated, based on the geometry of the slope being excavated, the average position of the zone, the size of bucket, and the filling characteristics of the buckets likely to be employed in oil sands mining generations. Separation of thinner zones would be very difficult without a great reduction in dragline productivity.

The method used by Techman/RC for the centre reject analysis was as follows:

a. Centre reject information was gathered from drill holes within each ore body or in proximity to the edge (less than 250 m). It was assumed that with the ore body selection and definition based on R-Factor $\geq$ 0.600 and GRAMT $\geq$ 400.00, the 'edge effects' would be minimized. A visual inspection of the geological cross sections through each ore body showed that limiting the gathering of centre reject data to drill holes within the delineated ore bodies was a valid approach.

   The information extracted from ERCB log interpretations showed:

i. number of occurrences of centre reject for each drill hole (ranging from 0 to 3).

ii. thickness of each occurrence.

iii. total centre reject thickness for each drill hole.

b. For Ore Bodies No. 1, 2 and 4, the drill hole density was assumed to be adequate (less than 1,524 m between drill holes) and the drill hole distribution was even. As a result, the ore bodies were treated as one large polygon, without the benefit of weighting each drill hole by its area of influence.

c. Frequency distribution and mean centre reject thickness were calculated for Ore Body No. 2 as follows:

Frequency* Distribution	No. of Occurrences	% of Total	Cumulative %
5 - 9.99 ft.	4	17.39	17.39
10 - 14.99 ft.	4	17.39	34.78
15 - 19.99 ft.	1	4.35	39.13
20 - 24.99 ft.	3	13.04	52.17
25 - 29.99 ft.	5	21.74	73.91
30 - 34.99 ft.	1	4.35	78.26
35 - 39.99 ft.	1	4.35	82.61
40 - 44.99 ft.	1	4.35	86.96
45 - 49.99 ft.	1	4.35	91.31
50 - 54.99 ft.	2	8.69	100.00

* Data collected from non-metric drilling information.

Average CR thickness for Ore Body No. 2 = 9.07 m
(averaging thicknesses for drill holes inside the ore body)
Average number of CR occurrences per drill hole = 1.15

The above results compare favourably with those calculated using the digitized data base that was later used for mine design and mass scheduling. An average CR thickness for Ore Body No. 2 of 10.63 m was obtained by analysis of the data base.

d. Frequency distribution and mean centre reject thickness were determined for Ore Body No. 4 as follows:

Frequency* Distribution	No. of Occurrences	% of Total	Cumulative %
5 - 9.99 ft.	5	33.33	33.33
10 - 14.99 ft.	5	33.33	66.66
15 - 19.99 ft.	1	6.67	73.33
20 - 24.99 ft.	1	6.67	80.00
25 - 29.99 ft.	0	0	80.00
30 - 34.99 ft.	0	0	80.00
35 - 39.99 ft.	0	0	80.00
40 - 44.99 ft.	0	0	80.00
45 - 49.99 ft.	1	6.67	86.67
50 - 54.99 ft.	1	6.67	93.34
55 - 59.99 ft.	0	0	93.34
60 - 64.99 ft.	0	0	93.34
65 - 69.99 ft.	0	0	93.34
70 - 74.99 ft.	1	6.66	100.00

* Data collected from non-metric drilling information.

Average CR thickness for Ore Body No. 4 = 9.67 m
(averaging thicknesses for drill holes inside the ore body)
Average number of CR occurrences per drill hole = 1.67

Average CR thickness using the computerized data base was calculated to be 8.13 m. As before, good agreement exists between the methods of calculation, confirming the ability of the computerized mine model to assess the overall characteristics of prevailing centre reject.

It should be noted that geophysical log data were not complete for a large number of drill holes within Ore Body No. 1. About 50% of the holes showed only the total centre reject thickness with no indication given of the number of occurrences or their individual thicknesses. Because an analysis of the remaining 50% of the drill holes could prove misleading, Techman/RC decided that the results of the statistical analysis for Ore Body No. 2 could be used for Ore Body No. 1, since the two ore bodies are joined at a point south of Kearl Lake, and indications are that the mining and economic geology of these ore bodies are similar.

The following interpretation has been made:

a. The statistical data suggest that the number of CR occurrences in Ore Body No. 2 ranges from 0 to 3, but the average is 1.15. A visual inspection of geological cross sections through Ore Bodies No. 1 and No. 2 indicates that only one CR occurrence is present within the McMurray Formation. It is concluded that during mining operations in Ore Bodies No. 1 and 2, at least one band of centre reject may be expected on any particular mining face.

b. The thickness of one particular band of centre reject for Ore Bodies No. 1 and 2 ranges from 1.5 to 16.8 m. The mean total CR thickness is 9.12 m for Ore Body No. 1, 10.40 m for Ore Body No. 1 B, and 9.07 m for Ore Body No. 2. A comparison of similar data derived by digitizing shows fairly close agreement.

c. Approximately 35% of the occurrences of centre reject in Ore Body No. 2 are between 1.5 and 4.6 m thick. For purposes of mine planning for large-size draglines in the one-bench dragline scheme, it has been assumed that occurrences of centre reject less than 3.66 m in thickness would be mined as ore-grade material. Efforts to selectively mine these bands of lean oil sands would be time-consuming and non-productive. An estimate has been made of the proportion of centre reject less than 3.66 m thick for Ore Body No. 2, based on the following calculation:

Percentage of CR less than 12 ft. thick is equal to percentage of CR between 5 and 9.9 ft. thick plus percentage of CR between 10 and 12 ft. thick,* or

$$17.39\% + \frac{2\text{ ft.*}}{4.99\text{ ft.}} \times 17.39\%\text{ , i.e. }24.36\%$$

Say 24% for Ore Bodies No. 1 and 2.

The effect of including these lean oil sands with pay zone material in the one-bench dragline scheme is discussed in detail in Subsection 5.1.3, which deals with the effects of selective mining.

d. The statistical data show that the number of centre reject occurrences in any one drill hole in Ore Body No. 4 ranges from 0 to 3, and the average is 1.67. The geological cross section through Ore Body No. 4 shows minor, discontinuous bands of centre reject within the oil-bearing formation. It is concluded that one and possibly two bands of CR may be expected on any particular mining face during dragline mining operations.

e. The thickness of a particular band of centre reject from a drill hole in Ore Body No. 4 ranges from 1.5 to 22.9 m. A total average CR thickness of about 10 m is expected.

f. For Ore Body No. 4, approximately 67% of the occurrences of centre reject are between 1.5 and 4.6 m thick. This is substantially more than that predicted for Ore Bodies No. 1 and 2, and would mean that much more lean oil sands would be mined as ore. Techman/RC has estimated the proportion of centre reject in the one-bench dragline scheme to be less than 3.66 m thick by the following calculation:

Percentage of CR less than 12 ft. thick is equal to

$$33.33\% + \frac{2\ \text{ft.}^*}{4.99\ \text{ft.}} \times 33.33\%\ ,\ \text{i.e. } 46.69\%$$

Say 47% for Ore Body No. 4

*Calculation based on non-metric drilling information.

5.1.3 EFFECTS OF SELECTIVE MINING

With higher-production mining equipment such as draglines and bucket wheel excavators, the mining recovery seldom approaches one hundred percent. Deductions are usually made from in-place mineable reserves to take into account factors such as mining loss and mining dilution. Additional adjustments must be made to reflect the overall ability of the prime excavator to work within the range of constraints imposed by the minimum and maximum ore thickness and general variability of ore found within the mine. This section discusses the effects of selective mining for three different oil sands mining schemes: one and two-bench dragline schemes, and a bucket wheel excavator scheme.

For a dragline mining operation, the separation of reject material from +5% oil sands is primarily dependent on the ability of the dragline operator to see the bucket while digging, and on the operator's ability to distinguish the reject material from the pay zone on the basis of color or perhaps diggability. A considerable amount of training and skill are required before operators can make this distinction with precision. Under less than ideal conditions, such as digging at depth and during night operation when the boom lights create shadows, it is questionable whether the visual distinction between +5% grade oil sands and reject material is possible. To maximize oil sands recovery, a non-selective mode of operation might be necessary during periods of marginal visibility.

In a bucket wheel excavator mining operation, the operator is much closer to the mining face, and has a much greater opportunity to distinguish between ore and waste. As a result, mining selectivity and plant feed quality should be improved over those attainable in a dragline mining operation. Selective mining with any equipment, it must be remembered, has a negative effect on overall productivity, and this effect must be balanced against any possible benefits.

Each pay zone/reject interface where selective mining is to be practiced is a potential source of dilution and mining loss. These interfaces are present at:

a. The base of the top reject;
b. The top and bottom of centre reject zones;
c. The top of the bottom reject.

The effects of selective mining for three different oil sands mining schemes for Ore Bodies No. 1, 2 and 4 are discussed as follows:

<u>CASE I: Techman/RC One-Bench Dragline Scheme</u>

<u>Assumptions</u>

a. Mining to full depth with large-size draglines, as illustrated in Figure 5.1.3-1, Oil Sands Mining Schemes.

b. Overburden stripping done with bucket wheel excavator to the base of the top-reject zone or top of the ore zone.

c. Centre reject less than 3.66 m in thickness mined as ore. A percentage of the total CR is added to the pay zone material as per the statistical analysis in Subsection 5.1.2. All reject material is assumed to have a bitumen grade of 3.33%.

d. Mining loss comes from the TR/TOP PZ and CR/BOTTOM PZ contacts. Dilution comes from the TOP PZ/CR and BOTTOM PZ/BR contacts.

e. The mean PZ thickness for Ore Bodies No. 1, 2 and 4 is about 49 m. Therefore mining loss and dilution may be calculated as follows:

$$\text{Mining Loss} = \frac{0.5\text{ m} + 1.0\text{ m}}{49\text{ m}} \times 100 = 3.06\%$$

$$\text{Dilution} = \frac{1.0\ \text{m} + 1.5\ \text{m}}{49\ \text{m}} \times 100 = 5.10\%$$

Taking into account nighttime visibility and other negative operating factors, mining loss and dilution are increased to 6% and 10%, respectively.

CASE II: Techman/RC Two-Bench Dragline Scheme

Assumptions:

a. Mining to full depth with large-size draglines on two benches. Each machine mines approximately half of the plant feed section, i.e., 1/2(Net PZ thickness + CR thickness) as illustrated in Figure 5.1.3-1.

b. Overburden stripping done with bucket wheel excavators to the top of PZ.

c. Every attempt is made to selectively reject lean oil sands material between pay zones.

d. Mining loss comes from the TR/TOP PZ and CR/BOTTOM PZ contacts. Dilution comes from the TOP PZ/CR and BOTTOM PZ/BR contacts.

e. The mean PZ thickness for Ore Bodies No. 1, 2 and 4 is about 49 m. Therefore mining loss and dilution may be calculated as follows:

$$\text{Mining Loss} = \frac{0.5\ \text{m} + 0.5.\ \text{m}}{49\ \text{m}} \times 100 = 2.04\%$$

$$\text{Dilution} = \frac{1.0\ \text{m} + 1.0\ \text{m}}{49\ \text{m}} \times 100 = 4.08\%$$

Taking into account nighttime visibility and other negative operating factors, mining loss and dilution are increased to 4% and 8%, respectively.

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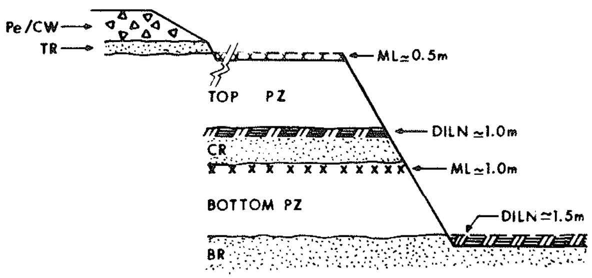
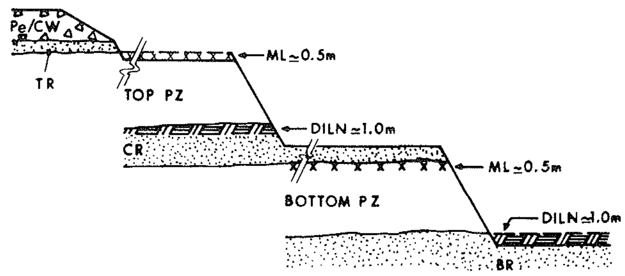
TM/RC ONE-BENCH DRAGLINE SCHEME:



NOTES:

1. Assume that at least one band of centre reject is exposed on the mining face.
2. Every attempt would be made to selectively reject lean oil sand material.
3. For the mining scheme, as illustrated:
   MINING LOSS = 6
   DILUTION = 10
   PLANT FEED = PZ + DILN - ML
   WASTE = Pe CW + TR + (CR - 0.4 DILN) + ML

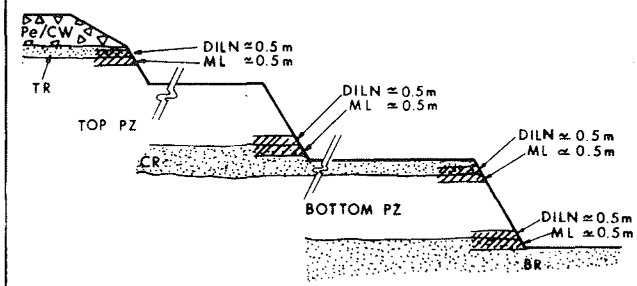
TM/RC TWO-BENCH DRAGLINE SCHEME:



NOTES:

1. For estimation purposes, it is assumed that one band of centre reject appears between two payzones of equal thickness as illustrated. In the event that the CR band appears only in the upper or lower bench, CR separation would be similar to the one-bench method.
2. Assume that the height of each bench would be equivalent to:
   1 2 (Net PZ thickness + CR thickness)
3. For the mining scheme, as illustrated:
   MINING LOSS = 4
   DILUTION = 8
   PLANT FEED = PZ + DILN - ML
   WASTE = Pe CW + TR + (CR - ½ DILN) + ML

TM/RC THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME



NOTES:

1. Assume that at least one band of centre reject is exposed by bucket wheel mining.
2. Assume that the height of each bench would be equivalent to:
   1 3 (Mining depth)
3. Every attempt would be made to selectively reject lean oil sand material.
4. For this bucket wheel mining scheme:
   MINING LOSS = 4
   DILUTION = 4
   PLANT FEED = PZ + DILN - ML
   WASTE = Pe CW + (TR - ¼ DILN) + (CR - ½ DILN) + ML

LEGEND			
Pe CW	Pleistocene Clearwater Formation material	BR	Bottom Reject
TR	Top Reject	ML	Mining Loss
PZ	Pay Zone	DILN	Dilution

NOT TO SCALE

OIL SANDS MINING SCHEMES

FIGURE 5.1.3-1

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CASE III: Techman/RC Three-Bench Bucket Wheel Excavator Scheme

Assumptions:

a. Bucket wheel excavators mine the total depth in three benches of approximately equal height, as shown in Figure 5.1.3-1.

b. Every attempt is made to selectively reject lean oil sands material.

c. It is assumed that as a result of the digging characteristics of the BWE, interfaces may be separated within 0.5 m of the contact because of the proximity of the operator's cab to the working face, provided the contact could be identified by color or other physical characteristics at all. In the case of routine blasting, a certain amount of mixing occurs in the immediate vicinity of the blasthole. Talus may obscure the working faces prior to removal and for short periods of time during operations, as materials slough off the face.

d. Dilution and mining loss would be expected to occur simultaneously at each interface, and are calculated as follows:

$$\text{Mining Loss} = \frac{0.5\ \text{m} + 0.5\ \text{m} + 0.5\ \text{m} + 0.5\ \text{m}}{49\ \text{m}} \times 100 = 4.08\%$$

$$\text{Dilution} = \frac{0.5\ \text{m} + 0.5\ \text{m} + 0.5\ \text{m} + 0.5\ \text{m}}{49\ \text{m}} \times 100 = 4.08\%$$

Effects of nighttime visibility and other negative operating factors are negligible due to adequate lighting and the proximity of the operator to the working face.

Through the action of mining, the original pay zone is diluted with lean oil sands. A portion of the pay zone is also lost as a mining loss. The net effect of this is that the volume of plant feed to be processed is usually substantially higher than the originally-calculated in situ pay zone reserve volume. Simultaneously, the plant feed bitumen grade is decreased from that calculated for the in-pit pay zone. The extent of such trends depends on the grade of the reject material adjaceent to the ore.

With respect to net bitumen yield, the trends favour the single-bench dragline operation. The highest bitumen yield is achieved from a given ore body, but at the cost of increased fines content, lower bitumen grade and higher quantity of total plant feed. The bucket wheel scheme results in the lowest net bitumen yield, as well as the lowest fines content, highest bitumen grade, and smallest plantfeed quantities. The two-bench dragline scheme is between the above two schemes in net bitumen yield and plant feed throughput. The above mentioned fines content variations will have only minor influence on the selection of the major excavating machinery in an oil sands mine. The major factors will be to the overall geology and the capital and operating cost of the equipment.

An ore body with one or at most a few thick intervals of centre reject can likely be mined with equal effectiveness using any one of the three mining cases described. The trends will be as described above. At the other extreme, an ore body with many intervals of centre reject can be mined with or without much attention to selective excavation of materials. When selective excavating procedures are employed, the dragline mine will produce the greater quantity of plantfeed, with the highest total bitumen content and, provided that the fines content is not excessive, also the highest net bitumen yield in the plant. The bucket wheel mine will produce substantially less plant feed, but at a much higher overall grade and much lower fines content. Alternately, the ore body with many reject intervals could be mined without special attention being paid to reject separation. In this case the plant feed from both types of mining methods would be very similar. Ore bodies with geologic characteristics between the two extremes mentioned are more likely to be encountered. In this situation the decision to incorporate selective practices as routine operating procedures will have to be based on local analysis of the face and an assessment of the excavating capabilities of the prime mover in the given situation. Depending on the frequency of centre reject occurrence, thickness of intervals between cen tre reject, thickness of reject bands, and thickness of ore grade oil sand, the optimum solution may be either to operate selectively or non-selectively in order to optimize bitumen recovery. Nonetheless, the higher fines contents and greater plant throughput associated with the ore and two-bench dragline schemes may result in a higher capital and operating cost for the extraction plant and tailings disposal facilities. From an environmental perspective, the decision regarding selectivity must tend to minimize pit size per barrel and reduce the volumes of sludge produced.

The mining loss and dilution analysis presented in this section is only one of many possible approaches. If drilling data is more comprehensive, more detail may be included, and more flexibility as to the manner of treatment prevails. In the case of this study, Techman/RC are of the opinion that the treatment is as reliable as possible considering the level of detail in the data used. The Consultants believe that the conclusions would not differ significantly even if other approaches were taken to analysing the problem of prime mover mining selectivity. The mine simulation programs, developed later in the study, use the selectivity criteria described in this subsection as well as other adjustments reflecting plant recovery as fines and ore grades change.

Plant feed and waste material data are presented for Ore Bodies No. 1, 2 and 4 in the following tabulations:

Table 5.1.3-1, Ore Body No. 1A after Mining Loss and Dilution
Table 5.1.3-2, Ore Body No. 1B after Mining Loss and Dilution
Table 5.1.3-3, Ore Body No. 2 after Mining Loss and Dilution
Table 5.1.3-4, Ore Body No. 4 after Mining Loss and Dilution

The impact of mining method on bitumen yield, fines content, sludge production, and plant throughput can be seen in the following tables, which present plant feed and waste material data for Ore Bodies No. 1, 2 and 4. For the purposes of comparing mining methods only, the entire ore body was assumed to have one large centre reject band.

Table 5.1.3-1

ORE BODY NO. 1A AFTER MINING LOSS AND DILUTION

	Oil Sands Volume (bank m³ x 10⁶)	Oil Sands Weight (tonnes x 10⁶)	Bitumen Weight (tonnes x 10⁶)	Oil Sands Grade %	Fines Content %	Waste Material		Volume of Waste With Reclamation Potential (bank m³ x 10⁶)	Volume of Waste With No Reclamation Potential (bank m³ x 10⁶)
TWO-BENCH DRAGLINE SCHEME:						TWO-BENCH DRAGLINE SCHEME:			
Pay Zone	1,024.73	2,051.82	239.24	11.66	13.36	Pe/CW		211.99	
Less 4% (ML)	-40.99	-82.07	-9.57	11.66	13.36	TR			167.37
Subtotal	983.74	1,969.75	229.67	11.66	13.36	76% of CR =	95.96		
Plus 8% (DLN)	+81.98	+164.15	+5.47	3.33	46.68	Less 1/2 DLN =	-40.99		
Subtotal	1,065.72	2,133.90	235.14	11.02	15.92	CR=	54.97		54.97
Plus 24% of CR	+30.30	+60.67	+2.02	3.33	46.68	ML		______	40.99
Plant Feed	1,096.02	2,194.57	237.16	10.81	16.76	Totals		211.99	263.33
ONE-BENCH DRAGLINE SCHEME:						ONE-BENCH DRAGLINE SCHEME:			
Pay Zone	1,024.73	2,051.82	239.24	11.66	13.36	Pe/CW		211.99	
Less 6% (ML)	-61.48	-123.11	-14.35	11.66	13.36	TR			167.37
Subtotal	963.25	1,928.71	224.89	11.66	13.36	76% of CR =	95.96		
Plus 10% (DLN)	+102.47	+205.18	+6.83	3.33	46.68	Less 40% DLN =	-40.99		
Subtotal	1,065.72	2,133.89	231.72	10.86	16.56	CR =	54.97		54.97
Plus 24% of CR	+30.30	+60.67	+2.02	3.33	46.68	ML		______	61.48
Plant Feed	1,096.02	2,194.56	233.74	10.65	17.39	Totals		211.99	283.82
THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:						THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:			
Pay Zone	1,024.73	2,051.82	239.24	11.66	13.36	Pe/CW		211.99	
Less 4% (ML)	-40.99	-82.07	-9.57	11.66	13.36	TR (less 1/4 DLN)			157.12
Subtotal	983.74	1,969.75	229.67	11.66	13.36	Total CR =	126.26		
Plus 4% (DLN)	+40.99	+82.07	+2.73	3.33	46.68	Less 1/2 DLN =	-20.50		
Plant Feed	1,024.73	2,051.82	232.40	11.33	14.68	CR =	105.76		105.76
						ML		______	40.99
						Totals		211.99	303.87

Table 5.1.3-2

ORE BODY NO. 1B AFTER MINING LOSS AND DILUTION

	Oil Sands Volume (bank m^3 x 10^6)	Oil Sands Weight (tonnes x 10^6)	Bitumen Weight (tonnes x 10^6)	Oil Sands Grade %	Fines Content %	Waste Material		Volume of Waste With Reclamation Potential (bank m^3 x 10^6)	Volume of Waste With No Reclamation Potential (bank m^3 x 10^6)
TWO-BENCH DRAGLINE SCHEME:						TWO-BENCH DRAGLINE SCHEME:			
Pay Zone	926.60	1,855.33	212.99	11.48	14.09	Pe/CW		345.60	
Less 4% (ML)	-37.06	-74.21	-8.52	11.48	14.09	TR			76.83
Subtotal	889.54	1,781.12	204.47	11.48	14.09	76% of CR =	137.69		
Plus 8% (DLN)	+74.13	+148.43	+4.94	3.33	46.68	Less 1/2 DLN =	-37.06		
Subtotal	963.67	1,929.55	209.41	10.85	16.60	CR =	100.63		100.63
Plus 24% of CR	+43.48	+87.06	+2.87	3.33	46.68	ML			37.06
Plant Feed	1,007.15	2,016.61	212.28	10.53	17.89	Totals		345.60	214.52
ONE-BENCH DRAGLINE SCHEME:						ONE-BENCH DRAGLINE SCHEME:			
Pay Zone	926.60	1,855.33	212.99	11.48	14.09	Pe CW		345.60	
Less 6% (ML)	-55.60	-111.33	-12.78	11.48	14.09	TR			76.83
Subtotal	871.00	1,744.00	200.21	11.48	14.09	76% CR =	137.69		
Plus 10% (DLN)	-92.66	+185.53	+6.18	3.33	46.68	Less 40% DLN =	-37.06		
Subtotal	963.66	1,929.53	206.39	10.70	17.20	CR =	100.63		100.63
Plus 24% of CR	+43.48	+87.06	+2.87	3.33	46.68	ML			55.60
Plant Feed	1,007.14	2,016.59	209.26	10.38	18.48	Totals		345.60	233.06
THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:						THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:			
Pay Zone	926.60	1,855.33	212.99	11.48	14.09	Pe/CW		345.60	
Less 4% (ML)	-37.06	-74.21	-8.52	11.48	14.09	TR (less 1/4 DLN)			67.57
Subtotal	889.54	1,781.12	204.47	11.48	14.09	Total CR =	181.17		
Plus 4% (DLN)	+37.06	+74.21	+2.47	3.33	46.68	Less 1/2 DLN =	-18.53		
Plant Feed	926.60	1,855.33	206.94	11.15	15.40	CR =	162.64		162.64
						ML			37.06
						Totals		345.60	267.27

Table 5.1.3-3

ORE BODY NO. 2 AFTER MINING LOSS AND DILUTION

	Oil Sands Volume (bank $m^3 \times 10^6$)	Oil Sands Weight (tonnes $\times 10^6$)	Bitumen Weight (tonnes $\times 10^6$)	Oil Sands Grade %	Fines Content %	Waste Material		Volume of Waste With Reclamation Potential (bank $m^3 \times 10^6$)	Volume of Waste With No Reclamation Potential (bank $m^3 \times 10^6$)
TWO-BENCH DRAGLINE SCHEME:						TWO-BENCH DRAGLINE SCHEME:			
Pay Zone	952.00	1,906.19	221.69	11.63	13.48	Pe/CW		181.14	
Less 4% (ML)	-38.08	-76.25	-8.87	11.63	13.48	TR			59.57
Subtotal	913.92	1,829.94	212.82	11.63	13.48	76% of CR =	150.74		
Plus 8% (DLN)	+76.16	+152.50	+5.08	3.33	46.68	Less 1/2 DLN =	-38.08		
Subtotal	990.08	1,982.44	217.90	10.99	16.04	CR =	112.66		112.66
Plus 24% of CR	+47.60	+95.31	+3.17	3.33	46.68	ML			38.08
Plant Feed	1,037.68	2,077.75	221.07	10.64	17.44	Totals		181.14	210.31
ONE-BENCH DRAGLINE SCHEME:						ONE-BENCH DRAGLINE SCHEME:			
Pay Zone	952.00	1,906.19	221.69	11.63	13.48	Pe/CW		181.14	
Less 6% (ML)	-57.12	-114.37	-13.30	11.63	13.48	TR			59.57
Subtotal	894.88	1,791.82	208.39	11.63	13.48	76% of CR =	150.74		
Plus 10% (DLN)	+95.20	+190.62	+6.35	3.33	46.68	Less 40% DLN =	-38.08		
Subtotal	990.08	1,982.44	214.74	10.83	16.68	CR =	112.66		112.66
Plus 24% CR	+47.60	+95.31	+3.17	3.33	46.68	ML			57.12
Plant Feed	1,037.68	2,077.75	217.91	10.49	18.04	Totals		181.14	229.35
THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:						THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:			
Pay Zone	952.00	1,906.19	221.69	11.63	13.48	Pe/CW		181.14	
Less 4% (ML)	-38.08	-76.25	-8.87	11.63	13.48	TR (less 1/4 DLN)			50.05
Subtotal	913.92	1,829.94	212.82	11.63	13.48	Total CR =	198.34		
Plus 4% (DLN)	+38.08	+76.25	+2.54	3.33	46.68	Less 1/2 DLN =	-19.04		
Plant Feed	952.00	1,906.19	215.36	11.30	14.80	CR =	179.30		179.30
						ML			38.08
						Totals		181.14	267.43

Table 5.1.3-4

ORE BODY NO. 4 AFTER MINING LOSS AND DILUTION

	Oil Sands Volume (bank $m^3 \times 10^6$)	Oil Sands Weight (tonnes $\times 10^6$)	Bitumen Weight (tonnes $\times 10^6$)	Oil Sands Grade %	Fines Content %
TWO-BENCH DRAGLINE SCHEME:					
Pay Zone	461.58	924.22	111.09	12.02	11.91
Less 4% (ML)	-18.46	-36.96	-4.44	12.02	11.91
Subtotal	443.12	887.76	106.65	12.02	11.91
Plus 8% (DLN)	+36.93	+73.94	+2.46	3.33	46.68
Subtotal	480.05	961.20	109.11	11.35	14.60
Plus 47% of CR	+31.00	+62.07	+2.07	3.33	46.68
Plant Feed	511.05	1,023.27	111.18	10.86	16.56
ONE-BENCH DRAGLINE SCHEME:					
Pay Zone	461.58	924.22	111.09	12.02	11.91
Less 6% (ML)	-27.69	-55.44	-6.66	12.02	11.91
Subtotal	433.89	868.78	104.43	12.02	11.91
Plus 10% (DLN)	+46.16	+92.43	+3.08	3.33	46.69
Subtotal	480.05	961.21	107.51	11.18	15.28
Plus 47% of CR	+31.00	+62.07	+2.07	3.33	46.68
Plant Feed	511.05	1,023.28	109.58	10.71	17.16
THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:					
Pay Zone	461.58	924.22	111.09	12.02	11.91
Less 4% (ML)	-18.46	-36.97	-4.44	12.02	11.91
Subtotal	443.12	887.25	106.65	12.02	11.91
Plus 4% (DLN)	+18.46	+36.97	+1.23	3.33	46.68
Plant Feed	461.58	924.22	107.88	11.67	13.32

Waste Material		Volume of Waste With Reclamation Potential (bank $m^3 \times 10^6$)	Volume of Waste With No Reclamation Potential (bank $m^3 \times 10^6$)
TWO-BENCH DRAGLINE SCHEME:			
Pe/CW		136.93	
TR			44.95
53% of CR =	34.96		
Less 1/2 DLN =	-18.46		
CR =	16.50		16.50
ML			18.46
Totals		136.93	79.91
ONE-BENCH DRAGLINE SCHEME:			
Pe/CW		136.93	
TR			44.95
53% of CR =	34.96		
Less 40% DLN =	-18.46		
CR =	16.50		16.50
ML			27.69
Totals		136.93	89.14
THREE-BENCH BUCKET WHEEL EXCAVATOR SCHEME:			
Pe/CW		136.93	
TR (less 1/4 DLN)			40.34
Total CR =	65.97		
Less 1/2 DLN =	-9.23		
CR =	56.74		56.74
ML			18.46
Totals		136.93	115.54

5.1.4 MINING GEOTECHNICS[1,2,3,4,5]

Introduction

The stability of mining slopes in oil sands depends on many parameters, all of which can be grouped into three categories: geological, geotechnical, and design. The geological parameters which are of significance to slope performance include lithology, stratigraphy, and structure of the oil sands deposit. General information on these aspects of the oil sands is available in the current literature. The regional geological parameters are discussed in Sections 3.1, 3.2, 3.3 and 5.1.1. Site-specific information pertinent to Ore Bodies No. 1, 2, and 4 was insufficient to allow for a detailed geological evaluation.

The significant geotechnical parameters include such factors as bulk density, Young's Modulus, Poisson's Ratio, shear strength, compressibility factors, permeability, and porosity. Again, site specific information for these parameters is not available but sufficient experience has been gained by operators in this field to allow reasonable understanding of the geotechnical parameters of typical pit slopes.

The practical application of geological and geotechnical parameters is only realized during the consideration of design parameters such as slope configuration (height, slope angle, bench widths, etc.), rate of excavation, set-back distance of excavating equipment, bench loading by equipment, ground dewatering method, blasting method, and surface water control. These parameters are developed through an evaluation of the geological and geotechnical characteristics of the oil sands, and the performance of slopes in other oil sands mining operations. Only the design parameters can be varied, as both geotechnical and geological properties are factual for any given situation. A clear understanding of the first two types of parameters allows a reasonable assessment of the latter. The following review of geological, geotechnical, and design parameters is intended to assess the influence of these parameters on reclamation.

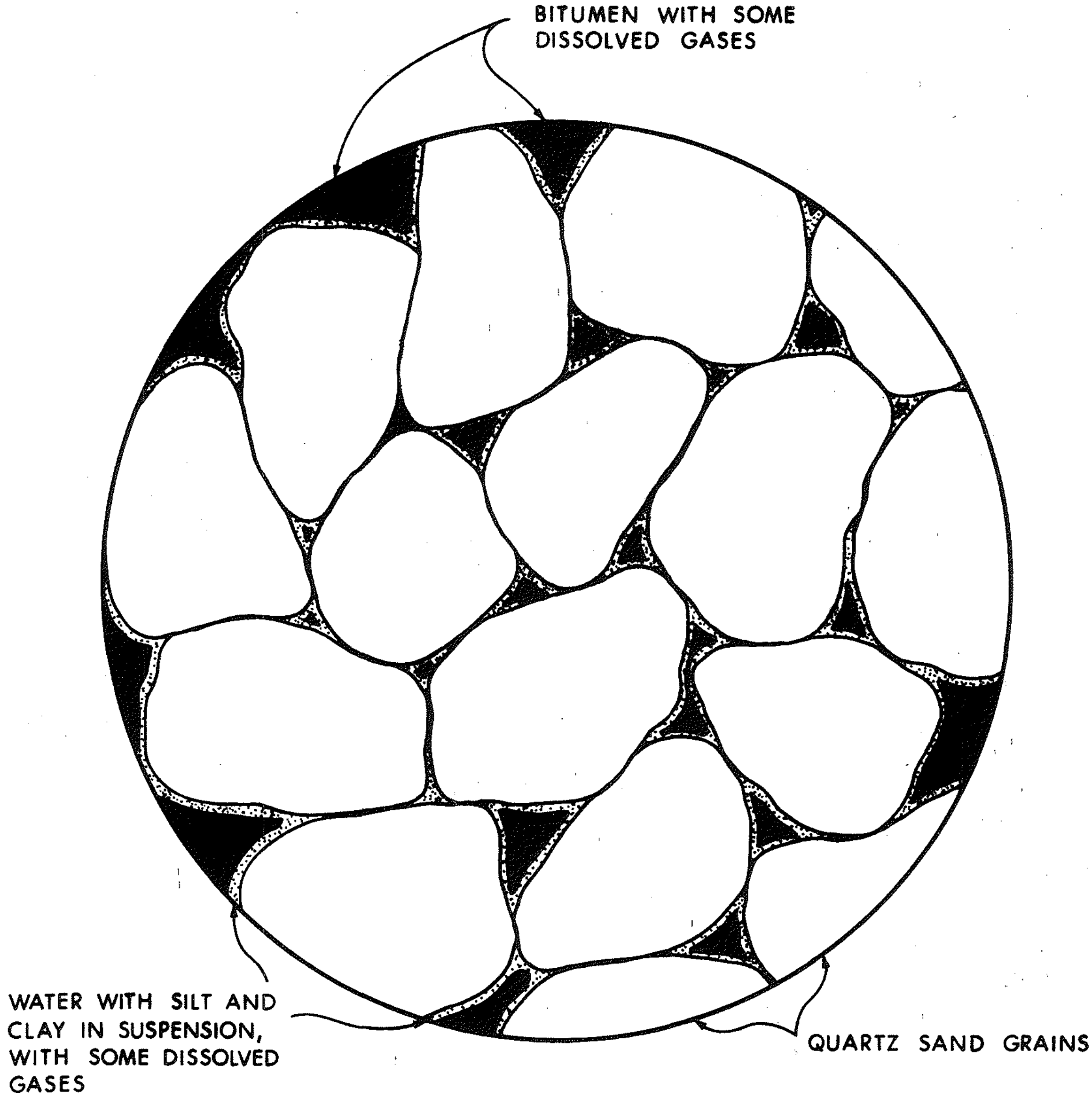
Physical Characteristics of the Oil Sands

Oil sand is a four-phase hydrocarbon solid (Figure 5.1.4-1) consisting of a solid phase (predominantly sand), liquid phase (water), gaseous phase (predominantly carbon dioxide, nitrogen and methane), and viscous hydrocarbon phase (black and dense bitumen, about 8° API). A typical analysis (by weight) of the in situ oil sand is 84% solids, 4% water and 12% bitumen.

The solid phase consists predominantly of fine- to medium-grained angular to sub-angular quartz sand with small quantities (generally less than 20%) of silt and clay-sized material. The clay minerals are usually kaolinite and illite, although some smectites are present. The quartz grains are at least 99% water-wet. The other minerals are hydrophilic and are concentrated in the water layer. Bitumen and gases occupy the remaining pore spaces in the oil sands material. In addition, appreciable volumes of gases are dissolved in the bitumen and water phases (at depth, liquid saturation is probably 100%). The mean bitumen content varies in the ore bodies, with the average grade varying from 11 to 14% in the mineable zones (+5% bitumen cutoff). The overall bitumen content varies from 0 to 28% through the Athabasca oil sands deposit. Typically, the bitumen content varies inversely with the clay content. The sum of the bitumen and water contents is roughly constant at 16 - 18 percent by weight.

Geotechnical Properties of the Oil Sands

Oil sands in situ are relatively dense compared to other granular materials, having a bulk density of approximately 2.20 to 2.25. The structure of the oil sands is inter-penetrative or "locked", with the grain-to-grain contacts mainly of the long and concavo-convex type. This locked structure has resulted from the process of dissolution and redeposition of quartz at the grain boundaries. This process has been active in the oil sands under applied loads over geological time. The locked structure is responsible for the dense nature of the oil sands.



HIGH MAGNIFICATION

OIL SANDS STRUCTURE AS A FOUR-PHASE HYDROCARBON SOLID

FIGURE 5.1.4.-1

25 - 8 - 78

Undisturbed oil sands have an apparent high strength, as evidenced by the steep natural slopes exposed along major drainages within the Athabasca region. This high strength has resulted from the highly interpenetrative structure of the oil sands. The oil sands exhibit a Mohr-Coulomb failure envelope (which can be approximated by a power law). At low effective stresses, the material is cohesionless and has high angles of internal friction (56° - 61°). At higher stresses, the angle of internal friction is reduced to approximately 30°. The process of shear invariably shows high dilatancy (increase in volume) at low confining pressures, but dilatancy decreases as the confining stress is increased. At low normal stresses, the high dilatancy is a result of grains riding up over each other, requiring high levels of energy, and resulting in high angles of internal friction. At high normal stress, dilatancy is restricted and grain shearing occurs, resulting in a lower angle of internal friction and the development of an apparent cohesion value.

Pit Slope Behavior in Oil Sands

High in situ horizontal stresses are known to exist within the Athabasca oil sands region. Excavation within the oil sands results in a rotation of the principal stress field such that the maximum principal stress (horizontal in situ stress) tends to parallel the slope face. The result is a high concentration of stresses near the toe of the slope, and the creation of tensile minimum principal stresses at the slope crest and at the top of the slope face. This modification in the in situ stress pattern can cause the development of tension cracks back from the crest of the slope. Progressive failure of the pit slope can begin from the toe of the slope. When shear stresses exceed shear strength, continued redistribution of stresses results as fracturing propagates. Finite element analysis carried out by Dusseault has shown that stress ratios on circular failure arcs within an excavated oil sands slope are greatest close to the slope face, and are generally less stable, therefore reducing the possibility of a deepseated rotational failure. Block slide failures evaluated using plane shear total stress analysis indicate that the factor of safety of oil sands is generally high. Low factors of safety occur when deep, water-filled cracks occur close to the crest. Dusseault's analysis did not take into account progressive failure caused by unequal stress distribution on structural disturbance at or near the face (including pit bottom heave from excess water pressure below the floor); however, the failure criterion used in that analysis is considered conservative because the in situ strengths can be much higher than those indicated in the laboratory test results.

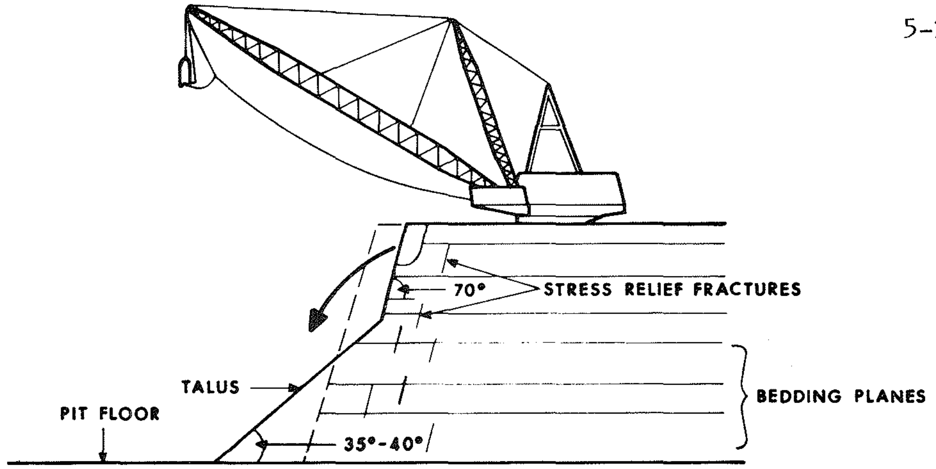
The three major types of failure modes (Figure 5.1.4-2) which can be expected to occur within excavated slopes are:

a. Slabbing and slope ravelling as a result of exfoliation jointing caused by stress relief during excavation.
b. Block slide failure along dipping cross-stratification and bedding planes, often combined with tension cracks opening up at the surface, or other structural defects such as faults resulting from differential compaction or solution collapse.
c. Shallow rotational slides caused when the modified shear stresses in the outer part of the slope face exceed the shear strength of the oil sands material.

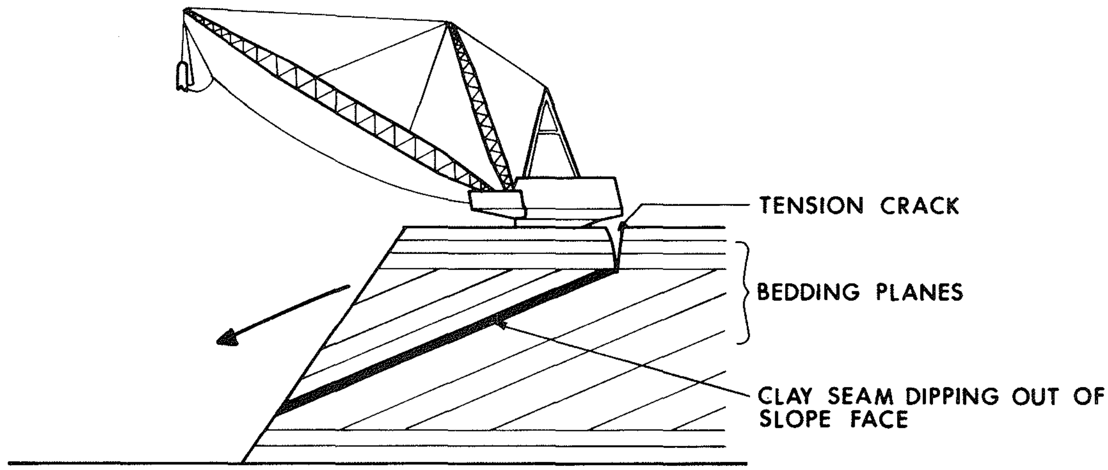
The most prevalent mode of failure may be the slabbing phenomenon which results in development of a compound slope (lower talus at 34° - 40° slope angle, with the upper half of the slope steepened to 70°). This slabbing phenonenon is sensitive to mining rate, depth of cut, and ambient temperature. The stability of a potential failure wedge within the slope can be reduced by water infilling the tension cracks and exerting a further driving force in the form of a hydrostatic head. Sliding can occur on weak clay seams dipping at relatively shallow angles (4° - 6°) into the pit.

Pit Slope Behavior in Overburden

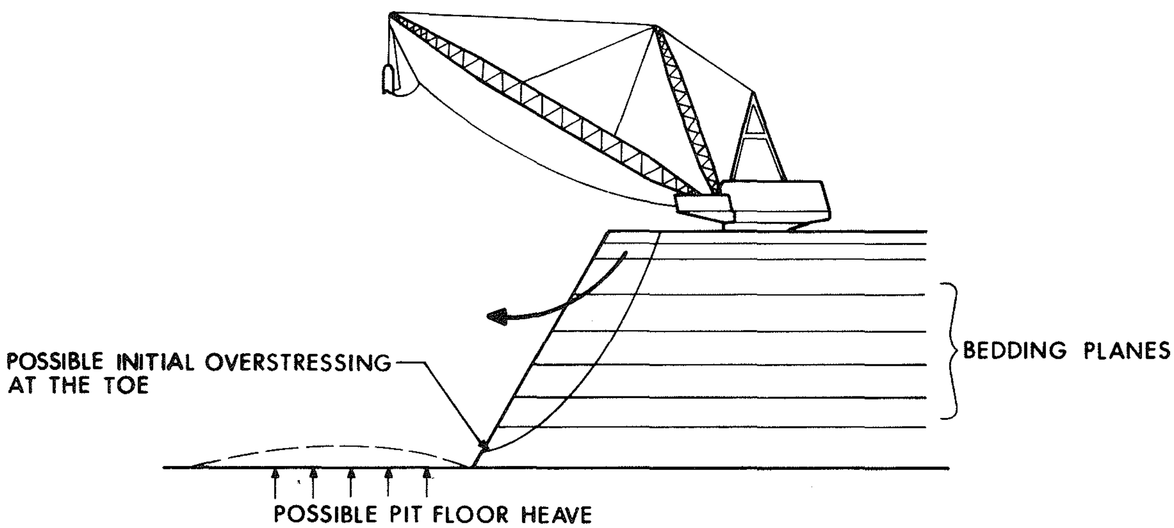
A physical description of the overburden encountered at all three mine sites is provided in Section 3.1, Geomorphology; and Section 3.2, Regional Geology. The geotechnical characteristics of the clays, silts, sands, and gravels are similar to those described in the literature and are not discussed further.



a) SLABBING AND SLOPE RAVELLING RESULTING IN DEVELOPMENT OF A COMPOUND SLOPE



b) BLOCK SLIDE FAILURE



c) SHALLOW ROTATIONAL SLIDES

NOTE: DRAGLINE SHOWN FOR ORIENTATION PURPOSES ONLY. SIMILAR MODES OF FAILURE CAN BE EXPECTED FOR BUCKET WHEEL EXCAVATED SLOPES. POSSIBILITY OF FAILURE WILL DECREASE WITH DECREASING BENCH HEIGHT, SLOPE ANGLE, AND MINING RATE.

MAJOR TYPES OF FAILURE MODES POSSIBLE IN OIL SAND SLOPES

FIGURE 5.1.4.- 2

7-6-78R

Materials having high clay and silt contents are of concern to mine operations. Construction of tailings dykes in areas containing clays and silts may require that consideration be given to the rate of dyke construction and lower embankment slope angles. The occurrence of troublesome horizons may require detailed work, but some indications can already be gained from recent work by the Research Council of Alberta, which provides extensive surficial geology mapping. As a result of this information, discussions with informed geotechnical consultants, and in-house experience, it was decided to utilize dyke slopes of 4h:1v in all cases for out-of-pit dykes.

Pit Floor Behavior

Mining removes the overburden and oil sands, and creates a pit floor at the bottom of the oil sands. This floor is lean oil sands or unconsolidated sands and clays. These formations are no longer under stress from the overlying material, and may heave from the unrestrained horizontal stresses and/or residual water pore pressure. If it heaves it may produce a surface with poor trafficability and low compressive strength.

When in-pit dykes are constructed, a compression load may develop under the dykes. Too rapid a build-up of this stress may lead to shear failure or pore pressure build-up, and finally dyke failure. Constructing the lower portion of the dyke as soon as possible after removal of the lowest oil sand bench is advantageous, since this limits the possibility of floor heave in the dyke area.

No practical experience exists as a basis for predicting the behavior of the pit floor during the construction of in-pit tailings dykes. Consequently, a conservative dyke slope angle of 6h:1v was adopted except in cases where a considerable depth of lean sands remained below the pit floor. In the latter situation, a 5h:1v dyke slope was adopted. Construction of the in-pit dykes should occur as soon as possible after removal of the lowest oil sands bench. Dewatering of the groundwater aquifers must be continued during and after the construction of the dyke, i.e., until the next adjacent pond is partially filled or the adjacent pit is backfilled with overburden against the dyke.

A number of years in advance of mining, groundwater aquifer depressurization must be started. Dewatering wheels are located to draw the water table down to below the pit bottom. The water table is allowed to recharge the area gradually, after backfilling is completed and the in-pit tailings dykes established.

The scale of the depressurization procedures depends more on the shape of the mine, overall mine layout, and the schedule of pit backfilling than any other factors. Whether draglines or bucket wheels are employed is insignificant. However, the manner in which these excavators are employed influences the location and density of dewatering wells required. A single bench dragline operation without a separate overburden bench allows the most distant well to be located closer to the toe of the oil-sands mining bench that is possible with a multiple bench dragline or bucket wheel system. The mean distance between the crest of the overburden bench and the toe of the lowest oil sands bench is greater than the mean crest to toe distance in a single bench operation. The array of dewatering wells must be designed to match the progress of backfilling operations and in-pit dyke construction sequence at the Minium and Improved Levels, and the general backfilling of overburden and dry tailings sand at the Enhanced Level. The greater the interval of time between the in-pit dyke construction, the longer the depressurization wells must be operated. The shape of the ore body has a direct influence on the design and arrangement of depressurizatin wells to be installed and required to be operational at any given time. The influx of groundwater is greater as the pit perimeter increases for a given size of mine, since both exposed pit wall and pit floor areas are increased.

Local dewatering of perched water tables is also required. Sumps are required to collect local run-off water.

Insufficient information exists to adequately estimate the capital and operating costs of the dewatering systems. Initial design and cost estimates indicate that the dewatering costs per barrel of bitumen are essentially the same in each of the twelve mine plans developed.

Geotechnical Design Criteria

On the basis of field observations at the G.C.O.S. Ltd. and Syncrude Canada Ltd. operations, the Shell test pit (Bituminous Sands Lease 13), numerous outcrop locations along rivers in the Fort McMurray area, and discussion with knowledgeable professionals, the following geotechnical design criteria have been developed for use in this study.

Slope Angles of Working Faces:

- Glaciofluvial outwash material such as sands and gravels have been observed to stand at a 2h:1v slope (27°) over a long term. For short term stability of pit slopes, the slope angle could be as much as 35°, provided that (perched) groundwater is drained well in advance of excavation.

- The Clearwater Formation is exposed in a number of locations along rivers in the Athabasca oil sands area. An analysis of these outcrops has shown that for clays and silts, such as those found in the Clearwater Formation, the following would apply:

Bank Height (metres)	Angle of Stable Slope
5	62°
10	45°
15	38°
20 or more	35°

- In numerous river exposures, compound slopes of glaciofluvial sands and gravels, glacial till, Clearwater Formation, Wabiskaw Member, and lean oil sands of the McMurray Formation have been observed to stand at angles as steep as 45°.

- Field observations at G.C.O.S. operations and the Shell test pit indicate that excavations in rich oil sands will stand at slope angles ranging from 45 to 70°, depending on the bitumen saturation and distribution of centre reject.

- Digging angles based on the foregoing and a reasonable factor of safety have been employed in mine design criteria for this study. Safe digging angles for overburden and plant feed grade oil sands are 35º and 45º, respectively.

Bearing Pressures:

- range from 0 to 1.5 kg/cm^2 (20 psi) for overburden silts and clays (i.e., Clearwater Formation) depending on the moisture content; the lowest being for water-saturated silts and clays, and the highest for dry material.

- have been observed to be less than about 3.5 kg/cm^2 (50 psi) for undisturbed oil sands of the McMurray Formation.

Trafficability:

In general, freshly-exposed oil sands and Clearwater materials provide a reasonable working surface for rubber-tired and crawler-mounted mobile equipment. However, if exposed to rainfall or runoff, the clays become slippery, and travel is difficult. Glacial tills rich in clay pose similar problems. In the case of wet oil sands, repeated passage of rubber-tired equipment causes the surface layers to become churned up and impassable. Traction under this loosened oil sand is not adequate and it may be necessary to spread a pad of gravel to improve trafficability.

Crawler equipment is expected to have minimal difficulties except possibly in areas with highly bitumen-saturated oil sands. Roadways intended to be used for a period of months by rubber-tired equipment require additional surfacing.

Tailings Dyke Slopes:

- out-of-pit dyke slopes of 4h:1v are conservative
- in-pit dyke slopes of 6h:1v or at maximum 5h:1v may be considered to be necessary until further field experience is gained by the industry

- rate of dyke construction has been assumed for the purposes of this study to be adequately compensated for by the selection of reasonable slope angles.

Impact of Geotechnical Constraints on Reclamation Potential

The effect of slope design criteria is noticed as a slight change to the average stripping ratio of the mine, and in dramatic changes to the incremental stripping ratio along the pit wall. In terms of impact on reclamation potential and costs, the effect of reducing pit slope angles is noticed in the following areas:

- increased volumes of overburden are removed as slope angles are decreased.
- significantly increased areal extent of surface disturbance as the slope angles are decreased.
- increase in tailings sands and sludge quantities produced at a given plant size since proportionally larger quantities of poor-grade material must be mined to achieve a given tonnage (assumes that the higher-grade and lower-fines-content oil sands are found in the lower half of the mined oil sands zone).

Economics dictate that the maximum allowable slope angle and, therefore, the most environmentally desirable slope angle be chosen. On the other hand, erosion control aspects necessitate a reduction in the individual bench slope angles thus minimizing in-pit erosion.

The final pit remaining after mining has ceased is filled with water, possibly pumped from the Athabasca River. The effect of the rising water level on the pit wall stability of the rising water level has not been determined. If a permanent lake is to be created it is necessary to mine to an overall shallower pit slope angle, which would remain stable during and after the filling of the pit with water. This leads to a greater areal disturbance compared to a pit which would be filled with solid wastes such as tailings or overburden.

Because the various working benches in an oil sands mining operation are staggered by a few hundred metres in the direction of mining, slope angles of working faces do not necessarily affect the reclamation potential directly. However, the number of benches and the distance allowed for each bench may, at times, be the limiting factor in permitting a rapid change from out-of-pit tailings disposal to in-pit disposal. This effect may continue by limiting the rate at which additional in-pit dykes can be established, leaving the mine with a continual shortage of in-pit storage capacity. As well, the volume of waste that must be stored in an out-of-pit waste dump increases with increased bench widths. In this study, the bench widths are chosen to permit efficient use of prime excavators, and to maximize flexibility in the location and method of tailings disposal.

It should be noted that the construction of tailings impoundment structures near the pit perimeter is assumed to be possible. However, the strength of underlying geological formations such as the Clearwater Formation must be determined. A decrease in the overall pit-wall slope angle may be necessary when the McMurray Formation contains weak zones. Total area disturbance increases with shallower pit slope angles. Site-specific conditions dictate the location of tailings ponds.

Ground bearing pressure affects the type and size of excavator employed. This results in a plant feed that reflects the mining ability of the prime excavator (discussed earlier in this Chapter). In this study, it is assumed that the ground bearing pressure is adequate for the operation of draglines and bucket wheel excavators. No direct link exists between the prime excavator selected and the potential to reclaim. This is further explained in Subsection 5.4.

Trafficability within the pit is more of a local and seasonal problem. As with ground pressure, the parameter of trafficability does not require specific consideration. Oil sands operators have demonstrated that machinery selected for the mine plans developed in this study can, in fact, be operated successfully.

In contrast, trafficability and ground bearing pressures in muskeg areas have an impact on reclamation potential and cost. Trafficability is the limiting factor in the process of acquiring muskeg for prepared soil, and consequently has a profound influence on the methods of prepared soil manufacture available to the operator.

Trafficability in areas to be reclaimed is also significant, being the most probbmatic during the spring and summer. Much of the hauling of prepared soil may have to be done in the fall and winter when the surface to be reclaimed is either drier of frozen.

5.2 PLANT PROCESS CONSIDERATIONS

This section of the report describes briefly the production and characteristics of wet oil sands tailings. While only the 120,000 BPCD operation is discussed in detail in this section, comments made apply equally to the other two plant sizes discussed in this study; however, the yield of tailings products varies with changes in the average characteristics of plant feed available from the three distinct ore bodies. Bitumen recovery from the tailings sludge is a possibility, and a plant constructed to recover bitumen could serve a dual purpose, making the thickening of tailings slimes more practical.

5.2.1 HOT WATER PROCESSES

A. General

Currently, oil sands operators use the Clark Hot Water process for extraction of bitumen. In this process, mined oil sands are processed continuously in the primary extraction operation as follows:

(i) Conditioning - hot water (82°C), caustic soda (sodium hydroxide) and steam are mixed with oil sands to liberate bitumen.

(ii) Settling - after screening, the hot pulp is allowed to settle. Aerated bitumen is skimmed from the surface as a froth.

(iii) Froth flotation - bitumen in middlings from the settling operation is recovered as a froth by froth flotation, and combined with the froth from step (ii).

Primary extraction tailings are the residue from settling and froth flotation steps. Oversize reject is produced from the screening of conditioned pulp. Bitumen froth from the primary extraction operation is diluted with naphtha and centrifuged in the froth treatment plant, producing tailings containing most of the remaining water and mineral in the froth, as well as some naphtha. A generalized process flow schematic and flow rates for a typical* 120,000 BPCD operation are given in Figure 5.2.1-1.

B. Tailings Description

Tailings typically produced from these operations have the following characteristics:

(i) Primary Extraction Tailings

	Feed Quality	
	Normal	Worst
kg/kg oil sands	1.55	1.72
kg/dm^3	1.45	1.39
% Water	49.5	54.5
% Bitumen	0.5	0.5
% Mineral	50.0	45.0
Temperature	±65°C	±60°C
pH	8.0-9.0	8.0-9.0

(ii) Froth Treatment Tailings

	Normal	Worst
kg/kg oil sands	0.069	0.125
kg/dm^3	1.13	1.07
% Water	77.0	87.3
% Bitumen	3.0	1.6
% Mineral	18.0	10.0
% Naphtha	2.0	1.1
Temperature	±75°C	±70°C
pH	±8.0	±8.0

* The balances shown will vary slightly with plant feed variations caused by changing geological conditions and mining methods.

(iii) Oversize Reject

	Feed Quality	
	Normal	Worst
kg/kg oil sands	0.069	0.10
kg/dm^3	2.21	2.21
% Water	10.0	10.0
% Bitumen	2.0	2.0
% Mineral	88.0	88.0
Temperature	±82°C	±82°C
pH	±8.5	±8.5

C. Tailings Deposition Method

Oversize reject is normally trucked to land fill disposal sites for dumping and compacting. Because of the temperature and clay content of this stream, bin storage or transfer via open conveyor is not advisable, particularly in winter. It may be crushed and diluted for dumping, but this has the undesirable effect of adding more clay to the tailings pond.

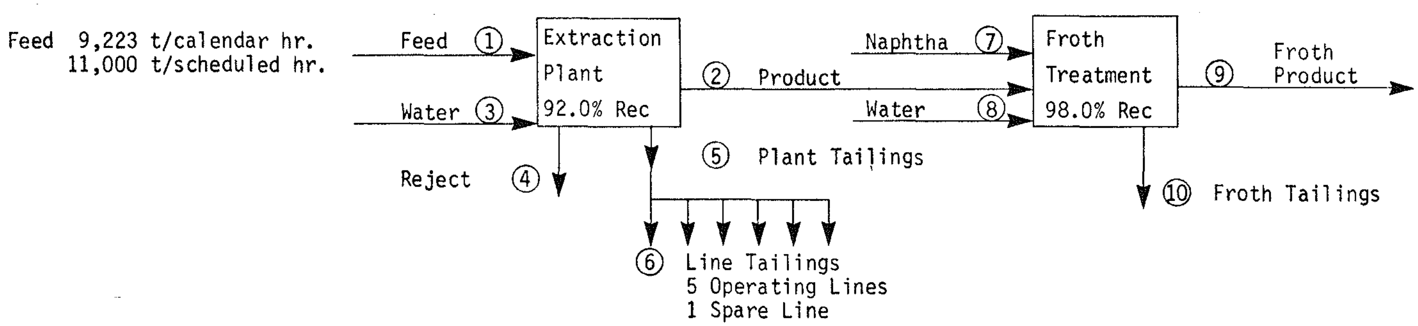
Primary extraction tailings are transferred to the tailings pond by multistage centrifugal pump trains. This operation is a subsystem of the main plant, incorporating distributor, sumps, pumps, pumphouse, and pipelines. Typically, four to six pump trains are employed, each with 3 or more pumps in series. The selection of pump and pipe size, the number of trains, and the number of pumps per train depends upon such factors as availability requirements, slurry critical velocity, and pressure rating of pump castings. The latter limitation can be alleviated by locating booster pump stations along the pipeline, but this should be avoided if possible to minimize cost.

Froth treatment tailings are pumped into the tailings pond in the same manner as primary extraction tailings, although the required capacity is much smaller. These tailings are pumped separately from primary tailings because of the naphtha content, which necessitates explosion-proof equipment.

Figure 5.2.1-1

EXTRACTION PROCESS MATERIAL BALANCE

ORE BODY NO. 2 - 120,000 BPCD



	1. FEED	2. PRODUCT	3. WATER	4. REJECT	5. PLANT TAILINGS	6. LINE TAILINGS	7. NAPHTHA	8. WATER	9. FROTH PRODUCT	10. FROTH TAILINGS
Total Flow kg/s	3,055.5	503.2	2,367.5	175.0	4,744.8	948.9	187.7	35.0	514.3	211.6
Density kg/dm^3	1.6	1.03	1.0	--	1.42	1.42	--	--	--	1.13
Flow L/s	--	489.6	2,367.5	--	3,341.4	668.2	--	--	--	--
Bit. kg/s	339.1	312.0	--	3.5	23.6	4.7	--	--	305.8	6.2
Water kg/s	149.7	151.0	2,367.5	17.5	2,348.7	469.7	--	35.0	22.9	163.0
Solids kg/s	2,566.6	40.2	--	154.0	2,372.4	474.4	--	--	2.1	38.1
Naphtha kg/s	--	--	--	--	--	--	187.7	--	183.4	4.2
% Bit.	11.1	62.0	--	2.0	.5	.5	--	--	59.4	2.9
% Water	4.9	30.0	100.0	10.0	49.5	49.5	--	100	4.4	77.0
% Solids	84.0	8.0	--	88.0	50.0	50.0	--	--	.4	18.0
% Naphtha	--	--	--	--	--	--	100	--	35.6	2.0

D. Tailings Pond Operation

Process Considerations

Pond dykes are constructed mainly from primary extraction tailings sand. The residual water, bitumen, and mineral enter the pond, and settling commences immediately. A significant amount of low-solids water separates from the tailings, and is recycled to the plant as process water.

A "stabilized" pond consists of the following:

(i) An upper layer of clarified water (2.5% clay), maintained at a depth of approximately 3 m.

(ii) A bottom layer containing 5 to 10% clay, and 5 to 40% silt and fine sand, depending on pond age and depth.

The bottom layer, or sludge, is continually growing as tailings enter the system. As well as containing most of the clay and silt, this pool contains the majority of the bitumen and naphtha lost to the plant tailings.

The rate of sludge accumulation depends upon the clay content of the oil sands feed to the extraction operation. A generalized model of tailings pond accumulation rates is outlined in Table 5.2.1-1.

E. Sludge Dewatering

The major technical difficulty in sludge dewatering relates to clays, which exhibit extensive swelling upon dispersion in water (eg. montmorillonite). The extraction process functions under conditions which promote clay dispersion. Use of pond recycle water in the process prevents the use of flocculants in the pond without consideration of the extraction operation. Reclamation of the tailings pond is further complicated by the presence of bitumen and naphtha in the sludge.

Improved reclamation necessitates some level of sludge treatment, with the following general objectives:

(i) to maximize bitumen recovery;
(ii) to maximize water recycle;
(iii) to enhance the final condition of the main tailings pond;
(iv) to satisfactorily impound any sludge treatment residue.

A variety of physical and chemical techniques have been proposed for sludge treatment. Most have been tested to a degree sufficient to show some technical promise, but at high cost.

Some candidates for consideration are filtration, flocculation, freeze-thaw, electrophoresis, and combinations of these and other techniques. Problems exist for all techniques, and it is not the purpose of this study to either rank them or dismiss them from consideration. Test work is continuing in this field, and it is expected that an acceptable process will eventually be developed, incorporating both physico-chemical and mechanical techniques. As well, it is probable that a practical process will be a compromise between cost and "reasonable" reclamation capability.

The sludge treatment facility utilized for this study is not proposed as the "solution". Some potential has, however, been demonstrated for the techniques suggested. The use of the suggested concept permits development of a model for capital and operating costs, but practical application must be confirmed in pilot-scale tests. The conceptual design material balance is as outlined in Figure 5.2.1-2, and involves the following operations:

(i) a floating barge is stationed on the pond surface, from which submersible pumps are suspended in the pond at a depth suitable to transfer somewhat compacted sludge to the surface. Pumps within the barge transfer the sludge via a floating pipeline to and over the dyke to the sludge treatment plant.

Table 5.2.1-1

PREDICTED TAILINGS ACCUMULATION*

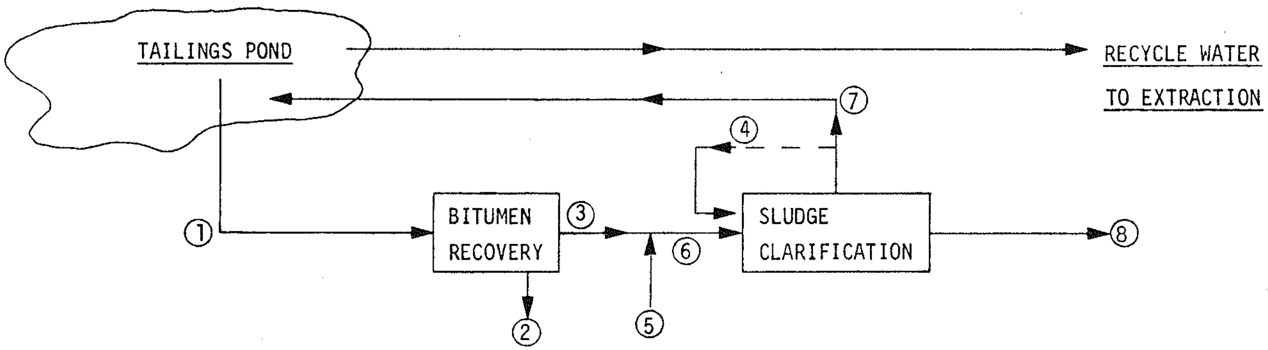
NAME	OIL SANDS COMPOSITION			MINERAL PARTICLE SIZE DISTRIBUTION			TOTAL	CLAY/ WATER RATIO	VOLUME	REMARKS
	TONNE/TONNE OF OIL SAND								m^3/t OIL SANDS	
	BIT.	WATER	MIN.	SAND	SILT	CLAY				
Oil Sands	.111	.049	.840	.709	.098	.033	1.000		.4994	Fines 15.6%
Oversize	.001	.006	.050	.022	.021	.007	.057		.0259	Clay/Fines .25
Subtotal	.110	.043	.790	.687	.077	.026	.943			Oversize 20% of Clay
Process Water	--	.637	--							
Separation Process Feed	.110	.680	.790	.687	.077	.026	1.580	.036		@ 50% Solids
Extraction Product	.101	.049	.013	--	.006	.007	.163			
Extraction Tailings	.009	.631	.777	.687	.071	.019	1.417			
Dilution Water		.137								
Pumped Extr'n Tailings	.009	.768	.777	.687	.071	.019	1.554	.025		@ 50% Solids
Froth Product	.100	.007	.001	--	--	.108				
Froth Tailings	.001	.042	.012	--	.006	.006	.055			
Dilution Water		.012					.012			
Pumped Froth Tailings	.001	.054	.012	--	.006	.006	.067			
Sand to Dyke			.687	.687	--	--	.687			
Contents of Void Volume		.238	.028		.022	.006	.266			
Dyke Material	.0	.238	.715	.687	.022	.006	.953	.025	.5082	
Runoff to Pool	.010	.584	.074	.0	.055	.019	.668			
Sludge	.010	.158	.074	.0	.055	.019	.242	.12	.1967	
Clear Water		.426					.426		.4260	

* @ 11.1% bitumen and 15.6% fines.

Figure 5.2.1-2

SLUDGE TREATMENT MATERIAL BALANCE

ORE BODY NO. 2 - 120,000 BPCD



Name	1. Feed to Bitumen Recovery	2. Bitumen Product (60% Rec)	3. Bitumen Plant Tailings	4. Recycle	5. Flocculant	6. Clarification Feed	7. Clarified Effluent	8. Refuse
Total Flow kg/s	701.97	38.42	663.55		165.44	828.98	498.10	330.88
Density kg/dm^3	1.20	1.10	1.21		1.00	1.16	1.02	1.45
Flow L/s	585.0	34.9	548.4		165.44	714.6	488.3	228.2
Bit. kg/s	25.61	15.37	10.24		--	10.24	.82	9.42
Water kg/s	486.77	17.29	469.48		165.44	634.92	478.90	156.02
Solids kg/s	189.58	5.76	183.82		--	183.82	18.38	165.44
% Bitumen	3.6	40.0	1.5		--	1.2	0.2	2.8
% Water	69.4	45.0	70.8		100.0	76.6	96.1	47.2
% Solids	27.0	15.0	27.7		--	22.2	3.7	50.0

(ii) the sludge treatment plant aerates and floats as much bitumen as possible prior to dewatering. The bitumen is de-aerated and heated with live steam, then pumped to the extraction plant.

(iii) the sludge is conditioned to adjust pH and/or contact flocculant as required. It is then centrifuged in deep-pool scroll centrifuges, with further flocculant addition. The refuse is a sloppy material of 50-60% solids, which is pumped or otherwise transferred to an impoundment area.

The centrate (water effluent) from this operation has a clay content more or less equivalent to pond recycle water. Total removal of this clay is extremely expensive and probably not necessary. Although the centrate may be unacceptable for direct re-use in extraction (due to pH and trace polyelectrolyte content), it can probably be reintroduced to the tailings without harm. Dilution with surface pondwater would probably buffer any possible negative process effects. Care should be taken in the design of the re-injection facilities to ensure that physical re-entrainment of sludge will not occur.

The water content of centrifuged sludge is difficult to predict. It may be possible to dewater to the point where the sludge can be handled as a solid, but the costs (as in completely demineralizing the effluent) are probably prohibitive. A basic problem in the physical chemistry of the clay must be overcome in producing a high solids refuse, i.e., the clay-to-water (c/w) ratio must be forced higher than experience predicts is feasible. Figure 5.2.1-3 indicates the theoretical degree of increase of c/w ratio required for refuse of various % solids. Years of settling in the pond have been shown to produce c/w ratios no higher than 0.15. Increasing solids content from 50% to 65% requires increasing the c/w ratio from about 0.3 to 0.6; in practical terms this may or may not be possible, but the cost is certain to be very high.

For this study, a refuse solids content of 50% is assumed. This material still requires impoundment, but is considerably more amenable to reclamation than the original pond sludge. Addition of a stabilizing agent might enhance the reclamation potential. No cost was included for this approach as considerable developmental work is required to confirm this approach as viable.

F. Tailings System Design - Ore Body No. 2 (120,000 BPCD)

Four major factors affect the design and subsequent operating costs of a tailings system:

(i) Throughput required;
(ii) Physical characteristics of the tailings slurry;
(iii) Line length and associated line heads;
(iv) Pump operating limits.

The design capacity of the tailings pumphouse for the 120,000 BPCD plant is estimated to be 3,340 L/s (litres per second) of 50% solids-by-weight slurry. The maximum estimated flow is 3,800 L/s at 45% solids. This quantity is handled by five 500 mm tailings lines in order to meet the operating stream velocity criteria. One spare line is built into the pumphouse in case of failure or scheduled repair of an operating line.

The characteristic that influences the overall design is the particle size distribution of the solids. If particle sizes are plotted, and a 50% passing size of 160 microns is indicated, the top size of the solid particles should be about 6.5 mm. In order to prevent settling- out of particles during transport along the pipeline, it is important that the proper line velocity be maintained for the material being transported. The design selected uses a 3.72 m/s transport velocity in a 500 mm line. This appears to be about 35% higher than the critical velocity as estimated from existing designs.

The location of the tailings disposal sites varies throughout the life of the mine. This results in varying line lengths and substantial variations in pipe replacement costs. Furthermore, since the head is a maximum only when pumping to a distant out-of-pit tailings pond,

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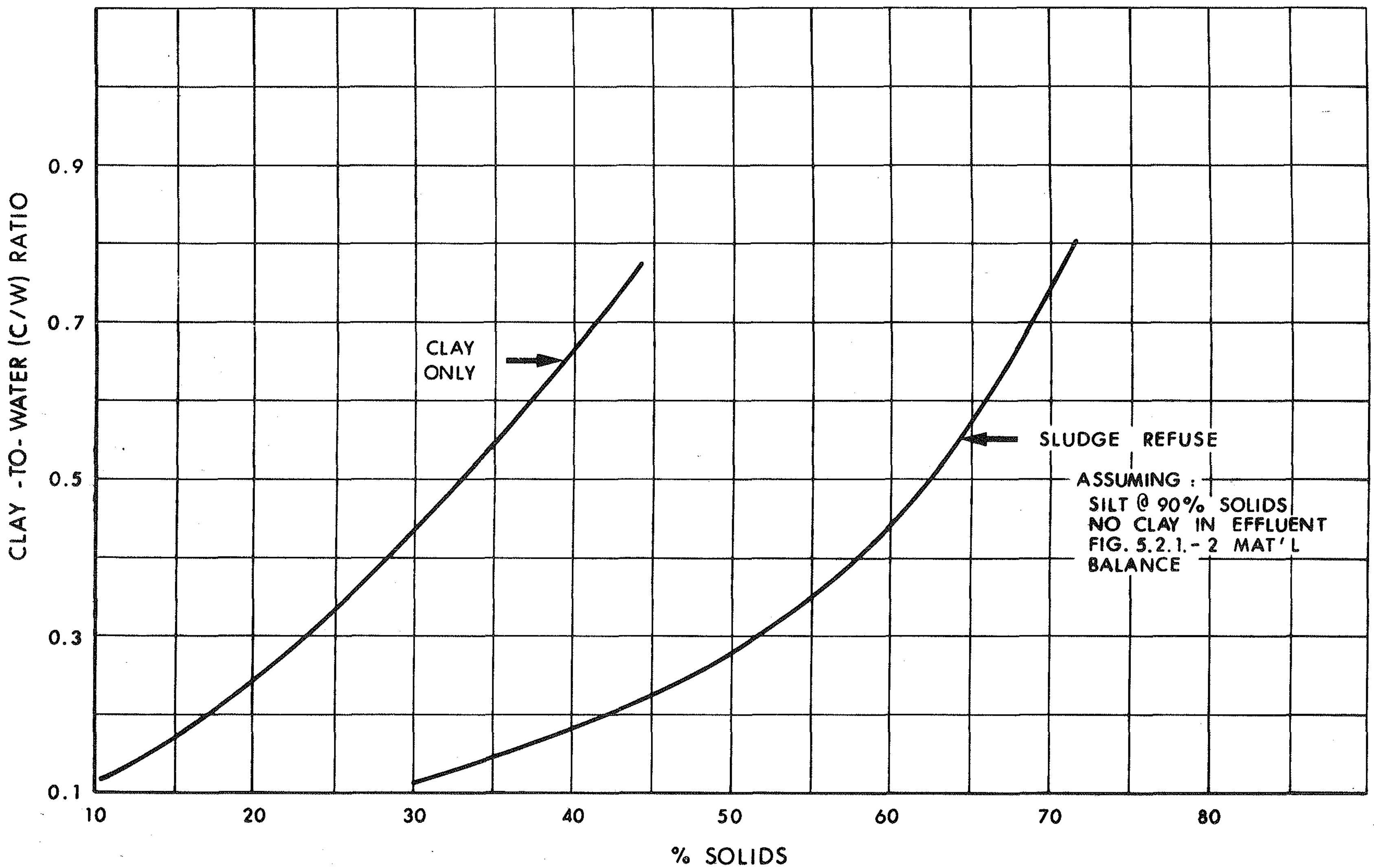


CLAY - TO - WATER RATIO VS. % SOLIDS

FIGURE 5.2.1.- 3

pumping horsepower and, consequently, replacement capital and operating costs vary over the life of the mine. Considering the extremely large volumes transported, even small decreases in unit costs (as the mine progressively advances from out-of-pit tailings storage through to various in-pit storage sites made available as the oil sands are mined) result in many millions of dollars reduction in capital and operating costs annually. The detailed concepts presented in various later chapters describe such sequences.

The static head of the system is fixed by the dam height. A maximum height of 60 m has been used for tailings pond dykes in this study. Additional head is taken up by line loss due to pipe wall friction (dynamic head). A friction factor of C = 120, plus 10 percent for slurry correction, yields an estimated dynamic loss of 2.66 m per 100 m for the system.

Slurry pumps of conventional design can withstand pressures up to 2758 kPa (400 psi). Due to this pressure limitation and other operating difficulties, it was decided that no more than five pumps are to be operated in series. This provides a pumping capacity of 760 L/s per line at about 183 m of head.

G. Tailings Systems Operating Considerations - General

In a 120,000 BPCD plant, tailings are fed to the pumphouse through four parallel lines from the extraction plant. Under normal operation all four of these lines are functional. The lines feed into a central distributor in the tailings pumphouse from which the tailings are fed into the pump sumps through dart valves. If the plant is operating at design capacity, five of the six tailings sumps are in operation. Recycled pond water is used to make up any variation from the design capacity in order that a constant volume of material is available for the pumps to take away.

The rubber-lined pump appears to be the most suitable type of pump for tailings slurry pumping applications. The main limitation of rubber-lined pumps is that particles 6.5 mm or greater in size tend to tear pieces off the liners and greatly reduce liner life. However, oil sands tailings are likely to contain very little plus 6.5 mm material, and that which does appear is likely to be in the form of well-rounded particles. That particle size is not likely to be a problem is demonstrated by the fact that both G.C.O.S. Ltd. and Syncrude Canada Ltd. have installed rubber-lined pumps in their tailings systems, and the G.C.O.S. experience to date is that such pumps perform satisfactorily in tailings pumping service.

A typical tailings line requires five pumps in series, each driven by an 800 HP motor. The first four pumps in series run at a fixed speed of about 540 rpm. The last pump in the series is driven through a variable speed hydraulic coupling, which is controlled by a signal from a magnetic flowmeter on the discharge line and allows the pump speed to be varied from 300 rpm to 540 rpm, as required, to maintain a specified minimum flow rate within the line. For oil sands tailings of the size distribution and density described previously, the magnetic flowmeter is set to demand a minimum flow rate of 669 L/s, or 18% above critical flow rate.

When pumping to the exterior pond, the pumphouse installations include the following line lengths and pumps:

Tailings Line Length*	Number of Pumps Main Pump House*	Number of Pumps Booster Station*	Total Horsepower
1-10900 m	5	5	8000
2- 8400 m	5	3	6400
3- 5900 m	5	0	4000
4- 5900 m	5	0	4000
5- 8400 m	5	3	6400
6-10900 m	5	5	8000
	30 Pumps	16 Pumps	36800

* Applicable to Concept 5 of 120,000 BPCD dragline mine discussed in Section 5.3.2.

Shorter line lengths and lower heads are required (in the 120,000 BPCD case) when in-pit storage is utilized. To accommodate the associated decrease in required pumping power, pumps are simply removed from each line. Otherwise the pumphouse facility remains unchanged.

H. Capital and Operating Costs

The capital and operating costs for plants and lines required in the handling of tailings, sludge and recycle water for Ore Body No. 2 (120,000 BPCD) are summarized in Section 7.4, Comparative Analysis. The costs for the 60,000 and 240,000 BPCD mine sizes appear in Sections 8.4 and 9.3, respectively. Additional costing details are provided in Chapter 6.0

5.2.2 DRY PROCESSES

A. General

Bitumen extraction by dry process has not passed the pilot stage of development. Nonetheless, it is viewed quite favourably because of its potential for higher recovery efficiency and lower environmental impact than the Clark Hot Water process. Typically the process is anticipated to involve flashing of bitumen from the oil sands at high temperatures in a retort, either stationary or rotating. Heat conservation is maintained by recirculating hot clean sand, while effluent sand is cooled to facilitate disposal.

Alternatively, a process involving anhydrous solvent extraction of bitumen might be commercially successful. The tailings are probably cooler than from the retort method but otherwise similar in handling characteristics. A possible problem created by a solvent scheme is that traces of solvent might be present in the tailings, which would complicate reclamation techniques. This study assumes that no such material is present in the tailings from a dry process.

B. Tailings Description

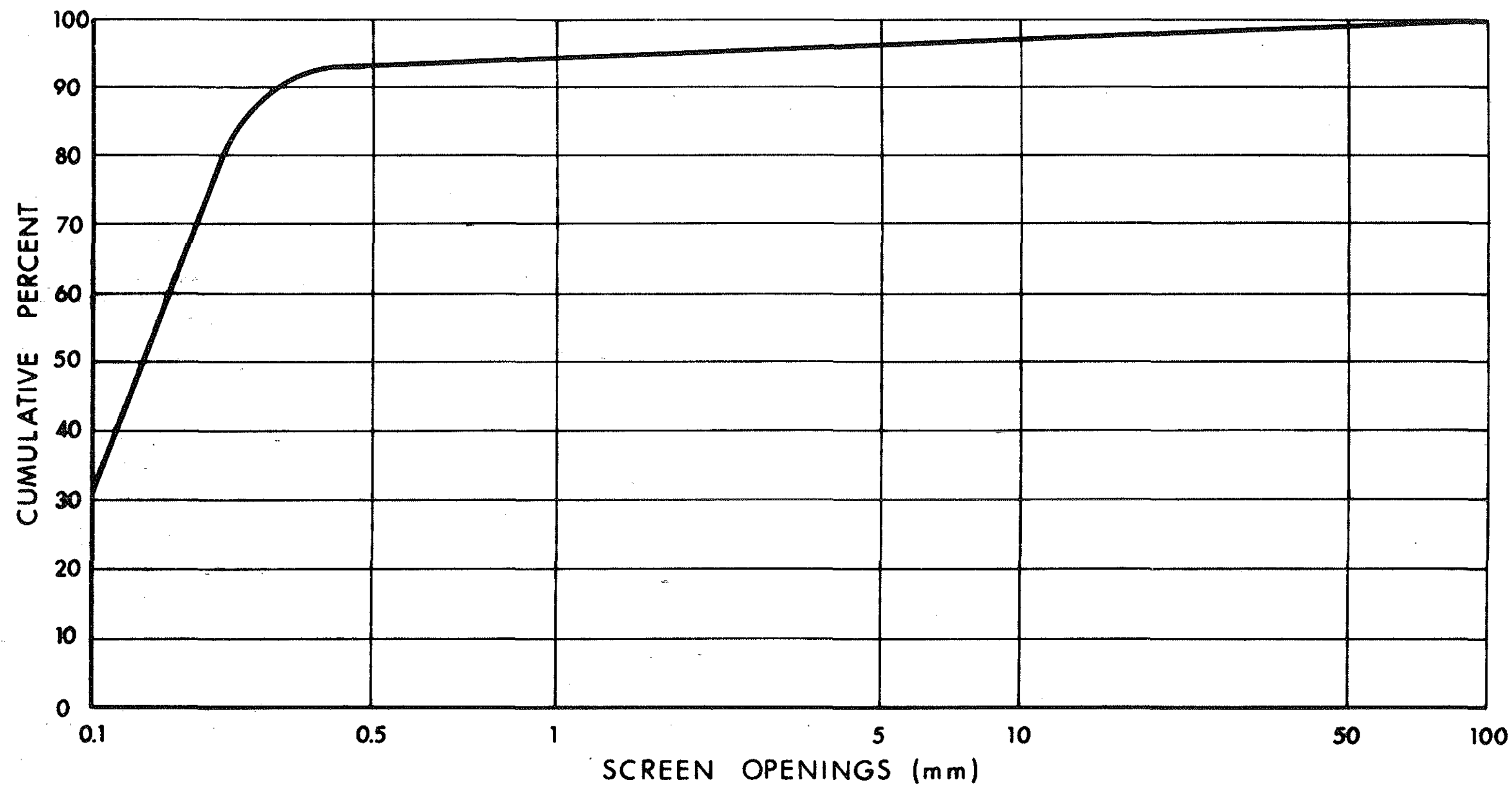
It is anticipated that some mechanism for lump and rock separation is to be incorporated into the "dry" extraction plant, but that clean sand and oversize are to be recombined at discharge. No grinding is anticipated, as the cost is probably prohibitive. The resulting size distribution is anticipated to be as shown in Figure 5.2.2-1.

The sand is essentially free of hydrocarbon, as any residue is in the form of dry coke. Dusting is probably a serious problem with this material, and conveyor transfer points require shrouding. As well, surface sprays of polymer to cake the sand surface are necessary along discharge conveyors.

For calculation purposes, the sand is assumed to have 5% moisture, but this may be lower. Temperature control is essential to avoid damage to conveyor belting, and sprayed water is likely to be utilized here.

Design characteristics are therefore as follows:

	Feed Quality	
	Normal	Worst
kg/kg oil sands	0.84	0.84
kg/m^3	1,600	1,600
% Bitumen	0	0
% Mineral	95	95
% Water	5	5
Temperature	54°C	66°C
pH	-	-



DRY PROCESS
ESTIMATED TAILINGS SIZE DISTRIBUTION

FIGURE 5.2.2-1

C. Tailings Disposition

Tailings from the dry process are conveyed by belt conveyor to disposal. Care is required to control dusting due to wind entrainment, or at transfer points. Covered conveyors may be required, or at least polymer sprays at strategic locations.

D. Capital and Operating Costs

The capital and operating cost summaries for handling dry tailings for the three mine sizes appear in Sections 7.4, 8.4 and 9.3. Additional costing details are provided in Chapter 6.0.

5.3 TAILINGS DISPOSAL TECHNIQUES

5.3.1 GENERAL DESCRIPTION OF TAILINGS DISPOSAL METHODS

Tailings generated during the extraction of bitumen from oil sands can be mechanically or physically separated into three main constituents: sand, sludge, and water. Minor amounts of process solvents and unextracted hydrocarbons may also be present, but are not of direct concern in this study. Depending on the methods of bitumen extraction, tailings disposal methods, and secondary mechanical treatment, the tailings will have different physical attributes. This study evaluates three individually distinct types of tailings classified as wet tailings, dewatered tailings, and dry tailings. The wet tailings are similar to those produced presently at commercial oil sands plants utilizing the Clark Hot Water Process. Dewatered tailings are generated by treatment of the sludge portion of the wet tailings to reduce water content and recover residual bitumen. The dry tailings are the waste product of still-to-be developed commercial extraction processes.

Wet Tailings Disposal

The primary purpose of a tailings disposal facility is to store the tailings sand in such a manner as to permanently impound the components of the plant process waste. Many mine-milling processes, including present commercial bitumen extraction techniques, require addition of large volumes of water to the plant feed. Consequently, the tailings products from such processes contain considerable amounts of water. Thus, in addition to the solid waste stored, a tailings disposal facility must impound the process water. However, large volumes of process water are usually only temporarily impounded because, once clarified by settling, the water may be recycled to the plant for re-use. Because of its toxicity, no tailings water is released to natural drainages.

Dewatered Tailings Disposal

The dewatered tailings concept is a modification of the wet tailings disposal scheme incorporating a facility to treat (in a manner as yet unproven in commercial practice) the sludge which forms in the tailings pond.

Typical wet tailings discarded by the extraction plant are pumped via pipelines to an active tailings pond, from which, after some time, settled fines (sludge) are pumped via a floating barge and pipeline to the sludge treatment plant. There, a portion of the residual bitumen is separated and pumped back to the extraction plant. The recovered sludge is treated with flocculant and centrifuged, and the thickened refuse is pumped to the treated-sludge pond for permanent impoundment. A more detailed description of the sludge treatment process has been provided in Section 5.2.1.

The main advantage of this scheme over the "wet tailings disposal" is that the sludge volume to be stored is reduced by approximately fifty percent. Another, and possibly even more beneficial advantage, is that the sludge pond may be reclaimable because the much more viscous treated sludge may allow surface layer solidification and subsequent re-surfacing with prepared soil or muskeg. Poldering techniques, such as those employed by Rheinbraun in reclamation of German lignite mines, might, with modification, be applicable. It must be noted that both the process of treating sludge and the techniques for its eventual reclamation are still hypothetical. The technology may soon be available, but initially the economics of applying that technology may prove prohibitive.

Dry Tailings Disposal

The dry tailings considered in this study are produced as waste in an extraction process which does not require the vast amounts of water needed in the Clark Process. No viable commercial process of this type yet exists, but it has been assumed to involve either high temperature bitumen extraction or bitumen extraction by solvents. The tailings consist of bitumen-free fine sand with essentially no traces of harmful chemicals. Oversize reject, also a dry product, is included in the tailings stream. The tailings disposal system in most cases incorporates transfer and disposal of centre reject from the mine.

A relatively small out-of-pit waste dump is required at the beginning of the mining operation. This out-of-pit waste dump contains centre reject and oversize reject, in addition to the dry tailings produced while the mine is expanded far enough that backfill operations can begin. Except for the initial out-of-pit waste dump, all of the waste material (overburden, centre reject, tailings sand, and oversize) is backfilled into the mined-out pit.

All waste materials are transferred on belt conveyors and placed by a spreader. Deep troughing belt conveyors and the introduction of overburden and centre reject into the tailings sand stream will minimize the need for wetting, except at transfer points. Operational experience may show that transfer point covers with several lengths of conveyor covers adjacent to a transfer point may eliminate the need for wetting. Complete covering of all conveyors is probably operationally impractical and uneconomic.

While the need for dust control wetting during conveying may not materialize, it may be necessary to incorporate some water into the tailings sand to control the placement (flow) of spreader-dumped wastes during the backfilling of the mined-out pits. Erosion and dust control can be achieved by overburden blanketing of embankment slopes which are to remain exposed for periods of time or, to a limited extent, by sprinkling. Differential placement of materials, i.e., keeping overburden separate from reject and tailings sand during certain phases of embankment construction, may also be necessary.

<u>General Site Selection Criteria for Out-of-Pit Tailings Pond</u>

In evaluating an economically feasible tailings disposal facility, a large number of factors including annual tonnage, site acreage, physical properties of tailings, type of embankment, method of waste disposal, availability of construction materials, climate, terrain, hydrology, geology, and nature of the foundation must be considered. During the initial selection of potential tailings disposal areas for this study the following criteria were developed:

1. Tailings disposal areas should be located over non-mineable oil sands;
2. Tailings disposal areas should be located as close as possible to the plant site, to minimize pumping costs;
3. Tailings disposal areas should be located so as to minimize the area affected;
4. Tailings disposal areas should be located outside major active drainages;
5. Tailings dykes should be located over foundation materials possessing adequate strength, with attention given the local groundwater regime;
6. Tailings pond should be compact in shape (long, narrow designs should be avoided).

Potential plant site locations were evaluated simultaneously with tailings disposal areas because of the interrelationship between the two facilities, i.e., the two should be located as close as possible to each other to minimize pumping costs. Plant site selection was based on similar criteria to those employed in selection of tailings disposal areas, except that the concern of placing facilities over marginally economic oil sands was relaxed basically because the plant is considered to be a temporary structure, which becomes obsolete and eventually is dismantled, after which time the underlying oil sands may be extracted. The operation of the extraction plant results in the formation of ice fog and consequently affects the plant site selction, whereas ice fog from the operation of the out-of-pit tailings pond is considered to be minor and does not influence pond site selection.

STARTER DYKES

Typically, starter dykes for out-of-pit tailing ponds are constructed during the preproduction phase to allow sufficient storage capacity in the beginning of operations when the pond area is small and pond rise is rapid. Present operations have recommended that the minimum dyke height at the beginning of a construction season be approximately 2 metres higher than the planned pond elevation at the end of that season. This criterion significantly reduces the risk of overtopping should dyke building difficulties be experienced in the early years of operation.

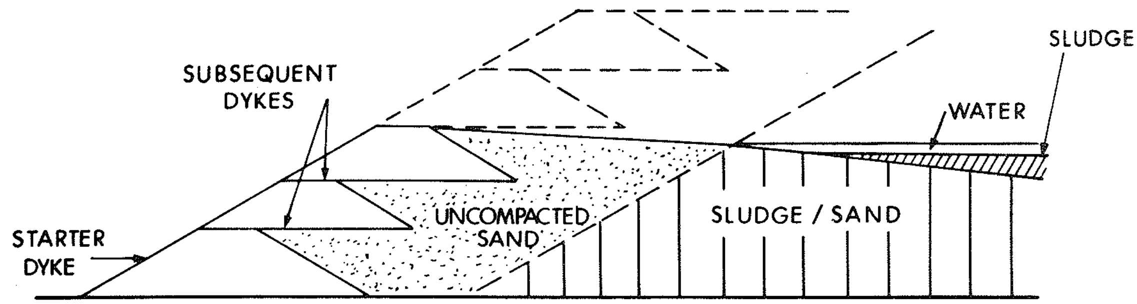
Materials used for starter dyke construction may be obtained from pre-production overburden removal at the opening of the mine unless a more economical alternative is available, such as opening a borrow pit within the limits of the tailings pond floor. The advantages of the borrow pit alternative include reduced starter dyke volume due to an increased capacity of the pond, and minimization of material haulage distances.

DYKE CONSTRUCTION METHODS[6,7,8]

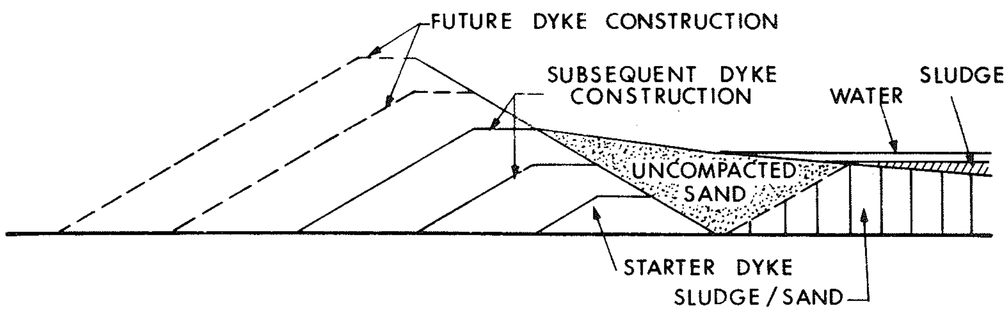
Where dykes are constructed predominantly of the coarse sand fraction of the tailings slurry three basic construction methods may be used: the downstream method, the centreline method and the upstream method (Figure 5.3.1-1). The final dyke configuration and the construction method can be modified depending on the utilization during dyke building of other materials, such as borrow materials, mine pre-stripped overburden, mine waste rock, coarse tailings, or any combination of the above.

In the downstream dyke construction method, the centreline of the dyke moves outward from the pond as the dyke is raised (see Figure 5.3.1-1). This method requires a relatively impervious starter dyke at the up-stream toe of the embankment to ensure retention of water in the pond. Cycloning may be required to separate the coarse tailings out for the dyke construction material. Drainage blankets are placed downstream from the toe of the starter dam to control seepage in the dyke.

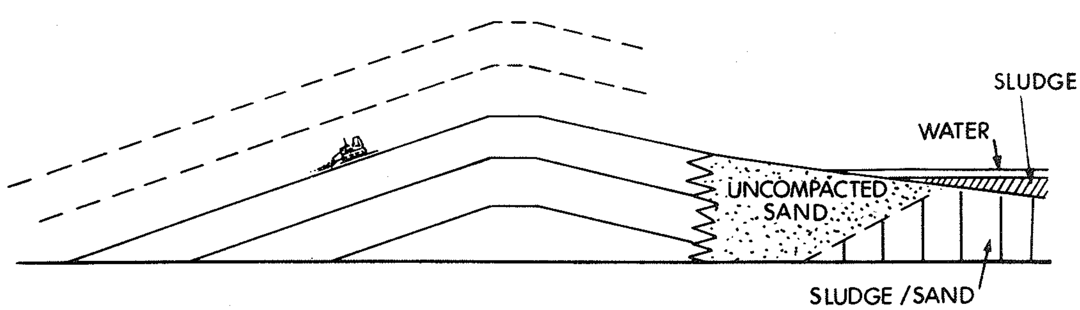
The centreline method is a modification of the downstream method (see Figure 5.3.1-1). Dyke construction is accomplished by cycloning the tailings, placing the coarse material on the dyke above the starter dyke, and mechanically pushing the material down the downstream face. The increase in additional material required in the dyke as the crest height increases is less than for the downstream method.



UPSTREAM METHOD



DOWNSTREAM METHOD



CENTRE LINE METHOD

TAILINGS EMBANKMENT CONSTRUCTION METHODS

FIGURE 5.3.1-1

(From Tentative Design Guide for Mine Waste Embankments in Canada Energy, Mines and Resources, Ottawa, 1972)

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Both the centreline and downstream methods of construction are best suited to small operations, and are too difficult to apply to oil sands tailings disposal facilities due to the large volumes of material which must be handled during dyke construction.

The upstream method of dyke construction (see Figure 5.3.1-1) yields the the most common dyke configuration. Dykes built by this method are constructed by placing successive lifts on the starter dyke in an upstream direction (towards the pond). The starter dykes forming the downstream toe of the dyke are either more pervious than the rest of the dyke, or contain filters to prevent the phreatic surface within the dyke from intersecting the downstream face. This method requires the least amount of dyke material, can easily employ tailings sand for dyke construction, and is the simplest configuration to construct. The stability of the final dyke is dependent on the shear strength characteristics of the tailings deposited upstream of the dyke, which are governed, in turn, by the gradation and the density of the solids, the consistency of the slurry, and the distribution of pore water pressures within the deposit after sedimentation. The design of dykes constructed by the upstream method must be based on conservative assumptions regarding the rate of pore pressure dissipation within the tailings. For a given set of design parameters, a dyke constructed by the upstream construction method will have a lower factor of safety than a dyke constructed by the downstream or centreline methods. If the stability of the structure is dependent on the shear strength of the tailings that remain loose and saturated, the embankment may be subject to failure by liquefaction.

Present oil sands operations employ a modified upstream stepover dyke construction method. The tailings dykes are constructed from a hydraulically-placed coarse sand fraction, which is separated from the sludge in shallow sedimentation cells continuously being constructed on the top of the dyke. Each dyke lift consists of a number of cells formed by dozing up small sand berms. Tailings are discharged from a pipe located at one end of the cell. As the tailings slurry flows down the cell, the coarse sand fraction settles out of suspension while fines (silt-and clay-size material) and water flow by gravity into a sluice box at the other end of the cell and subsequently out into the tailings pond via a pipe. Dyke sand is compacted by mechanical methods to yield a geotechnically stable structure. Dyke building is generally carried out only in the summer months, mainly because of the limited visibility created in winter months resulting from a combination of the hot tailings discharged and low ambient air temperatures. Other reasons for avoiding winter cell construction include poor compaction of the freezing sand, and possible freeze-up of compacting equipment (dozers).

The tailings pond dyke shown in Drawing No.BR22933-15-00 is an illustration of the modified upstream stepover method, which Techman/RC considers to represent the most effective utilization of a compacted material in the construction of a structurally sound and stable dyke. This method is suitable for the large volumes of tailings material produced in an oil sands operation. The main advantage of this method is that the factor of safety of the dyke is increased by compacting each subsequent lift. The full base of the dyke is compacted, i.e., there is no uncompacted sand underneath a compacted portion of dyke as is the case in the regular upstream method. The inclusion of sludge pockets near the face of the dyke may result in surface failures requiring repair of the slope surface. No pumping of water and sludge is required from the cells as they are constructed in such a manner that gravity moves the fluids into the pond. Also no major rehandling of sand is required. Above water beach slopes of 30:1 to 20:1 can be expected. Below water level the slopes are steeper, being in the range of 20:1 to 10:1. Depending on pond shape, ground surface topography, and volumes of tailings, various compound slopes are possible. The tailings pond designs in this study are based on 10:1 beach slopes since the rate of dyke building is always the minimum allowable, thus maximizing the submerged portion of the beach. The main disadvantage of the suggested modified stepover method is the large volume of material required initially to construct the wide base of the dyke. As beach slopes become more shallow, the height and volume of the starter dyke must be increased.

OUT-OF-PIT TAILINGS DYKE CONSTRUCTION

Scheduling of dyke construction for a tailings disposal facility is of prime importance to the pond design. Sufficient sand must be available during the summer months to construct enough dyke to contain the annual volume of sludge plus the sand that is overboarded during the winter. This problem is particularly critical in the initial years when the pond area is usually small and pond surface rise is rapid.

Scheduling difficulties may persist throughout the life of the mine if detailed and accurate mass balance calculations have not been made and completely documented. Out-of-pit and in-pit tailings disposal requirements must be calculated for the entire life of the mine to ensure that the size of the initial out-of-pit ponds is adequate, and that in-pit dykes are built in the most suitable location on time. Scheduling in the original plan must also consider changes in the long-term requirements for tailings disposition.

Out-of-pit tailings dykes located on the stronger foundation materials can have an overall outboard slope angle of 3h:1v. However, if poorer foundation conditions are encountered, the outboard slopes of the dykes must be reduced accordingly. For planning purposes an outboard slope of 4h:1v is used for out-of-pit dykes except where Clearwater shales are definitely known to exist within 10 m of the ground surface. Muskeg removal is considered unnecessary, provided that the dyke slope does not exceed 4h:1v.

IN-PIT TAILINGS DISPOSAL

To minimize out-of-pit storage requirements, tailings disposal is diverted to the mined-out pit as soon as sufficient space is available. In most mine plans, diversion of tailings to the pit is not possible until approximately 10 to 13 years after start-up, depending on the shape of the mine. At this time, sufficient space should be available in the pit to allow disposition of tailings throughout the remaining life of the operation. Successive tailings dykes are constructed in the pit as the mining advances.

Usually it is possible to construct in-pit dykes with tailings sand. In some cases, overburden may be required to supplement the available sand. In-pit sludge pond dykes at the Minimum Level may also be constructed from tailings sand. However, the dykes for thickened sludge (Improved Level) ponds must be constructed from overburden since the introduction of tailings water into the the thickened sludge pond would dilute the sludge.

Depending on foundation conditions, in-pit tailings dyke slopes of 6h:lv and 5h:lv were used for mine planning. The effects of foundation re-loading rate (i.e. rate of dyke construction), and the length of time between oil sands removal and dyke construction (i.e., amount of pit bottom heave and pore pressure reduction during that time) are unknown. Consequently, a shallow dyke slope angle of 6h:lv has been employed to ensure dyke stability on weak pit floors. The risk of liquefaction within the dyke itself, caused by an excessively rapid buildup of load compared to pore pressure dissipation must also be considered.

Present oil sands operations plan to reclaim the sludge from such in-pit impoundments, and either return it to the main surface storage area, or centralize sludge storage in one area of the pit. The latter plan facilitates reclamation of the out-of-pit tailings disposal areas, as ponds containing only sand tailings present less of a problem.

SANDING-IN TECHNIQUES

The liquid pond surface of conventional tailings ponds can be eliminated by a sanding-in beaching procedure. A gently sloping sand beach exists between the tailings pond dyke and fluid reservoir. This beach can be extended by gradually spigotting tailings further into the pond. Eventually, the entire water and sludge volume can be displaced by the spigotted sand. The possibility exists that gradual local variations can be made in the surface topography of the sanded-in pond to assist in the control of surface runoff. Overall, either a shallow dish-shaped surface or a rise can be formed. Dish-shaped surfaces would collect water in the lowest portion of the sanded-in surface. As the reservoir volume is reduced by displacement with the sand, difficulties may develop with clarification of water. The flow of recycle water may need to be interrupted once insufficient clarification occurs. If another tailings pond or an undewatered-sludge pond is already in operation, the recycle water can be pumped to this pond. If no other pond is available, then alternate disposal schemes must be considered. The difficulty of disposing of residual water exists with all wet tailings disposal schemes. It is not clear whether there would be a net regional accumulation of tailings water that cannot be recycled with the successive closing of oil sands mines.

In some mines it may not be possible to deposit tailings against pit walls. In this case, sanding-in can be achieved by the successive construction of narrow strip-like ponds[9]. A large portion of the sand volume in such a pond is used to construct the dyke. Sludge and water is pumped from behind the dyke to another containment facility and the void filled with spigotted sand.

5.3.2 TAILINGS DISPOSAL AT THE MINIMUM LEVEL OF RECLAMATION

A large number of mine plans and associated reclamation schemes are possible at each of the levels of reclamation in this study. These, of course, vary in cost and design and provide for somewhat different final landscapes and land uses. It should be remembered that the main purpose of Chapter 5 and, in particular, the following sections, is to demonstrate the impact of selected tailings disposal schemes on the Levels of Reclamation. In this chapter, the 120,000 BPCD mine (1 bucket wheel excavator and 4 draglines) serves as illustration, leaving the discussion of the application of similar schemes to both smaller and larger mines for Chapters 8 and 9.

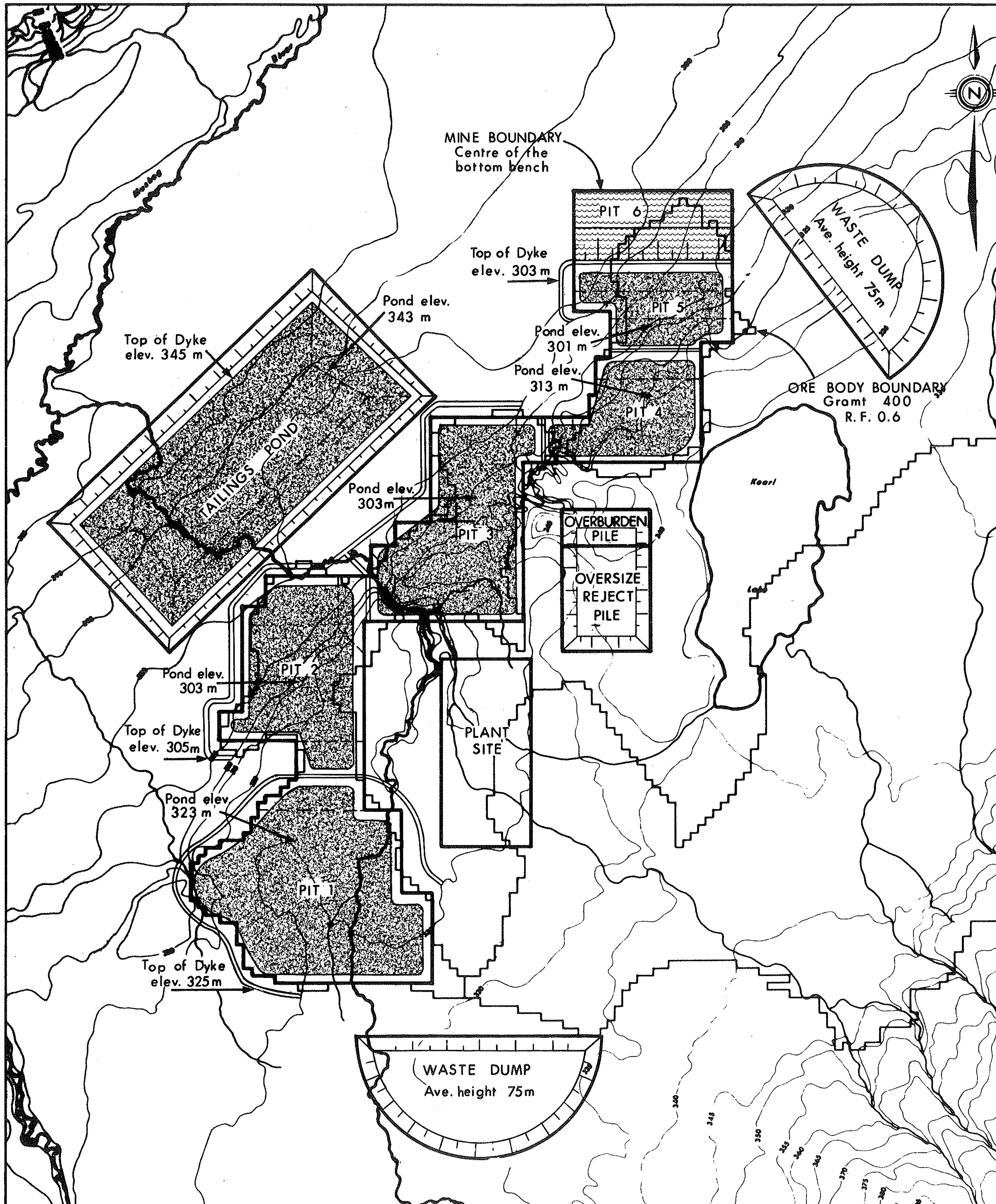
A number of criteria have been selected for the development of reclamation plans at the Minimum Level. These criteria, along with those established for the Improved and Enhanced Levels can be found in Table 4.1-1.

Five conceptual tailings disposal schemes have been outlined for the Minimum Level of Reclamation. The five concepts pertain to wet tailings, i.e., combinations of sand, sludge, and water. Concept No. 1 is the simplest disposal scheme presented, and is included only as a base case against which comparisons can be made. The concepts are illustrated graphically in the Figures 5.3.2-1 to 5.3.2.-5 "120,000 BPCD - Tailings Disposal and Reclamation Minimum Level, Concepts 1 to 5." A summary and comparison of the main features of the five concepts are tabulated at the end of this section.

Concept No. 1 (Base Case)

In this scheme, mining proceeds from south to northeast with tailings initially directed to the out-of-pit tailings pond, and later pumped to in-pit ponds dyked off from active mining operations. The final remaining pit is filled with water. No special attempt is made to separate or concentrate the various tailings components (sand, sludge, and water). As the volume of tailings generated is greater than the available disposal volume in the pit, the out-of-pit pond must be designed to contain the excess volume through the initial years of operation, while mining proceeds to the point where the first in-pit dyke can be constructed. (See Figure 5.3.2-1.)

For economic reasons, and for foundation stability, dykes are constructed at the narrowest locations of the pit. The dykes are constructed from the hydraulically-placed tailings sand, which is subsequently compacted. Water sands and basal clays underlying the bottom reject are considerations in the overall in-pit dyke stability. In addition to the reduction of in-pit dyke slopes, other measures such as dewatering of underlying water sands may be necessary to prevent dyke foundation failure. Dewatering may help in relieving possible pore pressure buildup in the basal clays. It should be remembered that these consolidated or overconsolidated basal clays are deeply buried until the overlying overburden and oil sands are removed. Since in-pit dyke construction follows the mining face quite closely, the amount of rebound in the basal clays may be minimized and subsequent reloading by the weight of the



NOTES

- NO REVEGETATION OF ANY AREA
- 8½ YEARS TAILINGS TO TAILINGS POND (2400 m x 5000 m)
- TAILINGS POND STARTER DYKE USES BORROW MATERIAL FROM TAILINGS POND CENTRE.
- TAILINGS POND FULL WHEN MINING HAS PROGRESSED SUFFICIENTLY TO ALLOW DYKE TO BE CONSTRUCTED AT THE NARROW PART BETWEEN PIT 1 AND PIT 2.
- TAILINGS DISPOSAL THEN DIVERTED TO IN-PIT POND.
- IN-PIT PONDS CREATED AS MINING PROGRESSES NORTH.
- DYKES CONSTRUCTED (SUMMER ONLY) OF HYDRAULICALLY PLACED TAILINGS SAND AND COMPACTED.
- IN-PIT PONDS REQUIRE SOME DYKING ALONG THE WEST EDGE AND DRAINAGE DITCHES ALONG THE EAST EDGE.
- SLUDGE COVERED BY 3 m OF TAILINGS WATER.
- TAILINGS POND HAS 2 m FREE BOARD.
- PIT 6 IS AN EMPTY END PIT WHICH IS FILLED WITH WATER TO MAKE A LAKE.
- PLANT SALVAGED BUT PLANT SITE NOT RECLAIMED.
- NO TOXIC MATERIALS EXPOSED, BUT ALSO NO ARTIFICIAL SOIL PLACEMENT.
- SITE LEFT FOR NATURAL PLANT INVASION.
- THIS CONCEPT CONSIDERED THE BASIC CASE (UNACCEPTABLE)

LEGEND

SLUDGE WITH TAILINGS WATER ON TOP

UNTREATED DYKES, BEACH, OVERBURDEN AND OVERSIZE REJECT PILES, PLANT SITE

EMPTY END PIT (FILLS WITH WATER)

METERS

1000 500 0 1000 2000 3000

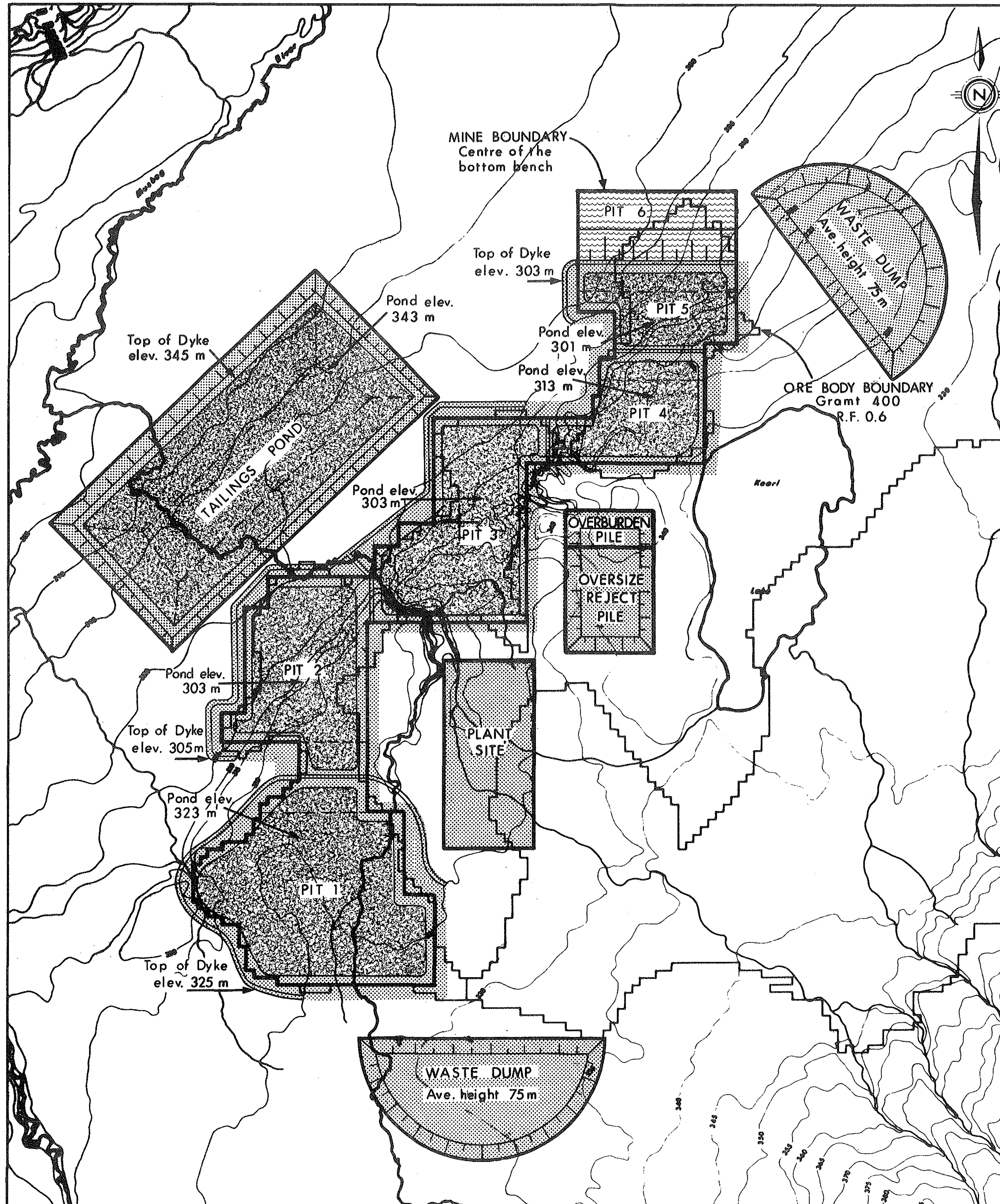
FIGURE 5.3.2-1

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT

PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH

TITLE: 120,000 BPCD
TAILINGS DISPOSAL AND RECLAMATION
MINIMUM LEVEL - CONCEPT No. 1

SCALE: AS SHOWN	DATE: AUG., 1978	DRAWN BY: TM ☒ RC ☐	TM DRAWING No.
APPR. PROJ. ENGR.	APPR. PROJ. MNGR. TM. ___ RC. ___	PROJECT No: TM ☒ RC ☐ TM - 229	RC DRAWING No.



NOTES

- TAILINGS DISPOSAL SCHEME SAME AS IN CONCEPT NO. 1 (MINIMUM LEVEL)
- PILES AND EXPOSED PIT SLOPES STABILIZED AT 3:1
- CAP OVERSIZE REJECT PILE WITH NON-OIL SAND-BEARING OVERBURDEN PRIOR TO RE-SOILING
- PLANT SITE SALVAGED AND COVERED WITH SUITABLE OVERBURDEN
- DYKES, BEACHES, PILES AND PLANT SITE COVERED WITH 0.4 m OF MUSKEG AND SELECTED OVERBURDEN MIXTURE.
- RECLAMATION SURFACES PLOWED 0.6 m DEEP
- SEEDED WITH GRASSES AND LEGUMES FOR EROSION CONTROL
- SHRUBS AND TREES PLANTED LATER IF CONVENIENT

LEGEND

- SLUDGE WITH TAILINGS WATER ON TOP
- 0.6 m Soil — TREATED DYKES, BEACH, OVERBURDEN AND OVERSIZE REJECT PILES, PLANT SITE
- EMPTY END PIT (FILLS WITH GROUND WATER)

METERS
1000 500 0 1000 2000 3000

FIGURE 5.3.2-2

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT

PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH

TITLE: 120,000 BPCD
TAILINGS DISPOSAL AND RECLAMATION
MINIMUM LEVEL - CONCEPT No. 2

SCALE: AS SHOWN	DATE: AUG., 1978	DRAWN BY: TM ☒ RC ☐ S.M.	TM DRAWING No.
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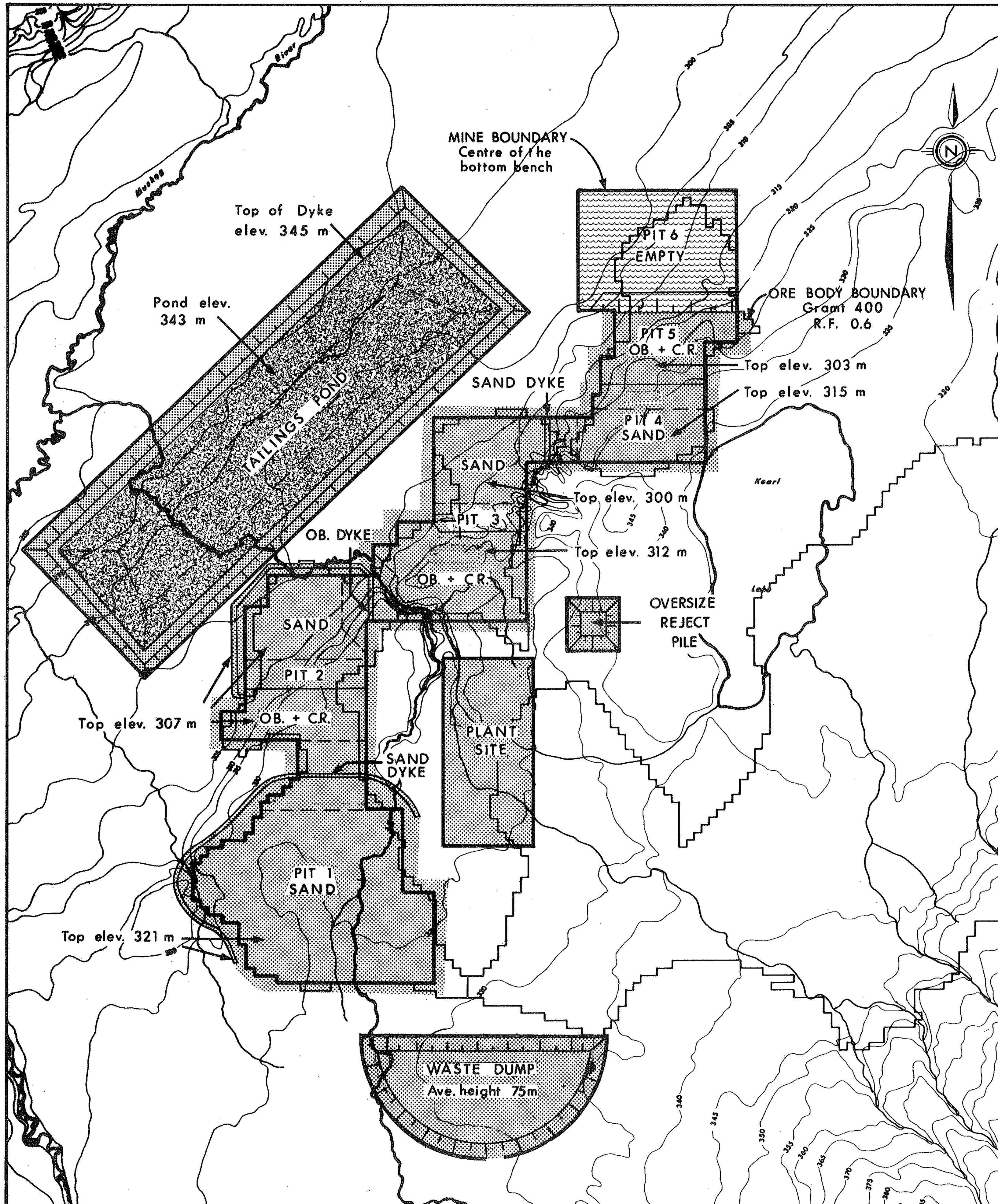


NOTES

- TAILINGS POND (2400 m x 7000 m) MUST CONTAIN 25 YEARS OF SLUDGE, 3 m OF WATER AND 2m FREE BOARD
- TAILINGS POND COVERS SOME ORE AT THE NORTH END
- TAILINGS POND DYKES AND BEACH TAKE 8½ YEARS OF TAILINGS SAND
- TAILINGS POND STARTER DYKE USES BORROW MATERIAL FROM TAILINGS POND CENTRE.
- OVERSIZE REJECT TRUCKED FROM PLANT TO PITS AFTER FIRST IN-PIT DYKE CONSTRUCTION STARTED (8½ YEARS)
- CENTRE REJECT FROM LOWER BENCH IS BACKCAST ONTO THE PIT FLOOR
- SLUDGE FROM PITS PUMPED TO TAILINGS POND
- PITS CONTAIN BACKFILLED OVERBURDEN, CENTRE REJECT AND TAILINGS SAND
- RECLAMATION AND VEGETATION ESTABLISHMENT SAME AS IN MINIMUM LEVEL - CONCEPT NO. 2

LEGEND

SLUDGE

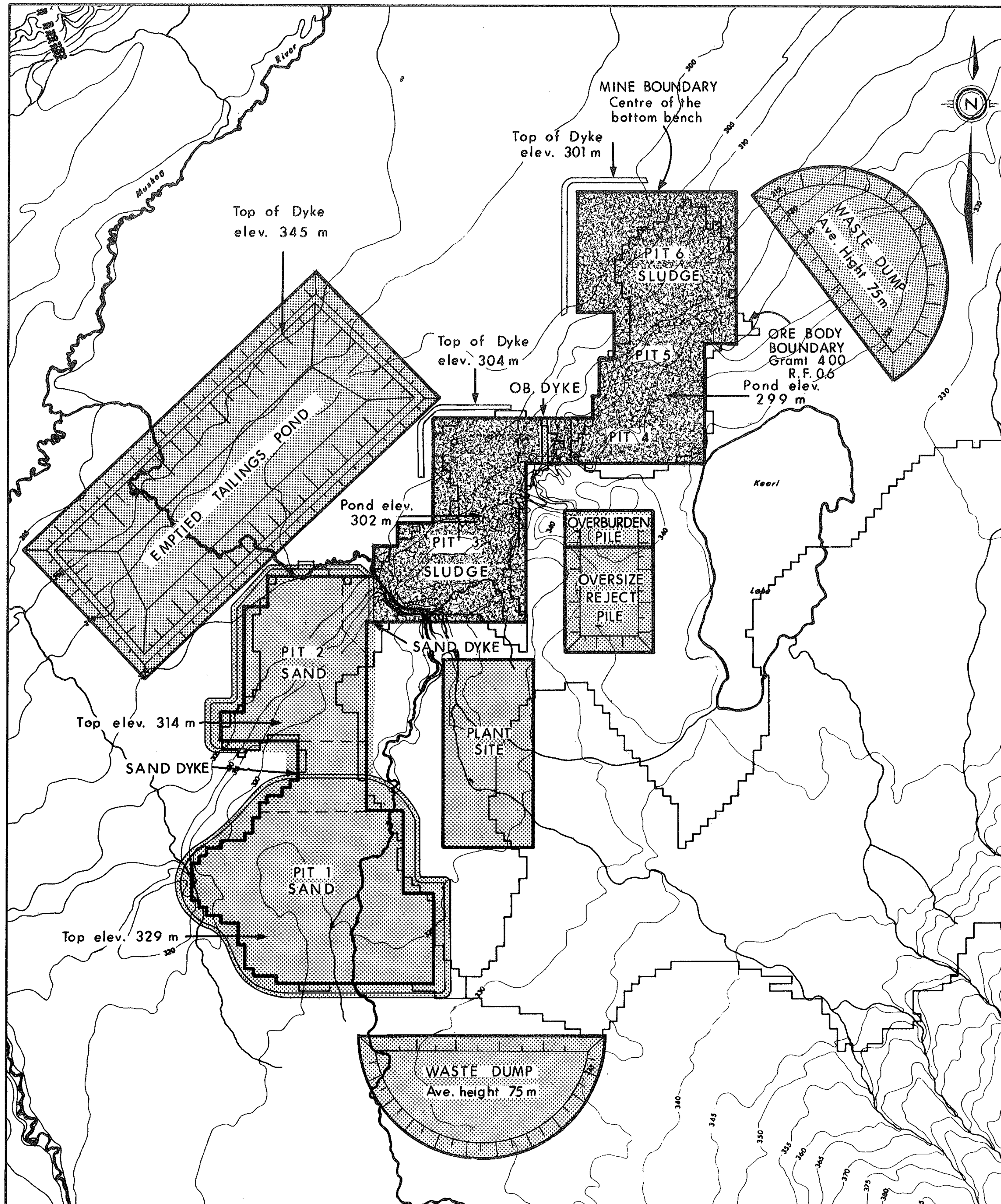
0.6m Soil — TREATED DYKES, BEACH, OVERBURDEN AND OVERSIZE REJECT PILES, PLANT SITE

EMPTY END PIT (FILLS WITH WATER)

METERS
1000 500 0 1000 2000 3000

FIGURE 5.3.2-3

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT			
PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH			
TITLE: 120,000 BPCD TAILINGS DISPOSAL AND RECLAMATION MINIMUM LEVEL - CONCEPT No. 3			
SCALE: AS SHOWN	DATE: AUG, 1978	DRAWN BY: TM ☒ RC ☐ S.M.	TM DRAWING No.
APPR. PROJ. ENGR.	APPR. PROJ. MNGR. TM. RC.	PROJECT No.: TM ☒ RC ☐ TM - 229	RC DRAWING No.



NOTES

- EXTERIOR TAILINGS POND (5500 x 2400 m) RECEIVES TAILINGS FOR 8½ YEARS.
- AFTER THAT PIT 1 IS BACKFILLED WITH TAILINGS SAND WHILE SLUDGE AND WATER ARE REPUMPED TO TAILINGS POND.
- SLUDGE AND WATER FROM PIT 2 TO PIT 3.
- SLUDGE AND WATER FROM TAILINGS POND TO PITS 4, 5 AND 6
- 16½ YEARS OF TAILINGS INTO PITS 1 AND 2.
- LOWER BENCH CENTRE REJECT BACKCAST ONTO PIT FLOOR
- OVERBURDEN, UPPER BENCH CENTRE REJECT AND OVERSIZE REJECT FORM STOCKPILES ABOUT 75 m HIGH.
- RECLAMATION AND VEGETATION ESTABLISHMENT SAME AS IN MINIMUM LEVEL - CONCEPT NO. 2.

LEGEND

SLUDGE WITH TAILINGS WATER ON TOP

0.6 m Soil — TREATED DYKES, BEACH, OVERBURDEN AND OVERSIZE REJECT PILES, PLANT SITE

EMPTY END PIT (FILLS WITH WATER)

METERS

1000 500 0 1000 2000 3000

FIGURE 5.3.2-4

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT

PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH

TITLE: 120,000 BPCD TAILINGS DISPOSAL AND RECLAMATION MINIMUM LEVEL - CONCEPT No. 4

SCALE: AS SHOWN	DATE: AUG, 1978	DRAWN BY: TM RC S.M.	TM DRAWING No.
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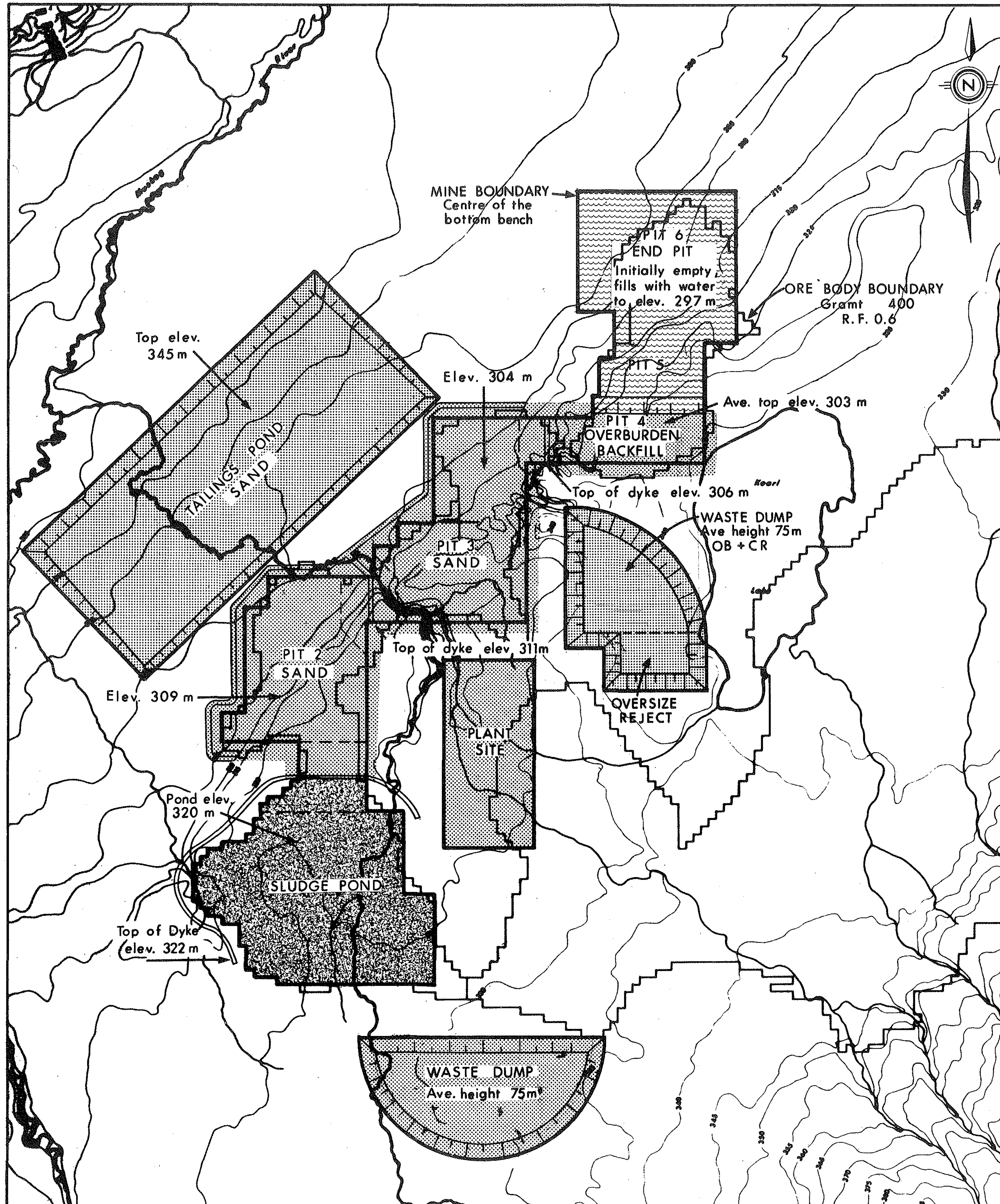

NOTES

- 14½ YEARS OF TAILINGS SAND TO TAILINGS POND (2400 m x 5400 m)
- 5 YEARS OF TAILINGS SAND INTO EACH OF PITS 2 & 3
- ALL SLUDGE FROM TAILINGS POND AND PITS PUMPED TO SLUDGE POND (STARTING IN YEAR 10)
- TAILINGS POND DYKES OUTBOARD SLOPE 4:1
- IN-PIT DYKES SLOPE 6:1, MUST NOT BE BUILT ON BACK-CAST LOWER BENCH CENTRE REJECT (CONVEYED TO OVERBURDEN PILES)
- CENTRE REJECT FROM LOWER BENCH IS BACKCAST ONTO THE PIT FLOOR
- OVERBURDEN AND UPPER BENCH CENTRE REJECT BACKFILLED TOWARDS THE END OF MINING (YEARS 18 - 25)
- RECLAMATION AND VEGETATION ESTABLISHMENT SAME AS IN MINIMUM LEVEL - CONCEPT NO. 2

LEGEND

SLUDGE WITH TAILINGS WATER ON TOP

0.6 m Soil — TREATED DYKES, BEACH, OVERBURDEN AND OVERSIZE REJECT PILES, PLANT SITE

EMPTY END PIT (FILLS WITH WATER)

METERS
1000 500 0 1000 2000 3000

FIGURE 5.3.2-5

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT

PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH

TITLE: 120,000 BPCD
TAILINGS DISPOSAL AND RECLAMATION
MINIMUM LEVEL - CONCEPT No. 5

SCALE: AS SHOWN	DATE: AUG., 1978	DRAWN BY: TM ☒ RC ☐ SM.	TM DRAWING No.
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dyke will return the vertical stress state of the basal clays underneath the dykes to approximately the level existing before mining.

A detailed geotechnical investigation is required to determine a suitable rate of basal-clay reloading. In this study, it was assumed that the rate of in-pit dyke construction is primarily governed by the material balance and the mining schedule (i.e., in Concepts No. 1 and 2, some in-pit dykes must be built quite rapidly: for example, the Pit 4 dyke in two summers and the Pit 5 dyke in one-and-a-half summers). Unless shown to be geotechnically sound, dykes are not built on backcast centre reject. The conveyor systems are designed to allow centre reject to be taken to overburden piles from both mining benches. The bottom bench centre reject is backcast onto the pit floor whenever the placement of reject does not interfere with dyke building.

The in-pit storage capacity is reduced by the volume of bottom-bench centre reject that is backcast onto the pit floor, and amounts to an average 6 m thick layer. Additional volumetric capacity for in-pit wet tailings storage results from the sloping topography. Since the surface of the wet tailings pond is horizontal, the amount of stored tailings is governed by the lowest edge of a pit. To offset this to some degree, dykes are constructed for the 120,000 BPCD mine along the western edges of the pits.

The out-of-pit tailings pond in this scheme is 2,400 m wide, 5,000 m long and up to 60 m high. Its starter dyke requires 22,600,000 m^3 of selected overburden material. The starter dyke ranges in height from 15 m along the west side to zero along the east side. A large starter dyke is necessary so that sufficent amounts of tailings sand are available for beaching in the initial year. The dyke is protected on the inside by beached sand, which pushes sludge away from the toe of the dyke and prevents it being deposited there. The out-of-pit tailings pond has dykes constructed with an outside slope at 4h:1v to allow for the presence of shallow discontinuous muskeg on which the major part of the dyke is built.

In this and all subsequent concepts, artesian groundwater conditions further complicate stabilizing measures by being a source of recharge in areas filled with in-pit tailings, even though intensity of inflow is greatly reduced by having placed relatively impermeable centre reject materials on the pit floor. Inflow from aquifers within the ore zone into or through the mined out areas may also occur. The minimization of this effect requires the placement of a vertically impermeable barrier zone, possibly formed by overburden or reject oil sands. Solutions to this potential problem require much more detailed knowledge of the geology and groundwater characteristics than is currently available. More importantly, the solution to groundwater inflow problems is different for each mine, a cluster of mines, or for a regionally-developed mine. The problem of groundwater recharge cannot be adequately addressed in this study because comprehensive hydrogeological information for the region is not available, the range of practical experience is limited, and the overall regional development plan is undefined.

At the conclusion of mining, the site would be abandoned and left for natural plant invasion to revegetate. In this and all subsequent concepts, artesian groundwater conditions could further complicate stabilizing measures by being a source of recharge in areas filled with in-pit tailings.

Concept No. 2

For this option, mining and tailings disposal are as in Concept No. 1; however, revegetation after abandonment is attempted. As dykes and waste dumps are completed, they are covered with a 0.4 m layer of muskeg-overburden mixture, which is then cultivated to a depth of 0.6 m using agricultural implements. Roughly equal amounts (0.2 m) of the two materials are spread. The surface is seeded with grasses and legumes (for erosion control), and later planted with shrubs and trees (see Figure 5.3.2-2).

Overburden dumps must be constructed so that lean oil sands are not exposed on outer surfaces. Dump slopes are expected to be geotechnically stable at about 3h:1v. Exposed pit walls must be recontoured to shallower overall slope angles. Any exposed oil sands must be covered with overburden prior to the application of the muskeg. The oversize plant reject pile is to be capped with 1.0 m of oil-free sandy overburden material prior to the spreading of prepared soil. The minimum recommended thickness of prepared soil is again 0.4 m. All roads, pipelines, transmission lines, and construction sites are to be graded. Stream diversions are to be left in place and adequate surficial drainage channels are to be constructed to control water erosion and ensure mine- site drainage.

Concept No. 3

At the initiation of mining, all tailings are pumped to the out-of-pit tailings pond. When sufficient space is available in the mine the tailings are disposed of in the pit (see Figure 5.3.2-3). As mining progresses and successive in-pit ponds are constructed, sludge and water are pumped out of these ponds and back into the out-of-pit tailings pond to allow complete backfilling of the pits with tailings sand, overburden, and centre reject. Concept No. 3 has the largest out-of-pit tailings pond of all the concepts because the pond must have the capacity to store 25 years production of sludge. The extra volume stored out-of-pit allows more than half the overburden and centre reject from the upper bench to be backfilled into the mined- out pits rather than in outside waste dumps. Dykes are constructed along the lower pit boundary at the southern and central parts of the mine to increase tailings storage capacity in the pit. Backfilling starts while the first in-pit dyke is being constructed, and the backfill material is placed between the dyke and the mining face. The disposal sequence (based on averages) results in the following:

Out-of-Pit Tailings Pond accommodates:
- 8 1/2 years coarse sand tailings
- 25 years of sludge

29,590,000 m^3 of suitable material is required for the starter dyke

Outside Overburden Dump is composed of:

- 9 1/2 years of overburden and upper-bench centre reject

Pit No. 1:

- contains 10 years of tailings sand
- all sludge and water pumped to out-of-pit tailings pond
- pit completely filled with sand
- Pit 1 dyke constructed of hydraulically-placed and compacted tailings sand.

Pit No. 2:

- contains 2 1/2 years of tailings sand
- contains 4 1/2 years of overburden and upper-bench centre reject
- overburden forms Pit 2 dyke
- all sludge and water are pumped to out-of-pit tailings pond
- pit completely filled with tailings sand and overburden/centre reject.

Pit No. 3:

- contains 2 years of tailings sand
- contains 7 years of overburden and upper-bench centre reject
- Pit 3 dyke constructed of hydraulically-placed and compacted tailings sand
- all sludge and water are pumped to tailings pond
- pit completely filled with tailings sand and overburden/centre reject

Pit No. 4:

- contains 2 years of tailings sand
- contains 3 years of overburden and 5 years of upper-bench centre reject
- overburden forms Pit 4 dyke
- all sludge and water are pumped to tailings pond
- pit completely filled with tailings sand and overburden/centre reject.

Pit No. 5:

- initially empty, filled with water (possibly from Athabasca River)

Centre reject from the bottom bench is backcast onto the pit floor as in all other concepts. It forms a layer 6 m thick. Oversize reject is redirected from the surface dump into the mined-out pits after the first in-pit dyke is started (in year 8 1/2).

Recontouring, soil placement and revegetation are as outlined in Concept No. 2.

Additional costs are incurred in this scheme through pumping of sludge and water from in-pit ponds to the out-of-pit pond. However, the surface area of wet tailings is reduced to less than half that of Concept No. 2, and about half of the outside overburden pile is eliminated because overburden can now be backfilled into the pit. Present oil sands mines tend to be relatively shallow, and when they are subsequently used for tailings disposal, tend to store material in such a way that large amounts of tailings pond surface areas remain. On the other hand, deeper mines (probably higher stripping ratios) are more desirable for in-pit tailings disposal, since the net pond surface area is reduced. In Concept 3, only one final sludge pond remains as an out-of-pit tailings pond, while most of tailings sand and overburden are impounded in the mined-out pits. In such a system, the sludge-to-water ratio is maximized, and more water can be recycled. However, the out-of-pit pond is considerably above the adjacent ground surface, and in the context of geological time, may eventually fail.

An interesting variation of Concept No. 3 is to construct additional elevated sludge ponds on top of backfilled overburden and reject. This may, in fact, be necessary when the outside tailings pond cannot be economically built large enough to contain all the sludge. The risks associated with the groundwater flow through the liquid portion of in-pit tailings ponds (the formation of springs) are then considerably reduced.

Concept No. 4

In this concept, the out-of-pit pond is used initially to store all tailings. Subsequently, Pits 1 and 2 are completely backfilled with sand, as sludge and water from these pits is pumped directly into the out-of-pit pond and Pit 3, respectively. Eventually all sludge and water is concentrated in Pits 3 and 4. To allow sufficient storage capacity in Pits 3 and 4 for all the sludge and water, no backfilling of overburden or centre reject (from upper bench) is possible in these pits. No fresh water lake is left in the pit for this option. The drained dish-shaped out-of-pit pond is eventually resurfaced with a prepared soil.

In Pits 1 and 2, sand is backfilled to a considerably greater depth than in other concepts, to ensure that there is sufficient volumetric capacity (with only minor dyking) in Pits 3 and 4 for retention of all the sludge.

Pit 3 is dyked off from the active mining face at Year 18, at which time sludge pumping begins. Pit 3 is full of sludge at the end of Year 25. After Year 25, sludge and water from the out-of-pit tailings pond is pumped into Pit 4. The disposal sequence (based on averages) results in the following:

Out-of-pit Tailings Pond:

- initially contains 8 1/2 years of tailings
- after 8 1/2 years, receives 9 1/2 years of sludge from Pit 1 (sludge rehandle)
- sludge is pumped to Pit 3 between Year 19 and 25
- remainder of sludge and water is pumped to Pit 4 (after Year 25).

Pit No. 1:

- active tailings disposal starts at 8 1/2 years and ends at 19.7 years
- contains 11.2 years of tailings sand
- sludge is pumped to out-of-pit tailings pond (until end of Year 18)

- remainder of sludge is reclaimed and pumped to Pit 3
- dyke constructed of hydraulically-placed sand
- pit completely filled with sand

Pit No. 2:

- active tailings disposal starts at 19.7 years and ends at Year 25
- contains 5.3 years of tailings sand
- sludge is reclaimed and pumped to Pit No. 3
- dyke constructed of hydraulically-placed sand
- pit completely filled with sand

Pit No. 3:

- receives sludge from Pit No. 1 and 2 and out-of-pit pond from Year 19 to 25
- contains no tailings sand
- dyke constructed of overburden
- partial perimeter dyking around pit

Pit No. 4:

- after termination of mining receives the remainder of sludge plus water from the outside tailings pond
- contains no tailings sand
- partial perimeter dyking around pit

This option involves high tailings handling costs because some sludge is pumped twice: first, from the in-pit active pond to the out-of-pit pond while sanding-in Pits No. 1 and 2, then back to the Pits 3 and 4. Sludge from Pit 2 is pumped directly to Pit 3. The benefit derived from such a scheme is that all water and sludge are stored below the natural ground surface. Some measures need to be taken, however, to ensure that the emptied out-of-pit pond does not accumulate water from precipitation and runoff sources. The dyke structure may have to be opened in at least one location if accumulation of surface runoff water is to be prevented. The runoff water may contain dissolved or partially dissolved pollutants from the original tailings, and presents an environmental hazard.

Some difficulty in reclamation of the internal pond slopes may be encountered due to the "weeping" of sludge from between layers of overboarded tailings sand.

Concept No. 5

In this option, the out-of-pit tailings pond is filled with tailings sand, sludge, and water while the initial pit is being opened. When this initial pit is large enough to accommodate the total volume of sludge generated during 25 years of mining, a dyke is constructed. Sludge is then recovered from the out-of-pit pond and pumped into the in-pit sludge pond. Active tailings disposal continues in the out-of-pit pond as sludge is removed, until eventually this pond is completely filled with tailings sand. The yearly rate of sludge pumping from the out- of-pit tailings pond to the sludge pond is 2 1/2 times higher than the average yearly sludge production. When the entire out-of-pit pond is filled with tailings sand, active tailings disposal is diverted into the pit. Sludge is transferred from these in-pit ponds to the initial in-pit sludge pond, and tailings sand is accumulated as backfill. As in all the previous cases, except Concept No.4, the final empty pit remaining is a consequence of logistics rather than intentional design. This pit is filled with water (see Figure 5.3.2-5).

The out-of-pit tailings pond (2,400 m x 5,400 m) in Concept No. 5 is somewhat larger than that in Concepts No. 1 and 2, because tailings disposal in the pit is delayed by 6 years as result of the utilization of the first mined-out pit as a sludge pond. The out-of-pit pond contains 6 1/2 years of tailings sand, in addition to the total tailings (i.e., sand, sludge, and 3 m of tailings water) for the initial 8 1/2 years.

In this option, in-pit dykes need not be built as fast as in the first two concepts because volumetric capacity for all 25 years' sludge is made available prior to in-pit tailings disposal, thereby reducing subsequent pond storage requirements. As a result, possible pore pressure buildup in the basal clays may not be as critical during in-pit dyke construction. Dykes are not built on backcast centre reject as this may jeopardize foundation stability.

The larger out-of-pit tailings pond and the higher perimeter dykes for in-pit tailings sand storage (Pits 2 and 3) result in some pit space being left for overburden and centre reject backfill during the final years of mining. The overburden backfilling is advantageous to the reclamation scheme, as it reduces out-of-pit waste dump requirements. Calculations indicate that approximately the last 5 years of overburden and 6 years of centre reject mined on the upper bench can be backfilled into the mined-out pit. When the bottom bench clears Pit No. 3 (after 17.7 years), these materials form an in-pit backfill that, with proper compaction and suitable material selection, functions as the Pit 3 dyke.

The out-of-pit tailings pond receives 14.6 years of sand. The sludge pond (Pit 1) receives 25 years of sludge, and Pits 2 and 3 each contain approximately five years of tailings sand production. In addition, each pit contains an average 6 m layer of centre reject backcast onto the pit floor by the lower-bench draglines. Tailings water is recycled back into the plant from each pond (including the sludge pond) as soon as it is clarified to about 3 m deep. There are interruptions in water recycling each time a new pit is opened, as sufficient water must be accumulated before pumping can begin. Interruptions last for about 3/4 year. Such interruptions will occur in all tailings disposal systems where a series of ponds are utilized. Water is recycled from the out-of-pit tailings pond after about 1 1/4 years. After Year 10, it is possible to pump some water from the sludge pond as additional consolidation takes place. In addition to water from rehandled sludge, the sludge pond also contains water from the hydraulic construction of the sludge pond dyke (tailings sand is used as dyke material). For a summary of the detailed mass balance calculations refer to Section 7.2 of this report.

The tailings pond and mine are recontoured, covered with prepared soil, and revegetated as outlined in Concept No. 2.

Added costs over Concepts No. 1 and 2 result from sludge transfer from both the out-of-pit pond and the in-pit ponds to the sludge pond, as backfilling with tailings sand proceeds. The benefits in this scheme are that the sludge is placed at, or near the natural ground surface level, the exterior pond is completely backfilled with tailings sands, (thus improving reclamation prospects and eliminating future drainage problems as is Concept No. 4), and all wet toxic materials are concentrated in one relatively small pond. An option to balance the hydraulic head in the sludge pond against the regional hydrostatic head of groundwater can be achieved by modifying quantities of backfill placed into the pit serving as the sludge pond. Another improvement over Concepts No. 1, 2, and 4 is the reduction of the volume of the north waste dump.

Comparison of Concepts at the Minimum Level of Reclamation

The five tailings disposal concepts for the Minimum Level of Reclamation presented in this chapter are summarized in Table 5.3.2-1 and illustrated in Figures 5.3.2-1 to 5. The quantitative comparison of the five concepts presented in Table 5.3.2-2 aids in the selection of an overall "best concept" at the Minimum Level of Reclamation.

A reduction of wet tailings surface area results in a reclamation plan that is more environmentally acceptable. The solid waste disposal areas can be reclaimed with the least effort, i.e., waste dumps may be topped with a certain amount of prepared soil and grassed or treed. Similar treatment, although considerably more costly, may be applied to sand dykes and beaches. However, no practical method exists to reclaim the wet portion of the tailings ponds. Therefore, at the Minimum Level of Reclamation, systems were described which have the smallest possible surface area of wet tailings, i.e., tailings water and sludge.

From Table 5.3.2-2 it can be seen that the total area disturbed by mining, bitumen processing, and tailings disposal is approximately the same for each of the five concepts. The surface area of wet tailings is minimized in Concept No. 5, while Concepts No. 1 and 2 have the greatest wet tailings surface area. A quantitative assessment of wet tailings surfaces for each concept is presented in Table 5.3.2-3. Environmentally speaking, over the long term it is preferable to have the

Table 5.3.2-1

TAILINGS DISPOSAL AND RECLAMATION FOR MINIMUM LEVEL CONCEPTS (120,000 BPCD)

SUMMARY TABLE

Minimum Level Concept No.	Main Features
1	Tailings into out-of-pit tailings pond and mined-out pits as mining progresses; tailings sand forms beaches; sludge is topped by tailings water; very large surface area of wet tailings; no reclamation; unacceptable. Figure 5.3.2-1.
2	Same tailings disposal scheme as Concept No. 1, except minimum reclamation undertaken; all areas of disturbance covered with 0.3 m (1.0 ft.) muskeg-overburden mixture; grass seeded; acceptable by present standards. Figure 5.3.2-2.
3	All sludge stored in out-of-pit tailings pond; pits back-filled with overburden, centre reject and tailings with water and sludge pumped out to out-of-pit tailings pond; large outside tailings pond, but surface area of wet tailings reduced; soil and revegetation as in Concept No. 2. Figure 5.3.2-3.
4	Out-of-pit tailings pond stores sludge and water only temporarily; as Pit 1 sand backfills, sludge is pumped to out-of-pit tailings pond; as Pit 2 sand backfills, sludge is pumped to Pit 3; at the end of mining the out-of-pit tailings pond is emptied into Pit 4; surface area of wet tailings is approximately the same as in Concept No. 3 but more stable storage space achieved below surrounding ground surface; soil and revegetation as in Concept No. 2. Figure 5.3.2-4.
5	All sludge concentrated at in-pit sludge pond; out-of-pit tailings pond and remaining pits filled with sand as sludge is pumped to sludge pond and water is recycled. Due to sludge separation and large depth of the sludge pond (Pit 1) this concept yields smallest surface area of wet tailings; soil and revegetation as in Concept No. 2; considered the most desirable of the five minimum level concepts. Figure 5.3.2-5.

Table 5.3.2-2

TAILINGS DISPOSAL AND RECLAMATION FOR MINIMUM LEVEL CONCEPTS (120,000 BPCD)

SURFACE AREA DISTURBANCE

	Minimum Level - Concept No.				
	1	2	3	4	5
Total Area Disturbed*	50.55	50.55	49.87	51.75	49.32
Area Disturbed Out-of-Pit*	25.33	25.33	24.65	26.53	24.10
Total Surface Area of Sludge/Water	27.65	27.65	10.42	12.40	7.84
a) In-Pit	20.53	20.55	0	12.40	7.84
b) Tailings Pond	7.10	7.10	10.42	0	0
Total Surface Area of Tailings Sand	7.52	7.52	21.06	26.02	22.98
a) In-Pit	2.62	2.62	14.68	12.82	10.02
b) Tailings Pond	4.90	4.90	6.38	13.20	12.96
Total Surface Area of Overburden and Centre Reject	8.65	8.65	11.07	8.65	8.85
a) In-Pit	0	0	6.79	0	1.85
b) Outside Piles	8.65	8.65	4.28	8.65	7.00
Surface Area of Oversize Reject in Outside Pile	1.68	1.68	0.57	1.68	1.14
Area of Empty Final Pit (Filled with Water)	2.05	2.05	3.75	0	5.51
Plant Site	3.00	3.00	3.00	3.00	3.00
	(All areas in $10^6 \times m^2$)				

* Excludes Out-of-Pit Conveyors, Roads, Muskeg Dumps, etc.
Surface Area = Horizontal Projection
Surface areas of final configurations shown.

Table 5.3.2-3

TAILINGS DISPOSAL AND RECLAMATION FOR MINIMUM LEVEL CONCEPTS (120,000 BPCD)

WET TAILINGS COMPARISON TABLE

	MINIMUM LEVEL - CONCEPT NO.				
	1	2	3	4	5
FINAL CONFIGURATION:					
Area of Wet Tailings ($10^6 \times m^2$)	27.65	27.65	10.42	12.40	7.84
Location of Wet Tailings					
Out-of-Pit Pond (above ground)	YES	YES	YES	NO	NO
In-Pit (below ground)	YES	YES	NO	YES	YES
Wet Tailings Surface Area as percentage of Total Area Disturbed	54.70%	54.70%	20.89%	23.96%	15.90%
DURING-MINE-LIFE CONFIGURATION:					
Area of Wet Tailings x Duration of the Disturbance (hectare - years)	33,406	33,406	29,879	26,040	20,041

semi-fluid portions of the tailings stored below the elevation of surrounding surface. Such an arrangement is more likely to remain stable indefinitely than sludge and water stored in an elevated tailings pond.

In areas where artesian conditions prevail, groundwater may flow into the semi-fluid portion of the pond. Such springs may result in the overflow of tailings water and sludge. In such cases this groundwater problem may be solved (if not completely, then at least partially) by controlled placement of overburden backfill, and the establishment of a final pond surface above the expected artesian groundwater heads. This is demonstrated by the slewing BWE mine plan discussed in Chapter 7. The solution of this dilemma requires a detailed assessment of field geology and hydrology as well as a risk analysis.

Oil sands developments not only have final impacts environmentally, but also a "dynamic" impact during operating years. Consequently, a new measure gauging the combined effects of area and time was adopted. This parameter equals the arithmetic sum of the products of wet tailings area of each pond (in hectares) multiplied by the existence period of these wet ponds during the mine life (in years). The parameter is tabulated in Table 5.3.2-3 under "During-Mine-Life" configuration in terms of hectare-years for each concept. It may be concluded from this table that the No. 5 Concept is overall the most desirable system of tailings disposal under the specified criteria. It has the smallest surface area of wet tailings, the wet tailings are located below the surrounding topography, and during-mine-life wet disturbance is minimum.

It should be remembered that the more favourable parameters of Concept No. 5 are the result of materials rehandling: namely, rehandling of sludge from an active tailings pond into one deep mined-out pit. The sludge rehandle also results in reduction of impounded tailings water. This is illustrated when comparing Concept No. 2, where each individual pond is covered with water, and Concept No. 5, where all the water is recycled and eventually only one pond (sludge pond) remains with a layer of water.

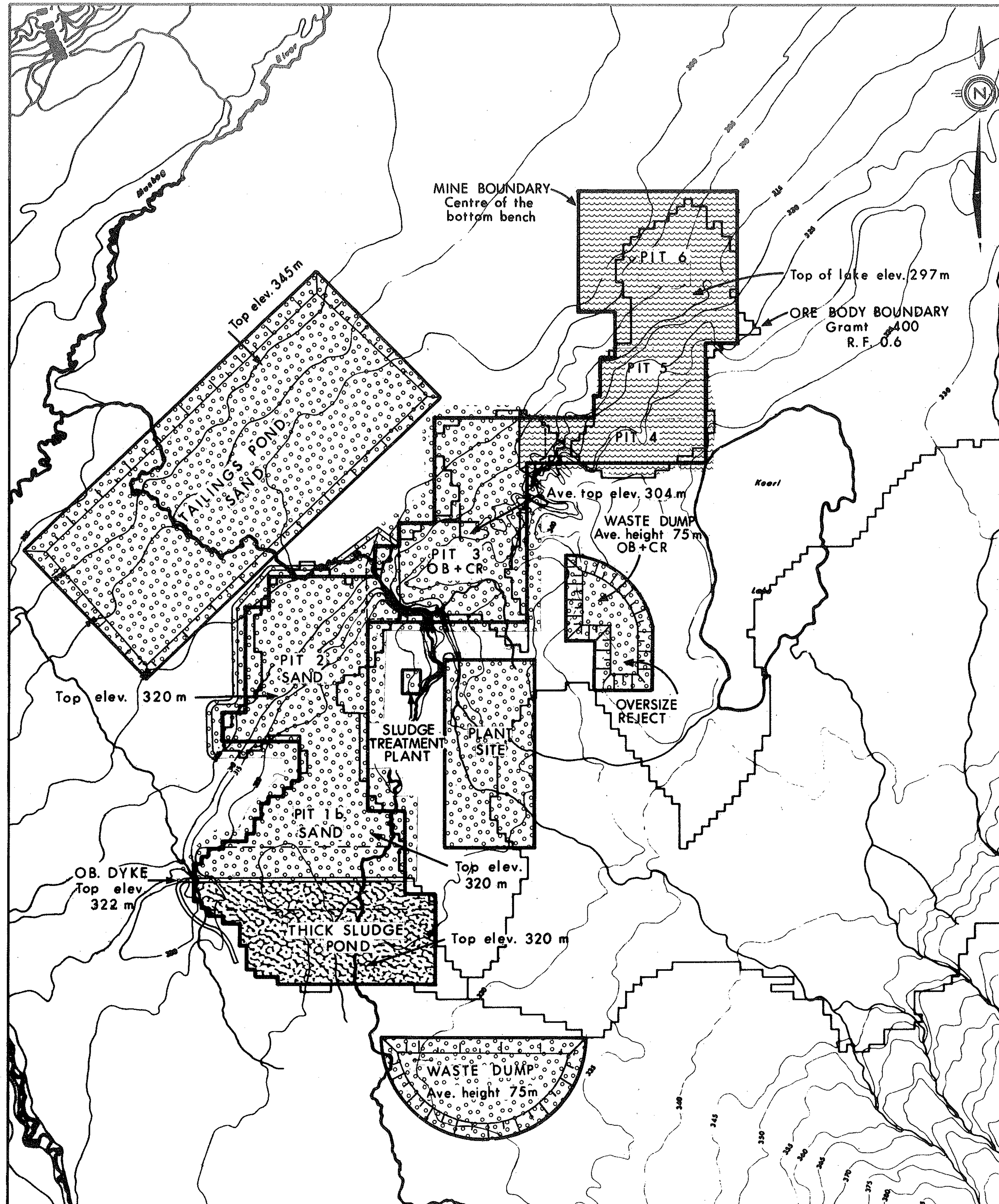
The tailings disposal schemes in Concepts No. 1 and No. 2 are identical, as both result in a series of shallow wet tailings ponds remaining at completion of mining. The available volumetric capacity to store wet tailings (sludge and water) in each pit is reduced by tailings sand, which is beached from the pit perimeter. Over half of the surface area disturbed is wet. The hectare-years index of wet tailings is the largest of all the schemes examined.

Concept No. 3 is superior to Nos. 1 and 2, but is not as desirable as No. 5. The concept uses the same idea of rehandling and concentration of sludge as Concept No. 5, but results in a larger wet tailings surface area and a more hazardous above-ground location. Concept No. 3 yields additional mined-out pit volume for overburden and reject backfill. However, it is usually more desirable to have environmentally hazardous wet tailings in the mined-out pits, and the overburden and reject in outside waste dumps.

Concept No. 4 is similar to No. 3 in rehandling of wet tailings, but in this concept, wet tailings are permanently impounded in the final pits, rather than above surface. Some double rehandling of sludge is required, however. During-mine-life wet tailings disturbance is greater than in No. 5, but less than in the other concepts. The environmental significance of the number of hectare-years of wet tailings should not be underestimated. An active tailings pond with warm tailings discharge is one of the first major water bodies in the area to be ice-free in spring; this feature may prove hazardously attractive to large numbers of migrating waterfowl.

5.3.3 TAILINGS DISPOSAL AT THE IMPROVED LEVEL OF RECLAMATION

The Improved Level of Reclamation differs from the Minimum Level in a number of ways (see Table 5.3.2-1), the most important of which is that a "sludge treating" process is utilized to lower the water and bitumen content of tailings sludge, thus reducing the wet-pond size further, and creating a potential for resurfacing and reclamation. In comparison to the Minimum Level, substantially greater amounts of prepared soil are



NOTES

- 14 YEARS OF TAILINGS SAND TO TAILINGS POND (2400 m x 5400 m).
- BALANCE OF TAILINGS SAND TO PITS 1b AND 2.
- OVERBURDEN AND UPPER BENCH CENTRE REJECT BACKFILL AFTER 13¼ YEARS.
- SLUDGE PUMPED FROM TAILINGS POND AND PITS TO SLUDGE TREATING PLANT.
- TREATED SLUDGE (50% WATER AND MOST BITUMEN REMOVED) PIPED TO THICK SLUDGE POND.
- SURFACE AREA OF THICK SLUDGE POND IS HALF OF SLUDGE POND IN MIN. LEVEL - CONCEPT 5.
- POSSIBILITY OF THICK SLUDGE POND SURFACE SOLIDIFICATION TO SUPPORT SOIL AND PLANT GROWTH, LEAVING NO WET TAILINGS.
- RESIDUAL WATER (INCLUDING PLANT BLOW DOWN WATER) INJECTED INTO SAND BACKFILLED PITS.
- MUSKEG FOR SOIL MANUFACTURE OBTAINED FROM WITHIN THE MINE.
- GENERALLY 1 m OF BLENDED MUSKEG AND OVERBURDEN, DEPTH INCREASED OR DECREASED AS REQUIRED BY CHARACTERISTICS OF UNDERLYING MATERIALS.
- SHRUBS AND TREES PLANTED.

LEGEND

THICK SLUDGE*

1 m Soil — TREATED DYKES, BEACH, OVERBURDEN AND OVERSIZE REJECT PILES, PLANT SITE

EMPTY END PIT (FILLS WITH WATER)

* POSSIBILITY OF SURFACING AND REVEGETATING

METERS
1000 500 0 1000 2000 3000

FIGURE 5.3.3-1

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT

PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH

TITLE: 120,000 BPCD TAILINGS DISPOSAL AND RECLAMATION IMPROVED LEVEL

SCALE: AS SHOWN	DATE: AUG., 1978	DRAWN BY: TM ☒ RC ☐ S.R.	TM DRAWING No.
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replaced. Although the total area disturbed is not reduced at this level, it may be possible for all disturbances to be reclaimed if a satisfactory technology can be developed to resurface the remaining (thicker) sludge pond.

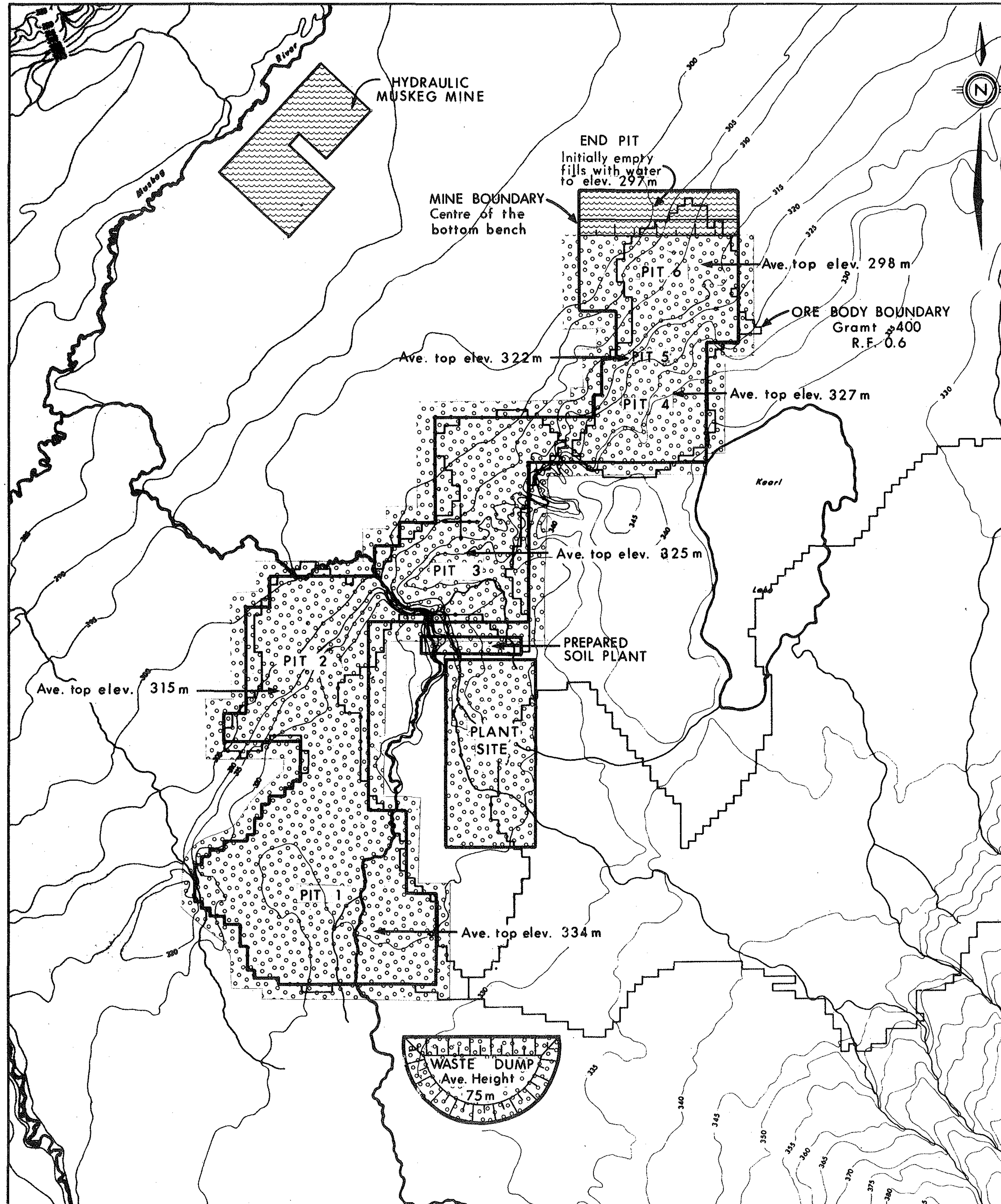
In this scheme, the out-of-pit tailings pond is the same size as that used in the Minimum Level - Concept No. 5 (see Figure 5.3.3-1); that is, it receives tailings for 13.8 years, by which time the pond is full of tailings sand. The difference is that the sludge that is being pumped out is treated in the sludge treatment plant, where some water and most of the remaining bitumen are removed. To avoid unnecessary overdesign of the sludge treatment plant, sludge intake should be kept constant. The use of a sludge treatment process reduces the amount of makeup water required by the extraction plant, and at the cessation of mining a comparatively small volume of fluid waste product remains. This remaining water might be disposed of by injecting it into the sand-filled pits.

As in the previous concepts, an out-of-pit tailings pond is constructed. A 24,000,000 m^3 starter dyke is constructed out of overburden during the preproduction period. Its outside dimensions are 2,400 m x 5,400 m. During summer, tailings are pumped to the tailings pond, where they are hydraulically placed and compacted to form the dykes. During winter, sands are deposited on the inside of the dykes to form a protective beach. For a detailed material balance on tailings pond dyke construction, see subsection 7.2.2. After 5 years, the mining has progressed far enough that the bottom bench has cleared the area where the first in-pit dyke must be constructed. This overburden dyke stores the entire 25 years' production of thickened (treated) sludge.

Between Years 5.0 and 13.8 the sludge treatment plant operates at the rate of 24.0 x 10^6 m^3 of wet sludge per year. During this period sludge is removed from the out-of-pit tailings pond and treated, while the pond is backfilled with tailings sand. When the out-of-pit tailings pond is full of sand (Year 13.8), the tailings disposal is diverted to the mined-out pits (Pit 1b and 2). By Year 16.0, a sufficient layer of sludge develops in the active tailings pond (Pit 1b and 2 at that time), to allow the sludge treatment operation to start again, but this time a lower rate of 21.3×10^6 m^3 of wet sludge per year is required to handle all sludge from in-pit tailings disposal. The sludge treatment plant must be designed for the higher rate (i.e., 24.0×10^6 m^3 wet sludge per year) to avoid the necessity of double-handling wet sludge and the construction of a temporary in-pit dyke Thus, while the increase in the sludge plant capacity is 17.6%, the benefits are that no overburden/tailings sand dyke is required between Pit 1b and Pit 2, and no double pumping of a total of 102.0×10^6 m^3 of wet sludge between the consecutive pits is required.

The out-of-pit tailings pond is backfilled with tailings sand as in the Minimum Level - Concept No. 5, but the reduction of the sludge pond size at this level of reclamation permits the remainder of the tailings sand to be stored in Pits 1b and 2, with the top-of-sand elevation in Pit 2 having to be raised from 290 m to 320 m. As a result, Pit 3 is available for backfill with overburden, upper bench centre reject, and oversize reject, reducing the size of the out-of-pit waste dump (waste dump next to the plant site) considerably. The slightly larger lake formed by the end pit (Pit 5) is the result of scheduling of mass movement rather than the intent of the design. For more details on mining and tailings disposal for this mine, see Subsection 7.2.2. and 7.3.2.

With regard to soil replacement and revegetation, Techman/RC maintain that adequate soil material must be made available if a permanent, self-sustaining vegetative cover is to be established. Although the precise thickness of this new rooting zone depends on a number of site-specific criteria (slope, infiltration, nature of prepared soil, type of vegetation to be established, etc.), 1 m of prepared soil is considered to be the average application depth. On level or gently sloping areas, the prepared soil is transported with trucks and placed by end dumping. On steeply sloping surfaces, additional dozer spreading is required. The prepared soil consists of a 1:1 mixture of muskeg and a high clay content surficial material that has been mechanically mixed prior to placement. The Improved Level of Reclamation requires considerably larger



NOTES

- THIS LEVEL OF RECLAMATION APPLICABLE ONLY TO DRY PROCESS BITUMEN EXTRACTION PLANT
- MINED-OUT PITS ARE BACKFILLED WITH SELECTIVELY PLACED OVERBURDEN, TAILINGS SAND, OVERSIZE REJECT AND CENTRE REJECT.
- WASTE DUMP CONTAINS PREPRODUCTION OVERBURDEN IN ADDITION TO FIRST 1¼ YEARS OF TAILINGS SAND, OVERBURDEN, OVERSIZE REJECT AND CENTRE REJECT.
- BACKFILLED PITS FINISHED TO BLEND INTO THE GENERAL TOPOGRAPHY OF THE AREA.
- MUSKEG FOR SOIL MANUFACTURE OBTAINED FROM HYDRAULIC MUSKEG MINE.
- GENERALLY 1 m OF BLENDED MUSKEG AND OVERBURDEN, DEPTH INCREASED OR DECREASED AS REQUIRED BY CHARACTERISTICS OF UNDERLYING MATERIALS.
- AREA SELECTIVELY PLANTED TO FORESTS AND MEADOWS FOR WILDLIFE HABITAT.

LEGEND

1m Soil — TREATED PILES, BACKFILLED PITS, WASTE DUMP AND PLANT SITE

EMPTY END PIT (FILLS WITH WATER)

METERS

1000 500 0 1000 2000 3000

FIGURE 5.3.4-1

CLIENT: ALBERTA DEPARTMENT OF THE ENVIRONMENT			
PREPARED BY: TECHMAN LTD. and RHEINBRAUN - Consulting GmbH			
TITLE: 120,000 BPCD TAILINGS DISPOSAL AND RECLAMATION ENHANCED LEVEL			
SCALE: AS SHOWN	DATE: AUG., 1978	DRAWN BY: TM ☒ RC ☐ Sm.	TM DRAWING No.
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quantities of prepared soil for resurfacing the disturbances (tailings pond, backfilled mine, waste dumps, plant site, etc.) to allow revegetation. To provide a layer of prepared soil of 1.0 m average thickness requires some 16,200,000 m^3 of muskeg. Details respecting the manufacture of prepared soil at this level of reclamation are provided in Section 5.4 and Subsection 7.2.2.

The benefits of this system as compared to Concept No. 5 of the Minimum Level include reduction of the sludge pond surface area by one-half, and an increased viscosity of thickened sludge. The thickened sludge may lend itself to further (or even complete) reclamation by the possible solidification of a top layer able to support soil and plant growth.

Costs incurred in this plan in excess of those encountered at the Minimum Level of Reclamation arise mostly from the operation of the sludge treatment facility. This cost is offset by the value of the recovered bitumen. No estimates were prepared for the additional capital and operating costs required for a bitumen recovery unit in the detailed plans in Chapter 7.0 and 8.0 utilizing the Improved Level sludge treatment technique.

5.3.4 TAILINGS DISPOSAL AT THE ENHANCED LEVEL OF RECLAMATION

At the Enhanced Level of Reclamation, a dry tailings system is utilized. Since no practical technology applicable on a large scale exists to date, certain assumptions as to tailings characteristics and environmental effects must be made. As discussed in Section 5.2.2, Dry Processes, a high temperature bitumen extraction process is assumed, yielding a non-toxic, dry-sand product. An environmental concern (and a problem not necessarily difficult to manage) is the stabilization and revegetation of tailings sands before wind erosion (duning) and water erosion become chronic conditions.

As soon as sufficient space is developed in the initial pit (about 1.5 years in parallel mining), benches can be established, from which continuous selective backfilling with tailings sands, overburden, centre reject (from the upper bench), and oversize reject (from the plant) may begin. Preproduction overburden stripping, plus 1.5 years of production overburden stripping, and tailings sands, including oversize and centre reject, must be stockpiled south of Pit 1. Selective placement of material allows the construction of a stable pile. Sufficient mixing of sand and overburden on the outside of the pile minimizes wind and water erosion prior to the placement of a prepared soil.

The final landscape is comprised of an out-of-pit waste dump and back-filled pits, which although somewhat elevated are nonetheless compatible with the general topography of the area. At the north end of the mine, a small final empty pit remains. To avoid costly rehandling of backfill material, it is considered acceptable to let this pit become a small fresh water lake (much smaller than the lake described in the concepts of the Minimum and Improved Levels of Reclamation).

Centre reject from the lower bench draglines is backcast onto the pit floor exactly the same way as in the previous examples except, at this level, no consideration must be given to in-pit dykes, as semi-fluid tailings are not produced by the extraction process. When only bucket wheel excavators are used, the centre reject from this lower bench is transported by conveyor to the dump area, and is incorporated into the tailings sands.

The area disturbed at this level is the smallest of all the schemes considered. Prepared soil placement is as outlined at the Improved Level, although the preparation of the manufactured soil is different (see Section 5.4).

5.4 RECLAMATION TECHNIQUES

At the outset of this project, the Consultants realized that novel methods of oil sands reclamation needed to be introduced to this study. However, Techman/RC also were aware that reasonable cost effectiveness of any plans or ideas was of ultimate importance. Therefore, during the course of developing mine plans and selecting equipment and materials handling techniques, the Consultants selected the options that seemed to optimize the greatest number of technical and economic factors. It is the expectation of the Consultants that the type of reclamation described for the Improved and Enhanced Levels, if implemented, would result in a standard of reclamation practices approximating those routinely accomplished in western Europe, though it is recognized that many physical, climatic, and operational differences exist.

5.4.1 MUSKEG REMOVAL ALTERNATIVES

Much of the area to be mined during the life of an oil sands mine is covered by muskeg ranging in depth from a very shallow veneer to possibly 10 m at the extreme. This material must be removed before overburden can be excavated. A number of prospective muskeg removal systems were examined. The first system (the only one proven to date) utilizes a conventional front-end loader and truck operation in which all materials are removed during the winter months. A long face is opened, and the operation takes advantage of freezing conditions to overcome difficulties respecting equipment mobility, muskeg drainage, and material selection and separation. The disadvantage of this system is that the end product contains very large blocks of muskeg which, when stored, take up considerable surface area and present difficulties in rehandling as a result of differential thawing. Unless technologically advanced methods of rehandling the frozen muskeg can be found, this material will be difficult to incorporate into the reclamation scheme at all levels of reclamation.

Alternative systems of muskeg removal that maximize the special capabilities of bucket wheel excavators and belt conveyors were also considered. Two concepts were visualized for muskeg removal with bucket wheel excavators. In both cases, the objectives were to use the bucket wheel excavators to selectively dig muskeg, mechanically incorporate it with overburden materials suitable for reclamation during the removal process, and then transport the mixture, via belt conveyors, to or close to the active reclamation sites.

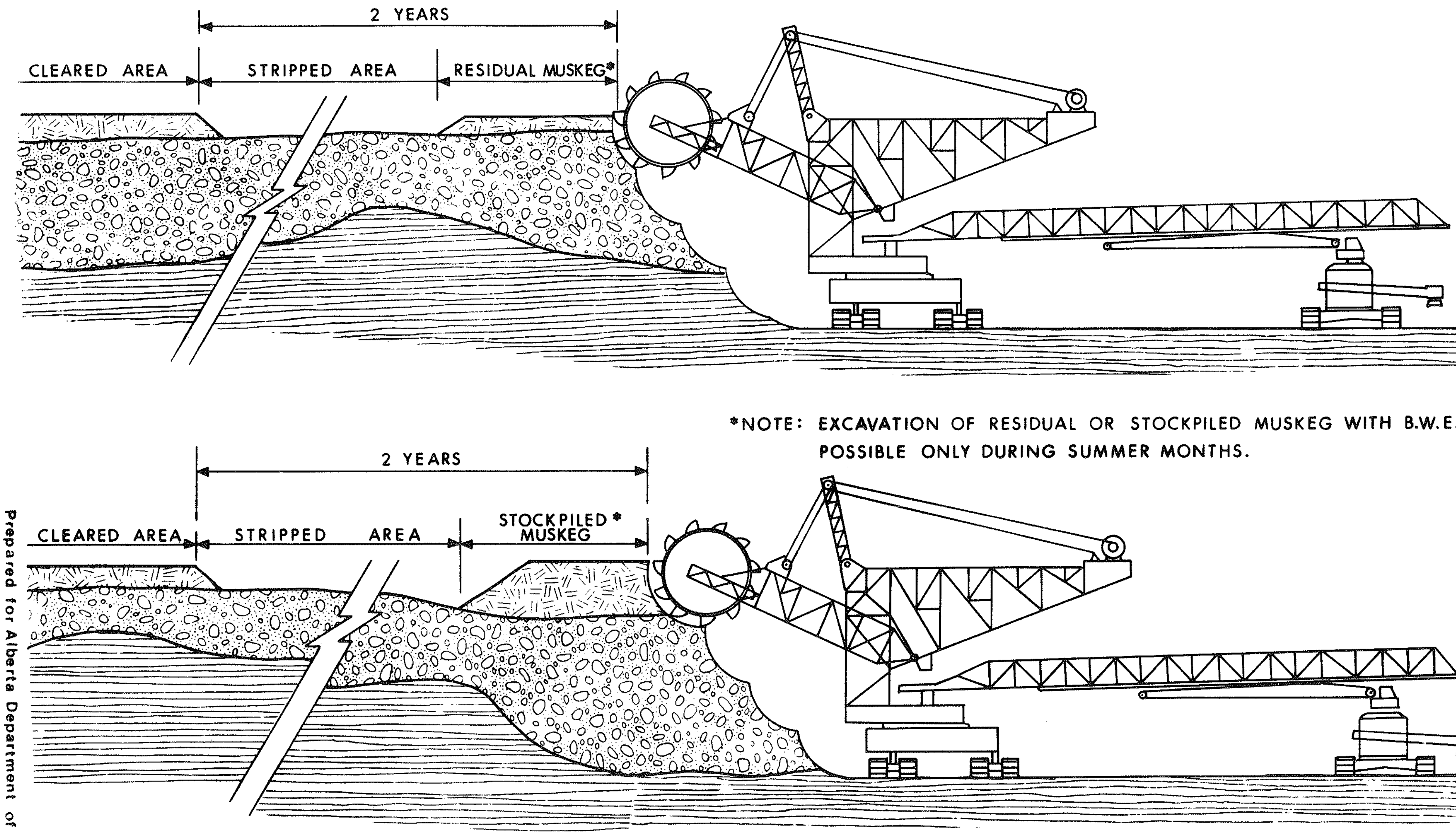
In the first concept (Figure 5.4.1-1), the uppermost layer of overburden is removed, together with the thin overlying layer of muskeg. This mixed material is stacked on top of spoil by the spreader, and later removed to intermediate stockpile sites or to the final reclamation sites using front-end loaders and trucks. This scheme has a number of potential drawbacks including:

- Somewhat oversized excavating equipment resulting from increased volume of production and lower operational efficiency of the excavator;

- A requirement for substantially improved (and more expensive) drainage of all muskeg to be removed by bucket wheel excavators, in order to reduce the risk of pit slope failures during excavation, and to minimize operating difficulties (mainly freezing and mud build-up) on excavators and belt conveyors;

- Substantial problems of scheduling that allow summer muskeg salvage only, and force acceptance of all underlying overburden for reclamation;

- Potential blockage of the bucket wheel excavator and conveyor transfer points by tree trunks, branches, and roots, unless a thorough prehandling procedure removes these obstacles.



SELECTIVE MUSKEG / OVERBURDEN REMOVAL WITH BUCKET WHEEL EXCAVATOR

FIGURE 5.4.1-1

RHEINBRAUN-Consulting GmbH
TECHMAN LTD.
Prepared for Alberta Department of The Environment

In the second bucket wheel excavator concept, also illustrated in Figure 5.4.1-1, the muskeg is prehandled in winter and piled in appropriate locations for later removal in summer. This approach to concentrating muskeg in strategically-located surface hollows or piles (if overburden is shallow) relieves some of the scheduling problems of the bucket wheel excavator and the belt conveyors, thus making the operation more manageable. A disadvantage of this concept is that the muskeg and overburden cannot be mixed as effectively at the digging face because of incomplete thawing. However, once loaded on conveyors this is not a serious drawback since further rehandling, spreading, and mixing by mobile equipment occurs. A minor disadvantage is that a selective prestripping operation to stockpile muskeg is required. Since muskeg is removed in any case, this does not add substantially to the gross costs of the prestripping operation.

Although several of the above-mentioned drawbacks are serious, the Consultants believe (based on knowledge and experience) that it is technically possible to overcome the difficulties of selectively removing and blending muskeg with overburden for all the levels of reclamation. Soil manufacturing alternatives for the reclamation requirements at the Minimum, Improved, and Enhanced Levels are described in later subsections.

5.4.2 GENERAL REMARKS REGARDING MUSKEG AND OVERBURDEN STORAGE

Although muskeg storage for later use as reclamation material is practiced to some degree at existing oil sands operations, observations and evaluations by Techman/RC identified a number of major problems:

- Frozen muskeg, when stockpiled to greater depths than the average depth of frost penetration (0.6 to 1.0 m) remains permanently frozen or at least frozen for many years. Storage at lesser depths requires substantial acreage. Mechanical breaking of frozen blocks is extremely expensive, if not effectively impossible.

- Muskeg not stored in a frozen state quickly loses its moisture and becomes powdery in texture, actually repelling water rather than retaining it; this condition is considered permanent. The Rheinische Braunkohlenwerke AG experienced similar difficulties with peaty soils in their Rhineland mines some years ago. After a period of storage, the moistening properties of the peat could not be restored, primarily because of the waxy layer on the dead plants of which it is comprised.

- Changes that may also take place in terms of microbiology, nutrient content, etc. are unknown, but this kind of degradation probably results in reduced organic content and some loss of nutrients.

As a result of these observations, Techman/RC have set the following as general suggested objectives:

- Every effort must be made to use a "direct transfer" system to bring newly-removed muskeg directly to active reclamation sites.

- Since direct transfer is not always feasible, storage facilities for muskeg must be strictly controlled to minimize biological degradation.

- If at all possible, muskeg should be mechanically shredded prior to deposition at reclamation sites, in order to improve both handling (uniform mixing) and growth characteristics. This may be accomplished either in a separate operation, or as a result of the operating characteristics of the reloading machine.

- Wherever possible, muskeg should be mechanically mixed with overburden materials prior to deposition. Mixing with overburden prior to storage is also recommended so that physical changes which take place during storage do not affect mixing ability.

- Overburden materials considered suitable for reclamation should be stored separately from wasted overburden, or stored in zoned overburden dumps when it is not possible to mix them with muskeg at the time of removal.

5.4.3 MUSKEG AND OVERBURDEN HANDLING FOR RECLAMATION

The Consultants determined that a muskeg/overburden removal and mixing operation completely integrated with the general overburden removal plan can be justifiably assumed to be feasible for all levels of reclamation. Nevertheless, systems must be designed to ensure that large amounts of prepared soil can be processed. Adequate technology is already available for developing a mixed tailings sands/overburden/muskeg layer to meet the requirement of 0.6 m soil depth at the Minimum Level of Reclamation. Therefore, Techman/RC additionally identified the major objectives in constructing a prepared soil at the Improved and Enhanced Levels of Reclamation.

The objectives are:

- to remove muskeg from the pits economically, possibly in winter when handling and drainage problems are minimal;

- to utilize a system that generates thawed muskeg amenable to both breaking and spreading;

- to utilize a system that makes it possible to thoroughly mix muskeg with other overburden materials, preferably prior to application on reclamation sites;

- to minimize or avoid scheduling problems that affect either the mining operation or the reclamation operation;

- to minimize capital equipment expenditures and operating costs; and

- to provide a "prepared soil" with superior characteristics as a growth medium.

The mixture of muskeg and overburden that is to serve as the soil for reclamation purposes has been identified by the term "prepared soil". At all three levels of reclamation a prepared soil is manufactured, but the method employed varies considerably. The depth of material varies from 0.6 m added at the Minimum Level, to 1.0 m at the Improved and Enhanced Levels of Reclamation (consult Chapter 4.0 "Definition of Levels of Reclamation"). The details of the manufacture of prepared soil are discussed in the following sections. A schematic summary of the component activities during the course of manufacture of prepared soil is given in Figures 5.4.3-1, -2 and -3, "Prepared Soil Manufacture at the Minimum, Improved and Enhanced Levels of Reclamation," respectively.

Prepared Soil Manufacture at the Minimum Level of Reclamation

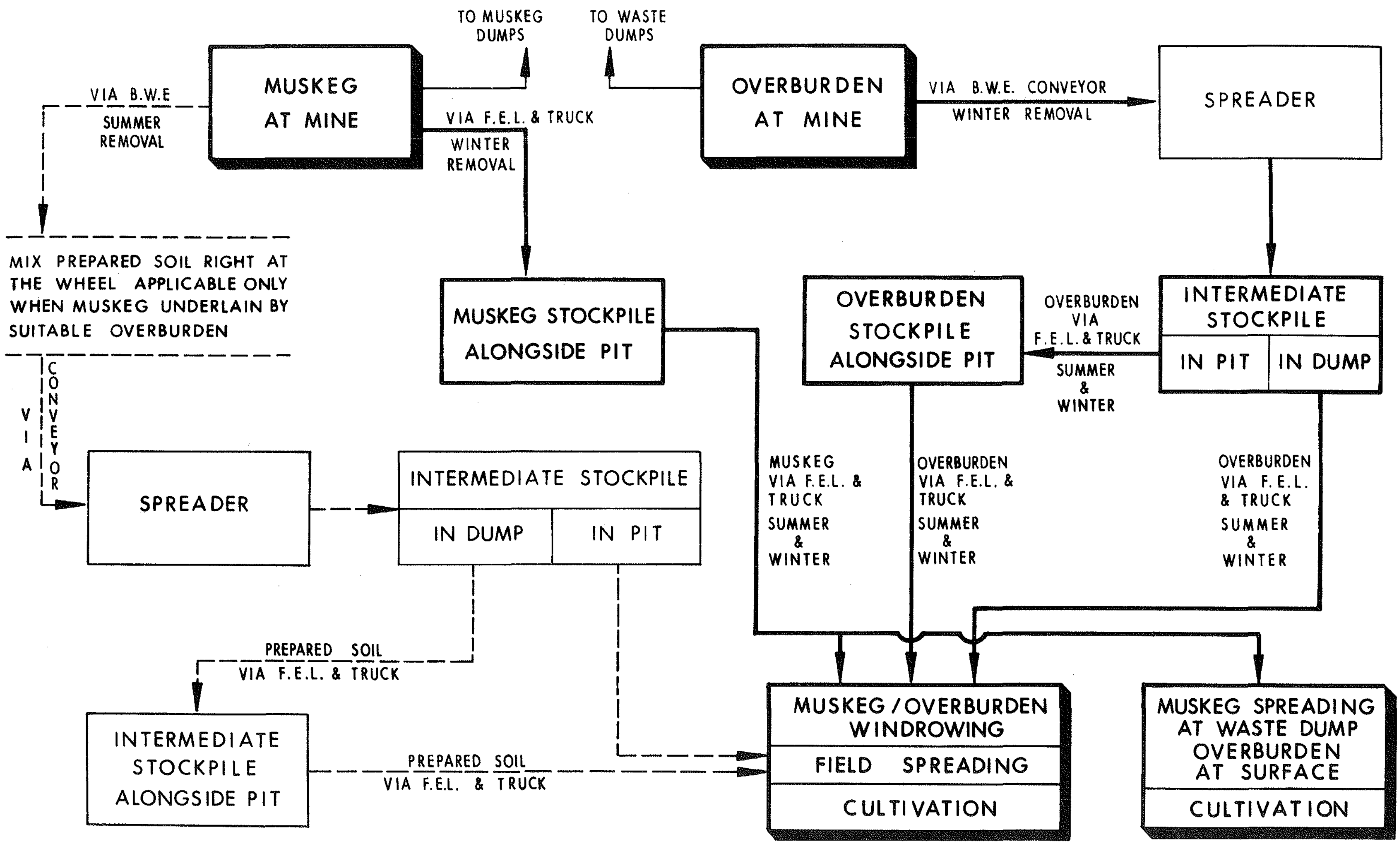
The basic concept to be instituted at the Minimum Level of Reclamation requires that suitable muskeg and overburden be excavated, transported, and placed separately. All blending of the three components (0.2 m muskeg, 0.2 m overburden, and 0.2 m tailings sand) is done in the field. Cultivation is to a depth of 0.6 m. Figure 5.4.3-1, "Prepared Soil Manufacture at the Minimum Level of Reclamation", shows some of the possible routes whereby the objectives of the Minimum Level of Reclamation can be achieved.

In the suggested prepared soil manufacturing scheme, overburden is selectively excavated by the BWE and transported by conveyor to the spreader. The spreader is simultaneously manoeuvered into position to place this overburden into an isolated intermediate stockpile. Depending on the plan and schedule of activities, these intermediate stockpiles may be in the pit or in the dump. If in the pit, the material must be removed to a stockpile outside and alongside the pit.

All muskeg is normally removed from the pit areas by a prestripping operation carried out during the winter, using front-end loaders and trucks. Muskeg is either placed permanently into muskeg waste dumps, or temporarily into muskeg stockpiles alongside the pit.

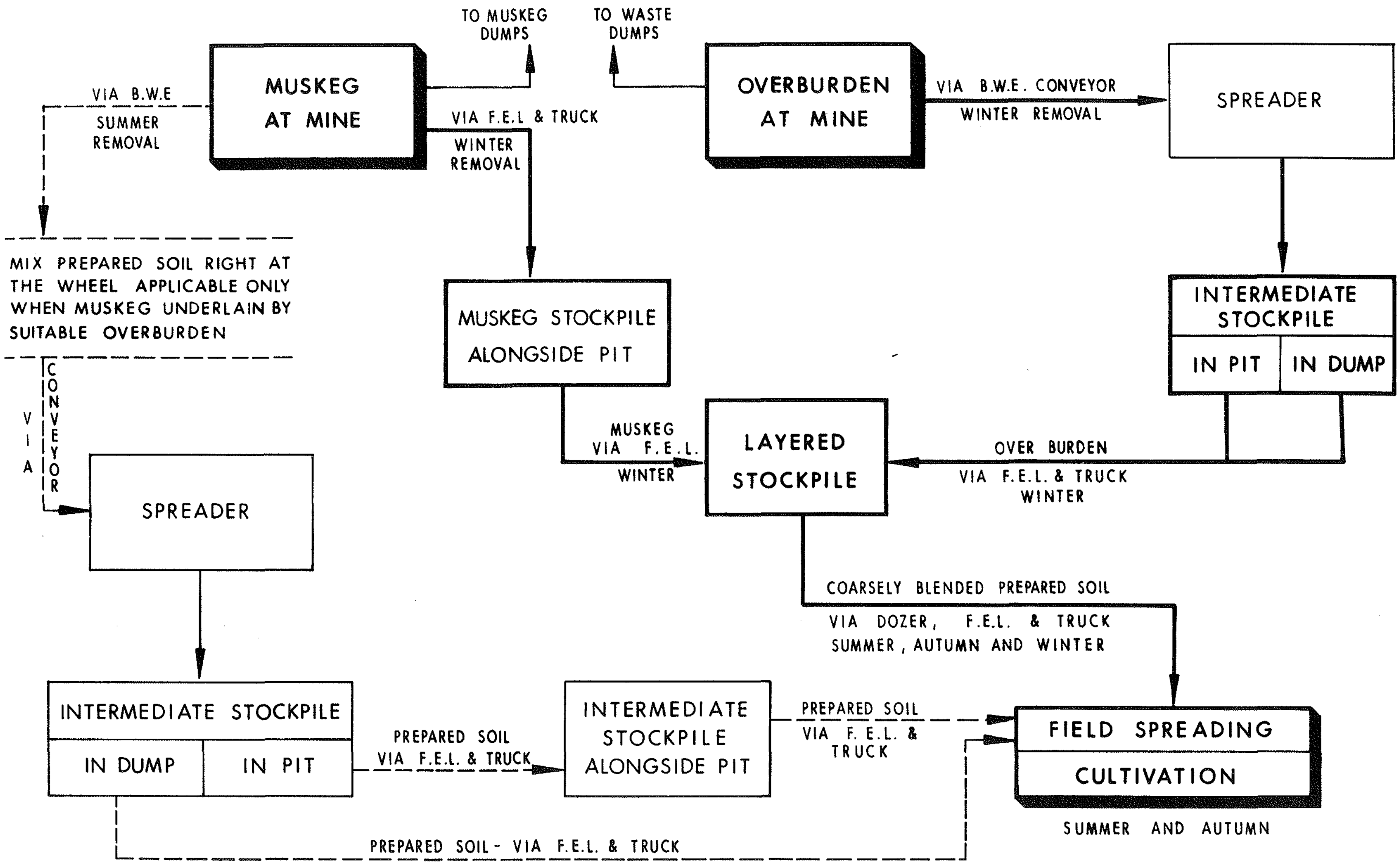
PREPARED SOIL MANUFACTURE AT THE MINIMUM LEVEL OF RECLAMATION

FIGURE 5.4.3-1



PREPARED SOIL MANUFACTURE AT THE IMPROVED LEVEL OF RECLAMATION

FIGURE 5.4.3-2



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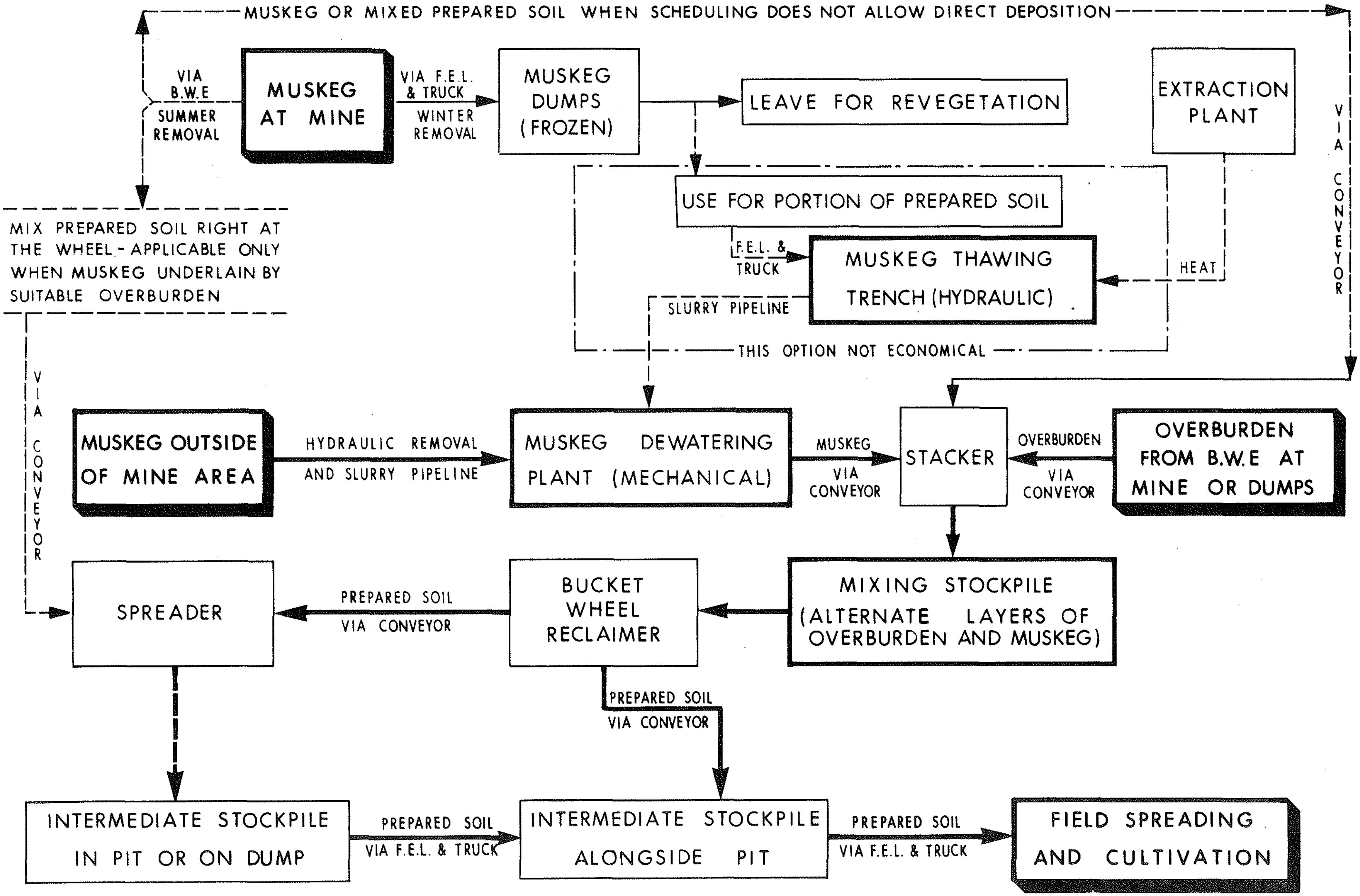
PREPARED SOIL MANUFACTURE AT THE ENHANCED LEVEL OF RECLAMATION

FIGURE 5.4.3–3



As required by reclamation schedules, materials are removed from the muskeg and overburden stockpiles by front-end loaders, and transported to the reclamation sites by off-highway trucks. Since the removed muskeg is largely semi-frozen, it is placed in windrows on the reclamation sites. The intent is that if muskeg is dumped in shallow piles not exceeding the average depth of frost penetration in muskeg (0.6 to 1.0 m), the piles will thaw and be ready for breaking and spreading later in the summer. During the summer or early fall, the thawed muskeg is spread with a dozer and graders to an average depth of 0.2 m. Subsequently, selected overburden is dumped on top, and is also spread to a depth of 0.2 m. The layers are then rototilled or plowed to a depth of 0.6 m and seeded. Figure 5.4.3-4, "Reclamation on Flat Surfaces - Minimum Level", illustrates this procedure. The major disadvantage of this handling scheme is that muskeg dries out and uniform mixing becomes difficult to achieve.

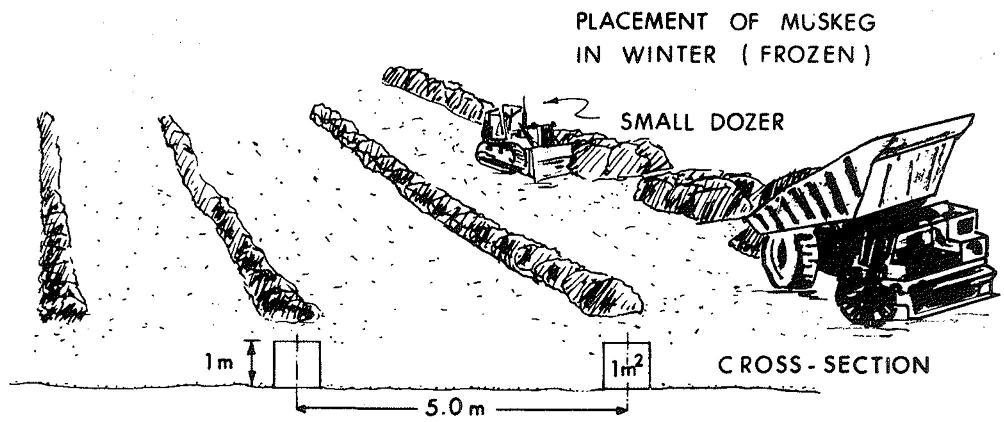
At the Minimum Level as well as at the Improved, portions of the sanded-in ponds may remain too wet for the construction of roads and spreading of prepared soil. This is especially true for the last areas to be sanded-in. It may take a number of years to drain and dry these localized, highly saturagted areas sufficiently.

On slopes (dykes, overburden and reject dumps) the system is somewhat modified as shown in Figure 5.4.3-5, "Soil Placement and Seeding on Slopes - Minimum Level". Materials are placed at the edge of the roadway, instead of in windrows, for thawing and later spreading.

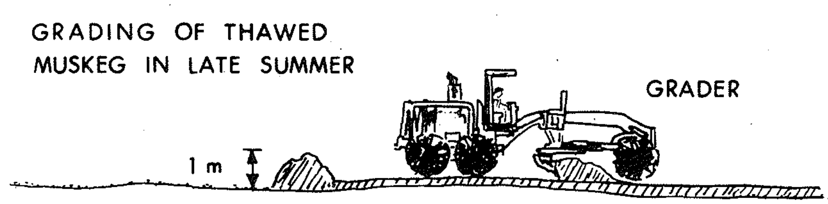
Again, it must be emphasized that direct mixing at the mining face with the bucket wheel excavator, if possible, is preferred over the above spreading technique. Blended overburden and muskeg are deposited at convenient locations alongside the overburden conveyor by the spreader. This material is either moved onto the reclamation site, or into intermediate storage piles for later transport to the reclamation sites.

The BWE must be scheduled so that it is in position to move during the summer and fall. The likelihood of this occurring is unknown, since suitability and the local mineability cannot be predicted without detailed field data. Consequently, this option, if possible at all, can only be incorporated into detailed operational plans.

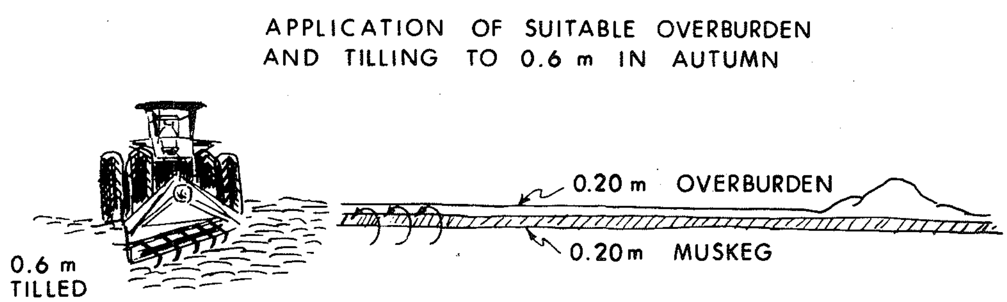
STEP 1.



STEP 2.



STEP 3.

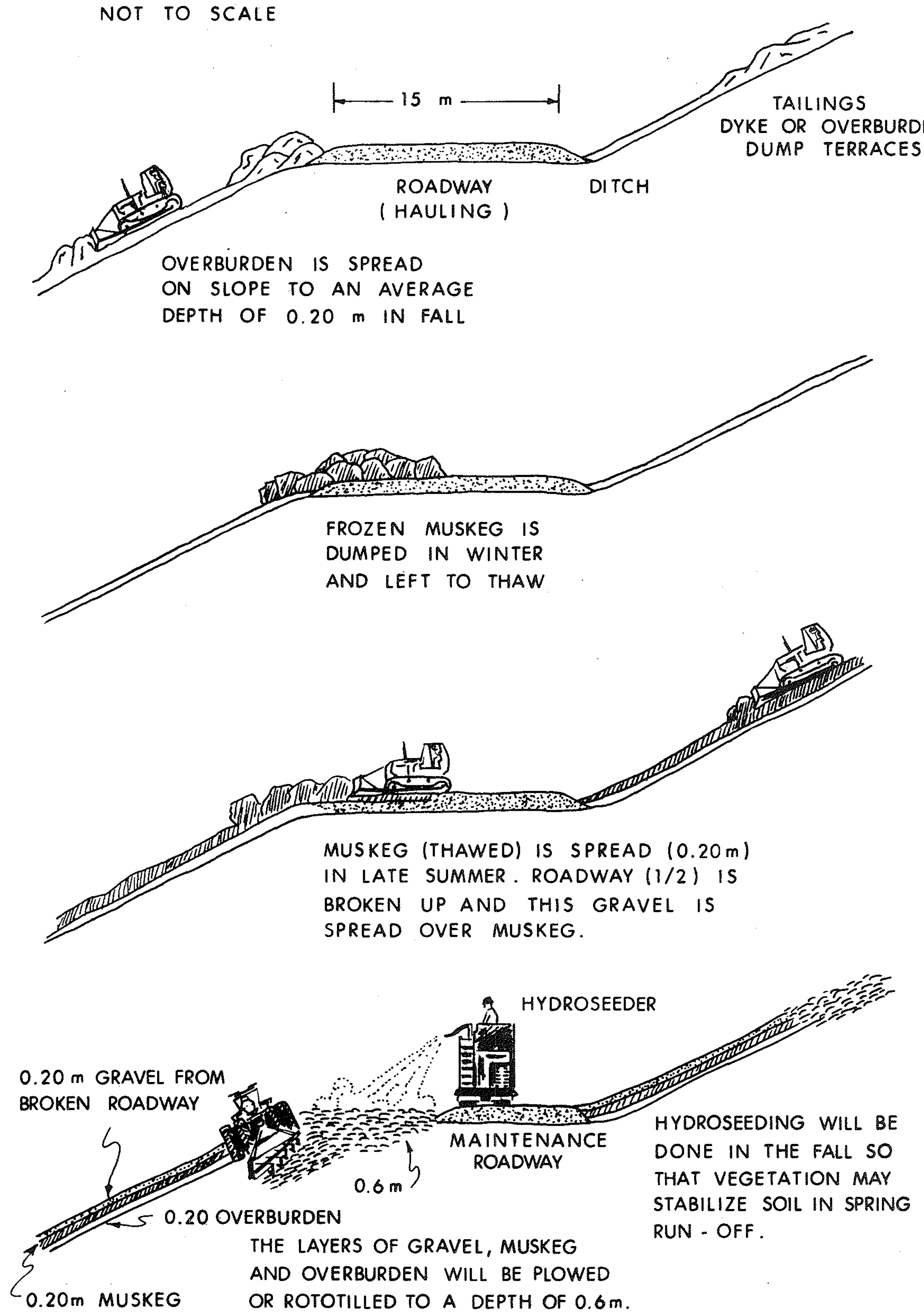


NOT TO SCALE

RECLAMATION ON FLAT SURFACES - MINIMUM LEVEL

FIGURE 5.4.3.-4

18-8-78



SOIL PLACEMENT AND SEEDING ON SLOPES - MINIMUM LEVEL

FIGURE 5.4.3.-5

21-8-78

Prepared Soil Manufacture at the Improved Level of Reclamation

At the Improved Level of Reclamation a total thickness of 1.0 m of prepared soil, consisting of an average of 0.33 m muskeg and 0.66 m overburden, is placed. Layered blending stockpiles alongside the pit allow an initial mixing of muskeg and overburden to occur. Additional mixing occurs during the field spreading operation. Details of the component operation are shown schematically in Figure 5.4.3-2, "Prepared Soil Manufacture at the Improved Level of Reclamation".

Selective overburden and muskeg salvage occurs as at the Minimum Level of Reclamation. However, instead of stockpiling the products separately, layered stockpiles are created alongside the pit. Layering is in the ratio of 0.33 m of muskeg to 0.66 m of overburden. This would be primarily a winter operation, although overburden could be added to the pile during the summer months. Materials are removed from these stockpiles as required by the reclamation schedule. Very large dozers (Caterpillar D 10-class, for example) will doze material down a sloped working surface designed to pass alternately across layers of overburden and muskeg. Soft areas will require a smaller, light weight dozer. Loading is done by large front-end loaders and hauling by off-highway trucks (see Figure 5.4.3-6 "Prepared Soil From Layered Blend Pile at the Improved Level of Reclamation"). Spreading in the field is done by smaller dozers (D 7 or D 8-class). Finally, the prepared soil is cultivated by deep plowing to a predetermined depth, thus providing additional blending. The depth is dependent on the uniformity of the 1 m layer after placement, the characteristics of the underlying materials, and the tree, brush, or grass species to be planted.

It may be possible to vary the prepared soil texture locally by spreading thinner or thicker, and incorporating various amounts of underlying tailings sand. To reduce hauling costs, the mixture transported to waste dumps may be richer in muskeg and subsequently diluted to the 1:2 ratio of muskeg to overburden by blending-in underlying overburden. Although hauling costs may be reduced, it may still be more economical overall to haul blended materials, since the blending of the underlying overburden by tillage requires considerable cultivation effort. This is in contrast to blending-in the tailings sand, which, due to its loose granular texture, mixes more readily. Flexibility in adjusting soil mixtures and depth should be used in order to achieve a desired soil permeability and water retention capacity, and thus accommodate a broader range of plant species.

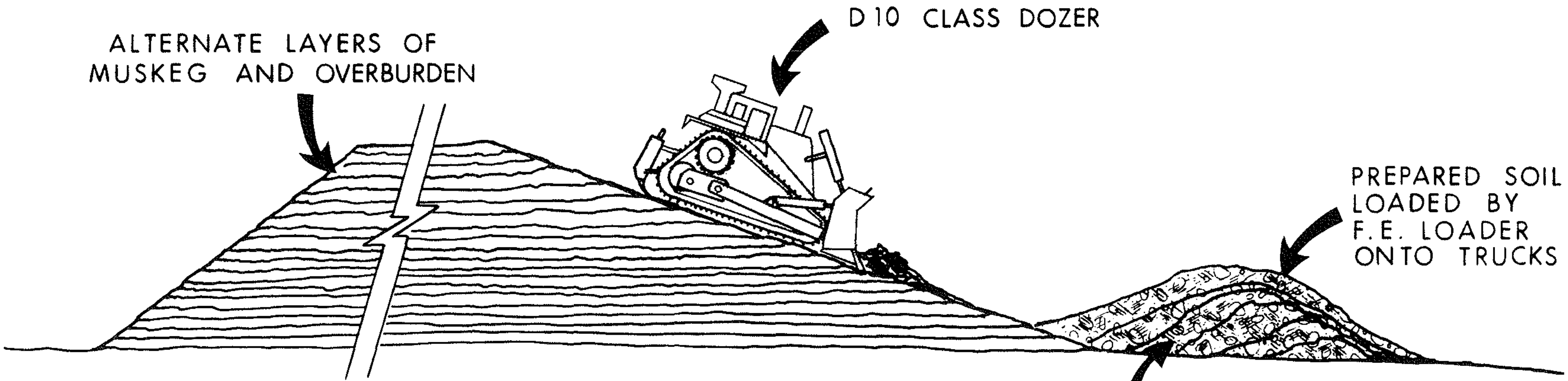
The methods of spreading are similar for both flat and sloping surfaces. After the material is dumped from the trucks, dozers push the material into final position. Soft areas are progressively covered, possibly requiring thicker layers of prepared soil. Final grading by dozer is likely to be adequate. Where excessive compaction has occurred, such as along haulage routes, ripping may be required.

The essence of this reclamation scheme is the ability to excavate, after years of storage, the layered muskeg and overburden materials contained within the stockpile. Only operational experience can determine whether the method of preference is to construct a frozen or a semi- frozen stockpile. Furthermore, the overall shape of the stockpile, the embankment slope angles, and the orientation of the stockpile faces to the sunlight need to be ascertained by experimentation. The Consultants suggest that the scheme, if operated in a controlled fashion, would result in an adequately-blended prepared soil.

As at the Minimum Level of Reclamation, the option of blending with the bucket wheel excavator remains. Operational comments made earlier for the Minimum Level apply as well at the Improved Level of Reclamation.

Prepared Soil Manufacture at the Enhanced Level of Reclamation

Compared to the prepared soil manufacturing schemes adopted at the Minimum and Improved Levels of Reclamation, the scheme recommended at the Enhanced Level is rather complex. Because of the effort to achieve a superior quality of prepared soil, the capital investments and operating costs are correspondingly higher. The thickness of the prepared soil layer, however, remains at 1.0 m as was the case at the Improved Level of Reclamation. Figure 5.4.3-3, "Prepared Soil Manufacture at the Enhanced Level of Reclamation", gives a detailed schematic summary of the component operations.



NOTE: ORIENTATION OF FACES AND SLOPE ANGLES DESIGNED TO SUIT DEGREE OF THAWING AND TRAFFICABILITY.

PREPARED SOIL FROM LAYERED BLEND PILE AT THE IMPROVED LEVEL OF RECLAMATION

FIGURE 5.4.3-6

Potentially, three sources of muskeg exist: in situ muskeg from the mine blended with selected overburden by the overburden BWE, muskeg from the front-end loader and truck prestripping operations, and muskeg from an on-pit and off-pit hydraulic muskeg mining operation. A brief discussion of the technically most viable option, hydraulic muskeg mining, follows. The option for utilizing muskeg from the prestripping operation requires a special pre-blending thawing process. The thawing process is also briefly described.

Hydraulic muskeg mining is a seasonal operation starting in May and lasting into October. Muskeg deposits, either within the mine site or outside of it, are developed into muskeg mines. Large floating barges equipped with shredders and pumps produce a muskeg slurry which is transported via a 500 mm diameter steel pipeline to a dewatering plant. After entering the plant, the slurry is initially dewatered on screens. Belt presses remove enough water to produce a stiff pulp which is conveyed to the stacker. The major mechanical components and operational features are depicted in Figure 5.4.3-7, "Schematic of Muskeg Mining and Dewatering for the Enhanced Level of Reclamation".

Muskeg slurry from the hydraulic muskeg mine and also from the thawing trench scheme is fed by slurry pipeline to the dewatering plant, from which the product is next transported by conveyor to a stockpile area where a stacker blends the muskeg with suitable overburden supplied by the BWE from the mine. Prior to severe freezing of the stockpile, the annual production is removed from the stockpile area with a small track-mounted reclaimer feeding a low capacity distribution conveyor. A radial stacker is used to form field stockpiles. Distribution to the reclamation sites occurs as required, using front-end loaders and off-highway trucks. The spreading procedures are the same as those of the Improved Level of Reclamation, i.e. using medium-sized dozers.

In one season, a track-mounted stacker places almost 400,000 m^3 of muskeg and over 700,000 m^3 of overburden into stockpiles approximately 1,000 m in length. The storage capacity is 350,000 m^3. The stacking process serves as the initial phase of mixing. Mixed muskeg and overburden are removed from these stockpiles by a small crawler-mounted reclaimer loading onto a 1,000 mm wide belt with a capacity of 1,000 m^3/hour. Removal is on a two shift per day basis and lasts from May to November. A range diagram for the stacker and reclaimer are seen in Figure 5.4.3-8, "Layout - Blending and Storage Yard".

The conveyor terminates at predetermined field stockpile sites. As the mining progresses, these sites change, being located as close as possible to the areas onto which the prepared soil is to be spread. A small radial stacker at the end of the conveyor forms stockpiles. The prepared soil transport system is detailed in Figure 5.4.3-9, "Sketch of Blending and Storage Yard/Distribution Belt Conveyor System". Some time later, say, five or more years, front-end loaders and off-highway trucks distribute the prepared soil. This distribution occurs generally from the month of May to the month of February, inclusive. Depending on the spring weather, it may be possible to begin the summer hauling season somewhat earlier in some reclamation areas. Adequate traction on the reclamation areas, and thawing and drying of the blend piles are the major considerations governing the length of the hauling period.

Another option, considered to be less feasible than the previously described hydraulic mining scheme, utilizes a thawing trench. A muskeg thawing trench is constructed near the extraction plant. A sufficient quantity of muskeg must be stockpiled adjacent the "thawing trench" during the winter removal season to provide for one year of production. A dozer pushes the frozen blocks towards and into the pond. The operation of the thawing trench would likely be seasonal. Since, at the Enhanced Level, no wet tailings are generated, closed-circuit hot water lines provide the heat to thaw the frozen blocks. A hydraulic dredge shreds and pumps this "muskeg slurry" to a dewatering plant. If the Enhanced Level type of prepared soil is desired at the Improved Level, tailings lines running through the thawing trench could be a potential source of heat.

Although the Consultants feel that the above system overcomes or minimizes many problems (such as scheduling, availability, environmental impact and other important factors), the system introduces all the diffi-

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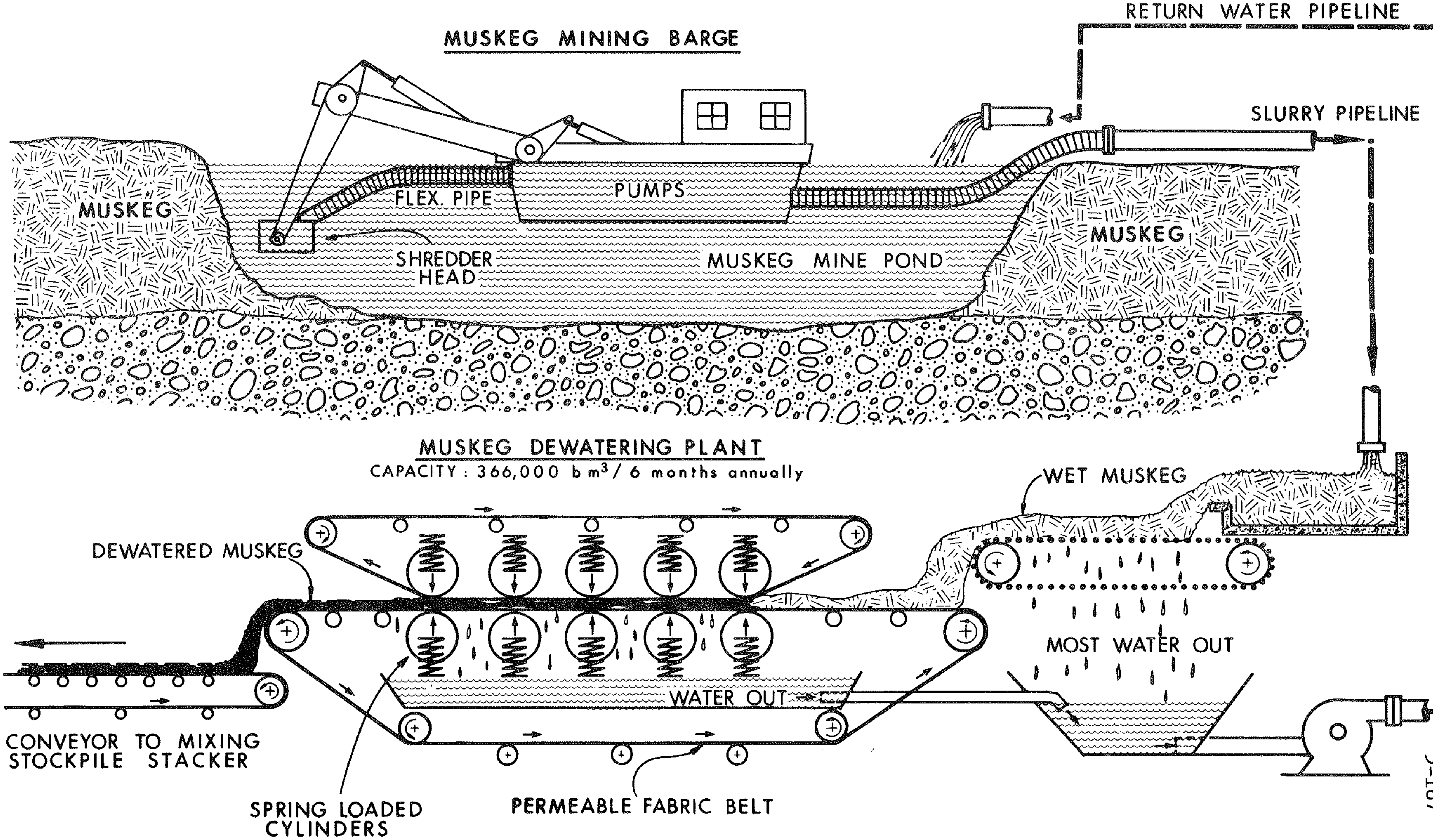
SCHEMATIC OF MUSKEG MINING AND DEWATERING FOR THE ENHANCED LEVEL OF RECLAMATION

FIGURE 5.4.3-7



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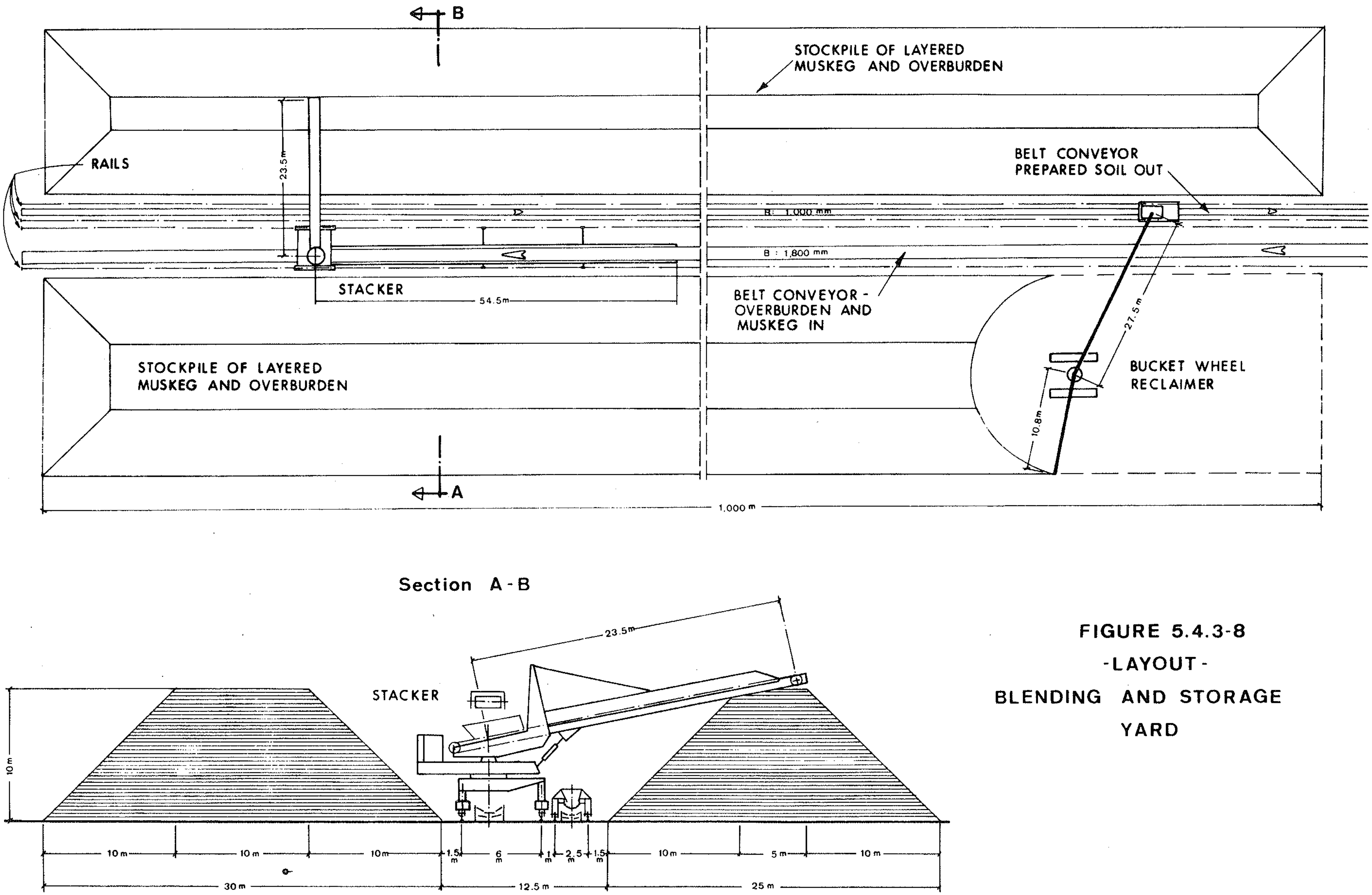

FIGURE 5.4.3-8
-LAYOUT-
BLENDING AND STORAGE
YARD

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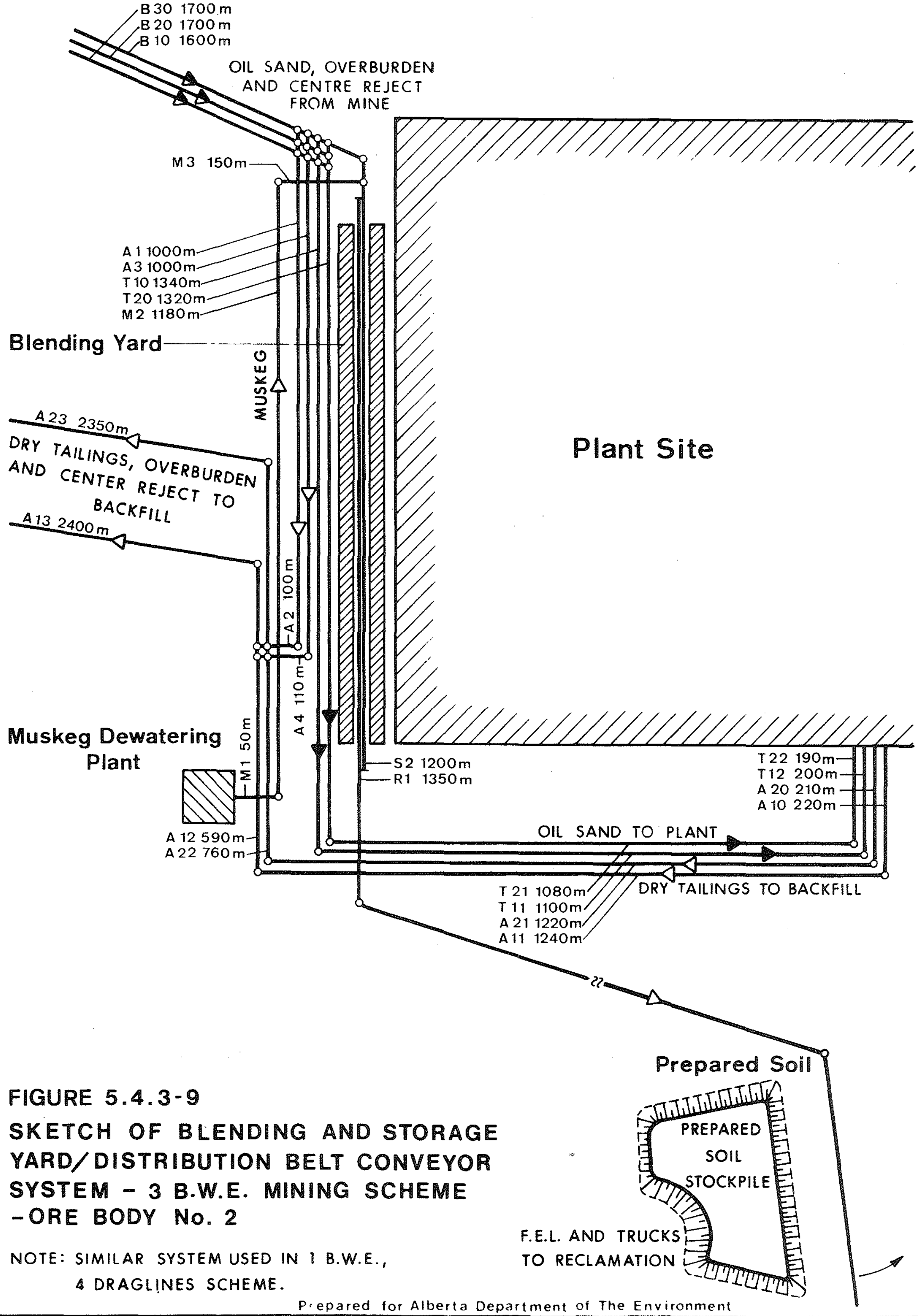

FIGURE 5.4.3-9

SKETCH OF BLENDING AND STORAGE YARD/DISTRIBUTION BELT CONVEYOR SYSTEM - 3 B.W.E. MINING SCHEME -ORE BODY No. 2

NOTE: SIMILAR SYSTEM USED IN 1 B.W.E., 4 DRAGLINES SCHEME.

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culties of operating a trench to process large volumes of muskeg. Maintaining adequate heat transfer, servicing of the heating pipes, and removal of silt and oversize materials could be particularly troublesome.

The major advantage of this system is that muskeg from within the mine, rather than from an outside "muskeg mine", would be used for prepared soil manufacture.

5.4.4 RECLAMATION OF TAILINGS DYKES AND DYKE ROADWAYS

A variety of slope angles and bench/roadway systems can be expected, depending on foundation conditions, on materials utilized in the construction of the dykes, and pond operating philosophy. Figures 5.4.4-1 and 5.4.4-2 show overall slopes of 3h:lv incorporating 4 and 3 roadways, respectively. The 4-roadway system has individual slopes of 5h:3v or 31°, a rather steep soil angle and consequently highly unfavourable for the establishment of vegetation. The 4-roadways system with individual slopes of 2h:lv or 26.5° is much more favourable, but still near the marginal in this regard.

Figures 5.4.4-3 and 5.4.4-4 depict an overall slope of 4h:lv including 4 and 3 roadways, respectively. Both systems have individual slope angles sufficiently shallow as to present few problems in reclamation. The 4-roadways system has individual slopes of 8h:3v or 20.5°. The 3-roadways systems has individual slopes of 3h:lv or 18.5°.

Figure 5.4.4-5, "Details of 20 m Wide Roadway", includes two cross sections of a typical roadway mentioned above; they outline reclamation procedures by showing the situations before and after reclamation. Consideration must be given to ensure that dykes drain effectively. Filters must remain exposed and protected with suitably-graded rip-rap. Drainage from filters must be channelled off the dyke surface to collecting basins. Improper handling of this aspect may endanger the stability of the dyke.

From a reclamation viewpoint, the 4:1 overall slopes are preferred since there is less likelihood of erosion either degrading the prepared-soil surface or displacing seed, and there are few restrictions on equipment utilization, allowing more versatility in soil preparation, seeding, and vegetation maintenance. A bench width of 20 metres will allow the construction of sufficiently wide roads and simplify the construction of turn-around points for small and mid-sized off-highway trucks. Small dozers would be used to push the materials down the slopes. However, it is also important to restrict the length of slopes between benches to minimize erosion. With respect to on-dyke roadways, the bench width necessary for placement of reclamation materials is kept within reasonable working limits (determined by operating characteristics of the haulage and spreading equipment).

During the operational life to the pond, culverts will be employed to pass runoff and seepage water under roads. Culverts are degradable as well as requiring annual inspection and possibly the removal of debris. Consequently, culverts cannot be considered for long-term runoff control without a commitment to maintenance after the cessation of operations. Surface runoff can be established between benches by preplanned longitudinal grading of benches and gentle inter-connecting ramps. This type of runoff management is feasible when the prepared soil depth is at least 1 m. The risk of damage to the slope goes up as less prepared soil is utilized and is the highest when seeding is done directly on tailings sand or only slightly amended sand. In the latter case internal drainage may require the employment of expensive internal drains but erosion by surface runoff will, nonetheless, continue to be a serious risk.

The recycle of large quantities of seepage water is greatly minimized or may not be required at all with sanded-in ponds. The mixing of runoff water with seepage water is, in most cases, sufficient dilution to avoid damage to vegetation. In those cases where seepage water toxicities are likely to be problematic, tolerant plant species should be selected. Perpetual maintenance of a seepage collection system is operationally unreliable and should be avoided.

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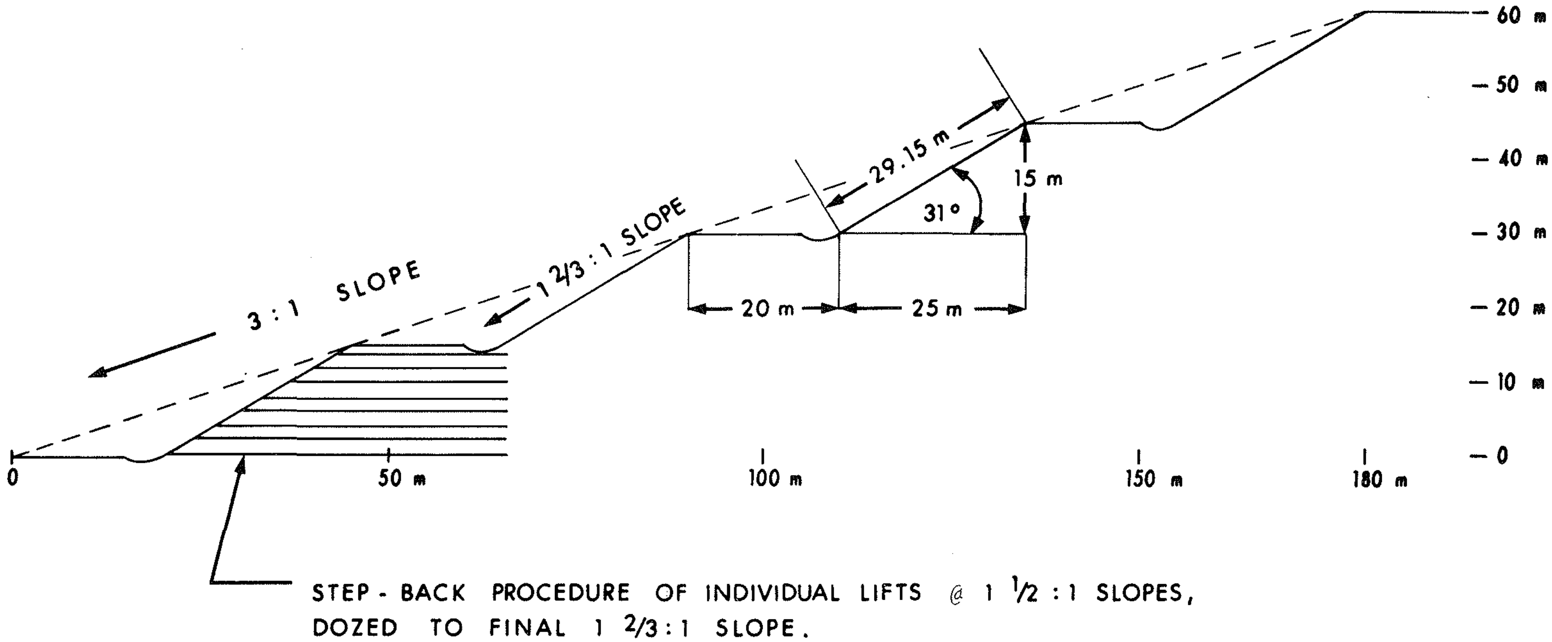

SCALE : 1 cm = 10 m

OVERALL 3:1 SLOPE INCLUDING 4 ROADWAYS

FIGURE 5.4.4 -1

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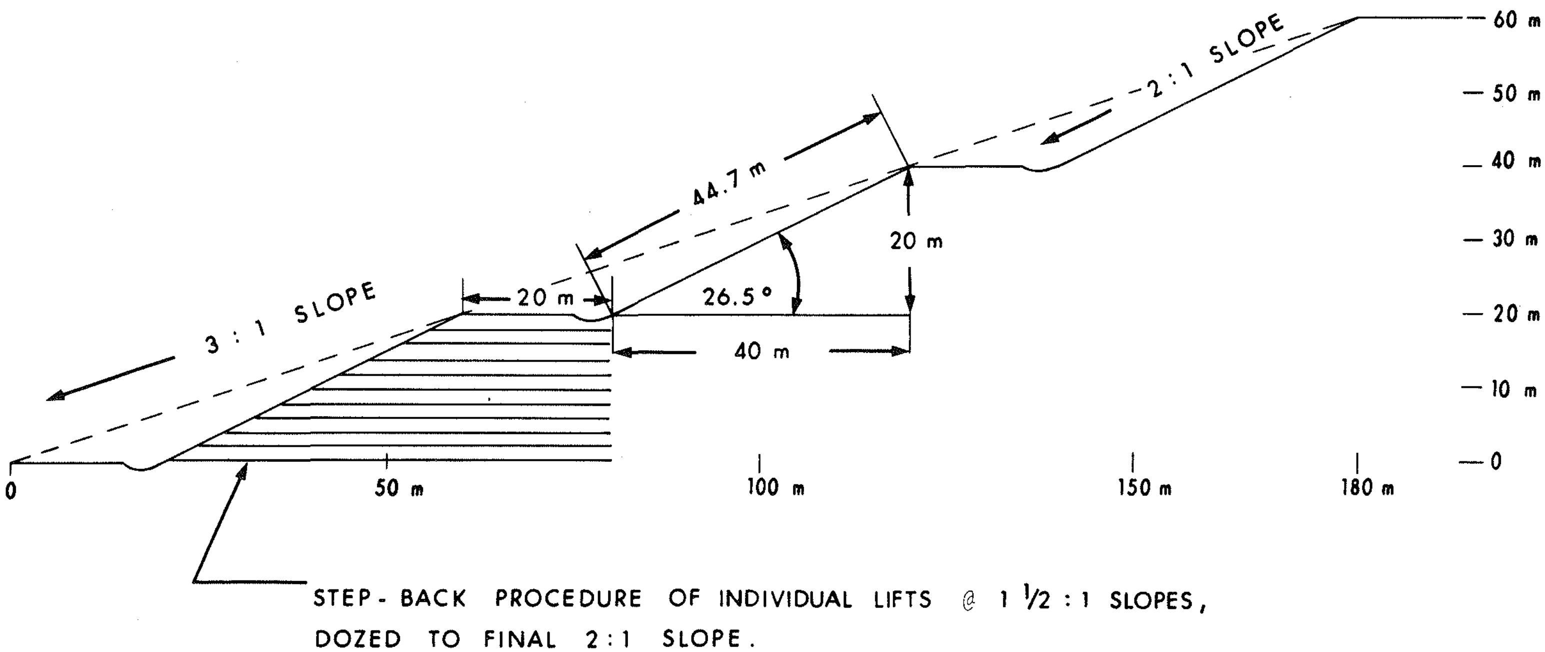
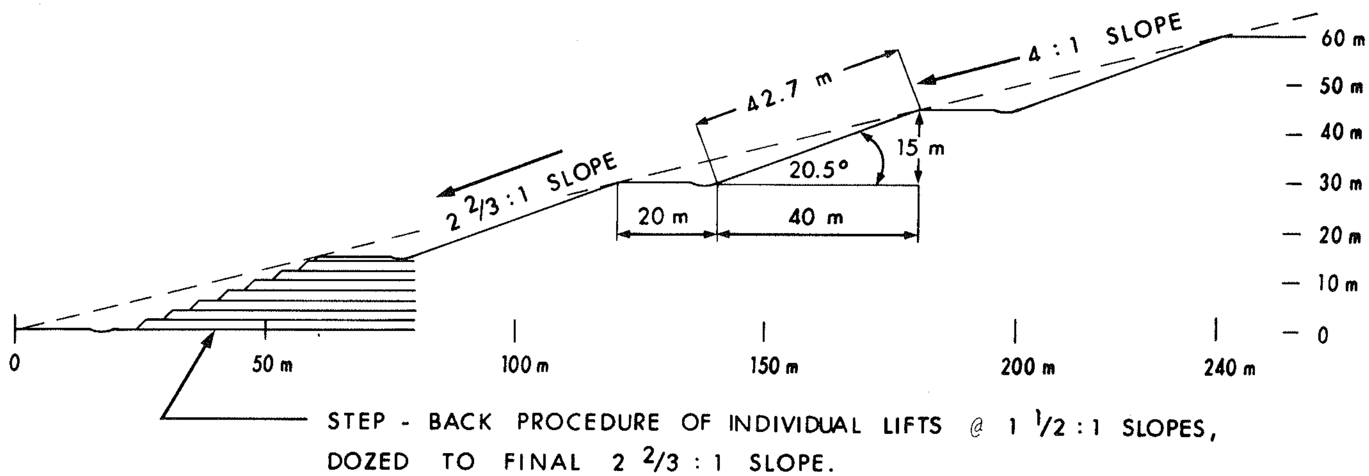


SCALE : 1 cm = 10 m

OVERALL 3:1 SLOPE INCLUDING 3 ROADWAYS

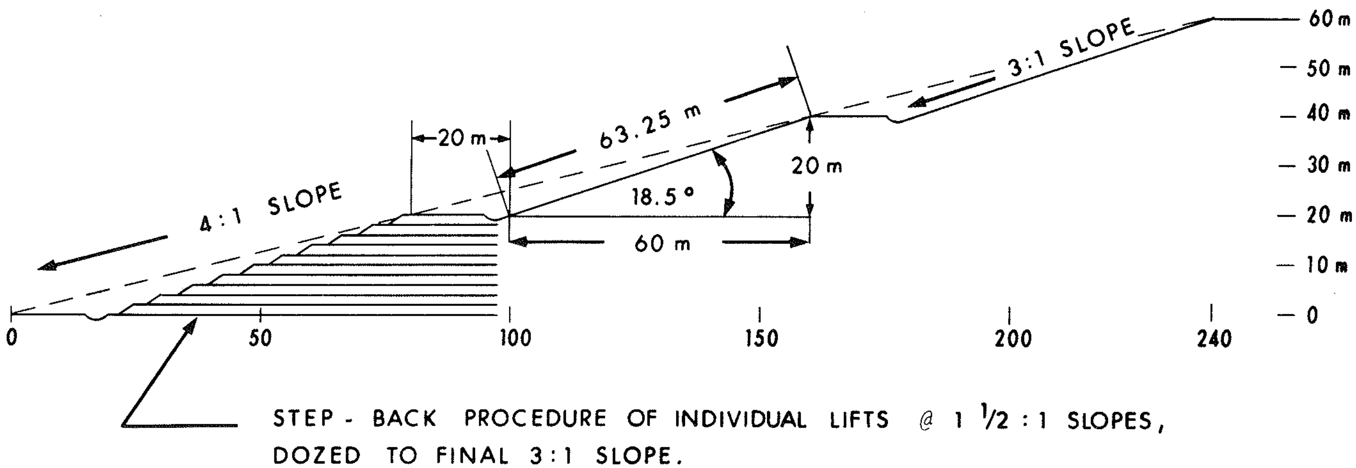
FIGURE 5.4.4-2



SCALE : 1 cm = 15 m

OVERALL 4:1 SLOPE INCLUDING 4 ROADWAYS

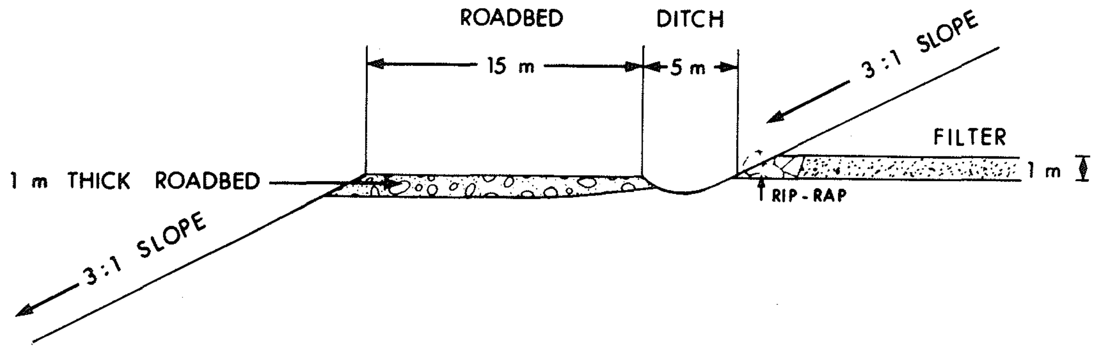
FIGURE 5.4.4 -3



SCALE : 1 cm = 15 m

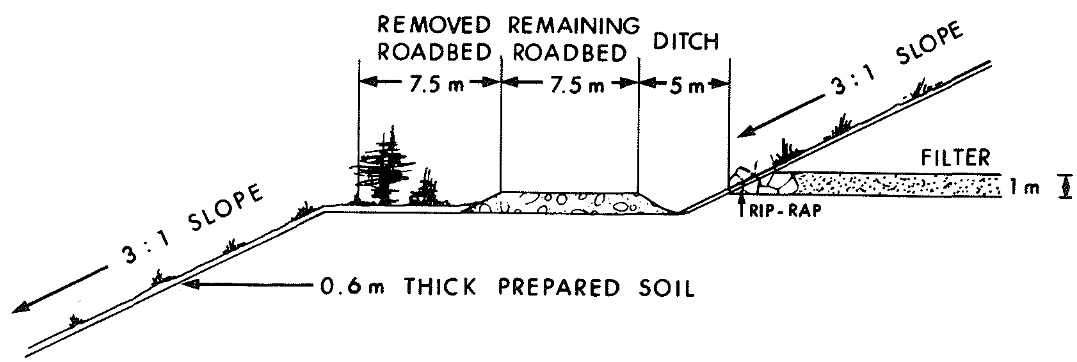
OVERALL 4:1 SLOPE INCLUDING 3 ROADWAYS

FIGURE 5.4.4-4



BEFORE RECLAMATION

SCALE : 1 cm = 4 m



AFTER RECLAMATION

SCALE : 1 cm = 4 m

REFER TO FIGURE 5.4.4-4 FOR OVERALL SLOPE

DETAILS OF 20 m WIDE ROADWAY

FIGURE 5.4.4-5

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5.4.5 RECLAMATION OF TAILINGS POND BEACHES

Since beaches form at a slope approaching 6h:1v, no equipment limitations are anticipated, with the possible exception of trafficability where the beached sands may support less weight than dyked sands and/or roadways. Beaches to be reclaimed include interior beaches off the tailings pond dyke, and beaches formed at the edges of in-pit ponds. Spreading prepared soil during the winter while the beach sands are frozen is likely to eliminate considerable road construction effort.

5.4.6 RECLAMATION OF DISTURBANCES RELATED TO ANCILLARY FACILITIES

Reclamation of disturbances related to ancillary facilities includes:

- breaking up and scarifying all roadbeds. In certain cases the addition of small amounts of prepared soil may be appropriate.

- salvage and removal of all temporary or permanent structures such as extraction and treatment plants, pipelines, transmission lines, tankfarms, washhouses, warehouses, sewage plant, maintenance sheds, mess halls, and bunkhouses. Resurfacing with prepared soil is required for refuse dumps and abandoned plant site areas.

- backfilling all small ponds associated with the plant site, except possibly large freshwater makeup ponds.

- salvaging or safe disposal of all chemical by-products such as coke and sulphur.

- construction of permanent mine site drainage courses to protect reclaimed areas and to preserve local watercourse quality.

None of the costs of the above items except for resurfacing of the plant site area are examined and costed in this report. A further explanation as to the approach taken here is provided in Chapter 6, Costing Methods.

5.4.7 RECLAMATION OF TAILINGS SLUDGE PONDS

Tailings systems such as those presently used by G.C.O.S. Ltd. and Syncrude Canada Ltd. produce not only large quantities of waste that is normally stored considerably above the natural ground surface, but also volumes of associated poor-quality water bearing the following problem contaminants: clay and silt fines, bitumen, caustic soda, naphtha, heavy metals, and phenols. Various methods of purifying or clarifying these waters to make them capable of supporting aquatic life and rendering them non-toxic to waterfowl and other wildlife are under investigation. These methods have been listed[7].

An alternative emphasis is to minimize the areal extent of fluid waste disposal areas. This may be done either by containing these liquids in small, but very deep ponds, or by treating the tailings to remove additional water (see details in Section 5.2, Plant Process Considerations, and Chapters 7.0, 8.0 and 9.0, Concepts and Costs of Development and Reclamation of 120,000, 60,000 and 240,000 BPCD Oil Sands Mines, respectively).

In those cases where fluid sludge can be dewatered and concentrated into a fairly small area, the Consultants have encouraging evidence to indicate that these ponds can eventually be resurfaced and vegetated. The development of "reclamation concepts" at the Improved Level and the detailed reclamation plans provided do not seriously address the reclamation of sludge ponds. However, it is encouraging to note that considerable research in other industries faced with toxic tailings pond problems is approaching the point of satisfactory breakthrough[8].

The approach for complete reclamation of treated sludge ponds from oil sands extraction processes probably involves a systematic consolidation or solidification of the surface layer of thickened sludge (probably by chemical means), and the subsequent beaching or poldering of a light soil (especially prepared fine sand, coke, flyash, or peat) into the pond. The surface should then be solid enough to support light-weight agricultural equipment. The costs for such reclamation appear high.

The blending of sludge into the dry tailings stream from an extraction plant producing dry tailings appears to be technically practical and achievable at considerably less cost than the reclaiming of a sludge pond surface. Since the dry tailings stream may need the addition of water to ensure dump stability and help control dust, the injection of sludge at transfer points is of interest. A detrimental side-effect may be the increase of dry tailings toxicity, but this can be countered by the selective dumping techniques achievable with a spreader. Non-toxic materials, if available, can be placed as the uppermost layer. Until dry extraction processes are commercially developed, intensive research into reclaiming sludge ponds should be conducted.

5.5 TECHNICAL REVIEW OF MAJOR MATERIALS HANDLING EQUIPMENT

The major earth moving equipment was selected to suit the materials handling requirements of each mine plan. BWE's, spreaders, draglines, dragline hoppers, and conveyors are the main types of machinery utilized. In addition, mobile machinery such as trucks, loaders, dozers, pipelayers, etc. are employed. These types of units are available in many size ranges from a large number of manufacturers. No specifications are provided for small mobile machinery since these are off-the-shelf units, and are not custom-built for the mine operator. This section provides typical technical data for the main materials handling equipment for the 12 developed mine plans by means of sketches and tables outlining the main machine specifications.

No manufacturer is inferred by any given machine specifications. During the course of the Consultants' customary activities, equipment is assessed, specifications are prepared, and design changes recommended to clients. However, neither Techman Ltd. nor Rheinbraun-Consulting GmbH manufacture or sell equipment, nor are the Consultants representatives or agents for any manufacturer or supplier of equipment.

Figure 5.5-1 shows a sketch of one of the three bucket wheel excavators employed in the 120,000 BPCD mine (Ore Body 2 - bucket wheel scheme).

Figure 5.5.-2 shows a sketch of a dragline dumping material into a hopper. This is a typical arrangement used in Ore Body 2 and 4-dragline schemes.

Figure 5.5-3 shows a sketch of a spreader, which is used for dumping waste material into piles. The dimensions indicated apply to all the spreaders used in the 12 mine plans, however, each plan requires a spreader with certain capacity. As the spreader's capacity changes, so does the installed power, weight and the capital and operating costs.

Figure 5.5-4 shows a sketch of the tripper car removing waste material from a conveyor belt onto a short boom which in turn dumps it onto receiving boom of a spreader (shown in dashed lines).

Figure 5.5-5 shows a sketch of a conveyor drive station. The drive station houses electric motors propelling the belt via head and drive pullies, as well as belt tensioning devices and associated pullies. The structure rests on two pedestals, but at time of conveyor shifting, a transport crawler (see Figure 5.5-8) lifts and moves the drive station into the new position.

Figure 5.5-6 shows a section, plan and side view of a typical belt conveyor. The conveyors used in the study employ high tension conveyor belts reinforced with steel cables and covered with a low temperature, oil-resistant rubber compound. The belt moves on prelubricated Garland-type idlers; return idlers have rubber discs to reduce material build-up on idlers. The belt surface is sprayed with diesel fuel to prevent oil sand from sticking to the belt surface. An oil sand build-up on the belt is detrimental to proper operation of pullies and return idlers, and if left uncontrolled, would damage the belt, pullies, idlers or other parts of the system.

Figure 5.5-7 shows a schematic of a belt conveyor shunting head. It is supported and moved as described for the drive station, however much less frequent shifting is required of the distribution point, which a shunting head is part of. The idea of a shunting head is, that without changing the actual length of the belt, the conveyor can extend and dump different materials on different conveyors (three shown). The two uppermost pullies are attached to a frame (shown in dashed line) which slides back and forth allowing accurate positioning. The distribution point is a very effective and simple tool for selective handling of mined materials without loss of mine production.

Figure 5.5-8 shows a section and plan of the transport crawler. The main function of this support equipment is to move the drive stations of the face conveyors. Other drive stations and possibly shunting heads will require use of the transport crawler much less frequent.

Table 5.5-1 indicates the main design parameters of BWE's, draglines and hoppers used in the study. For capital costs, see Table. 6.1-3.

Table 5.5-2 indicates the main design parameters of spreaders and tripper cars used in the study.

Table 5.5-3 indicates the main design parameters of belt conveyors used in the study.

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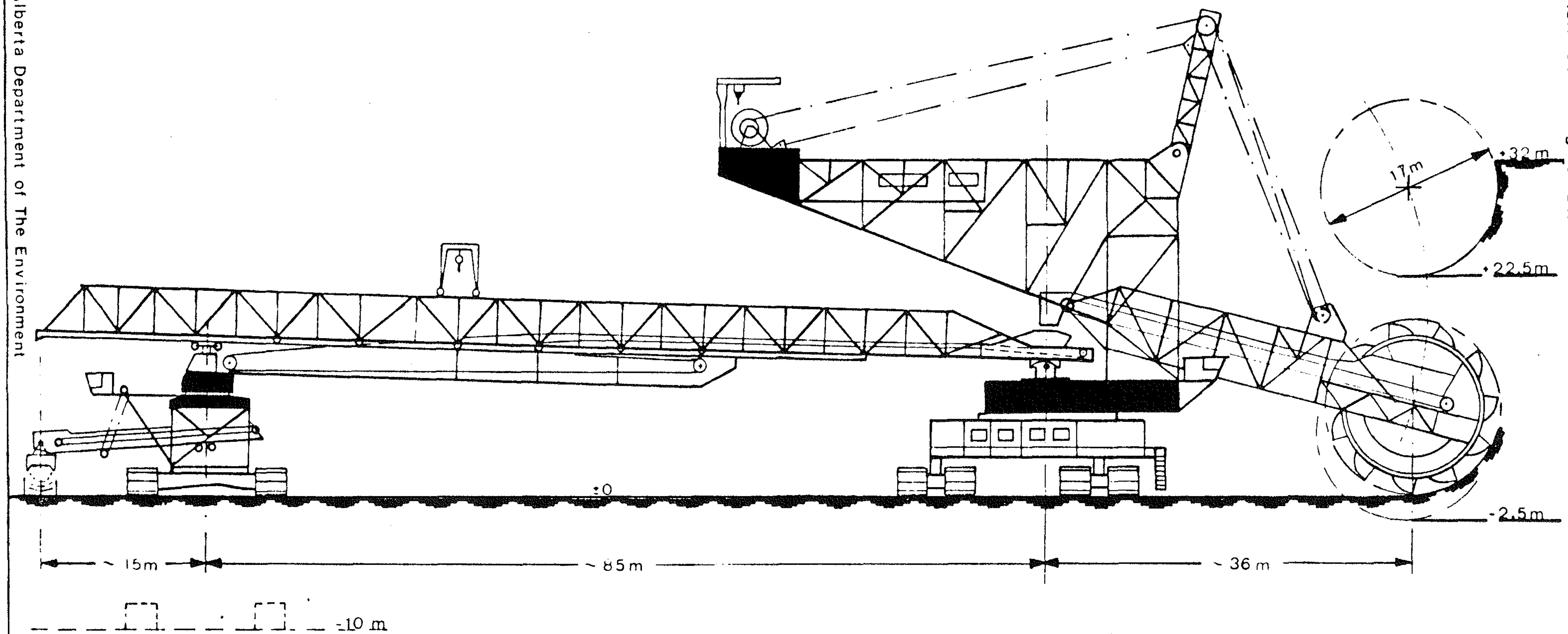

FIGURE 5.5-1

BUCKET WHEEL EXCAVATOR

(120,000 BPCD)

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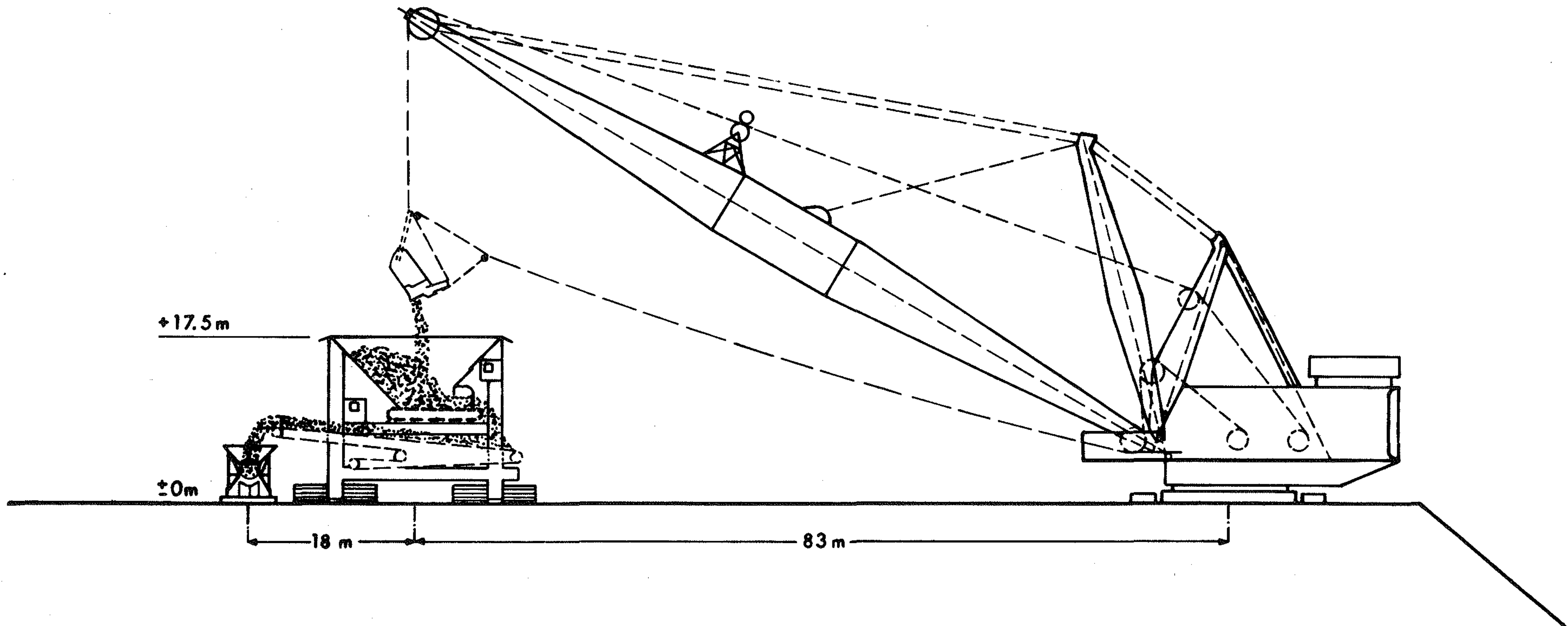

DRAGLINE WITH HOPPER

FIGURE 5.5-2

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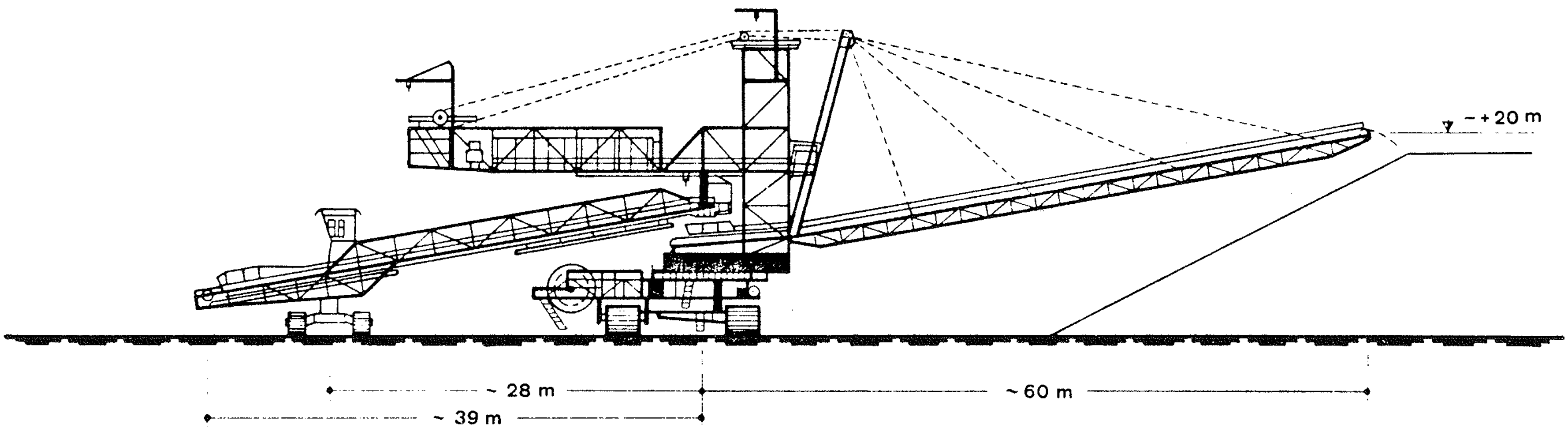

FIGURE 5.5-3

SPREADER

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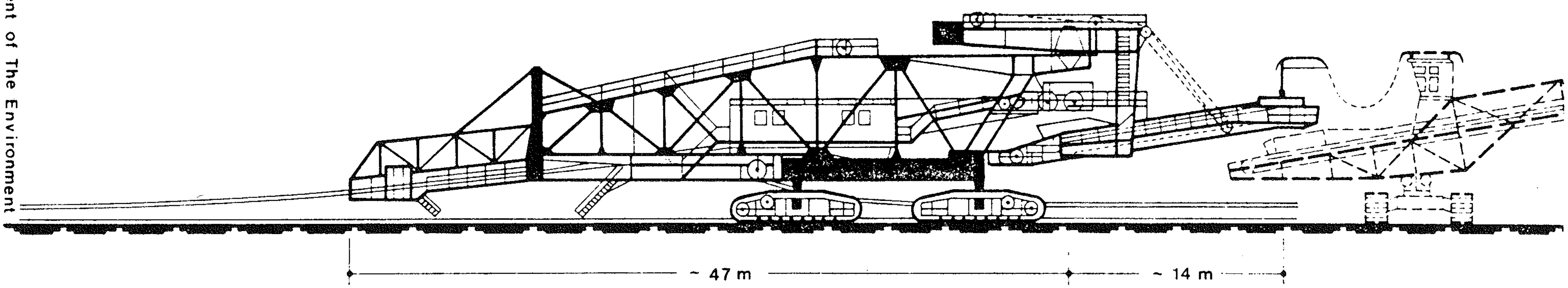

FIGURE 5.5-4

TRIPPER CAR

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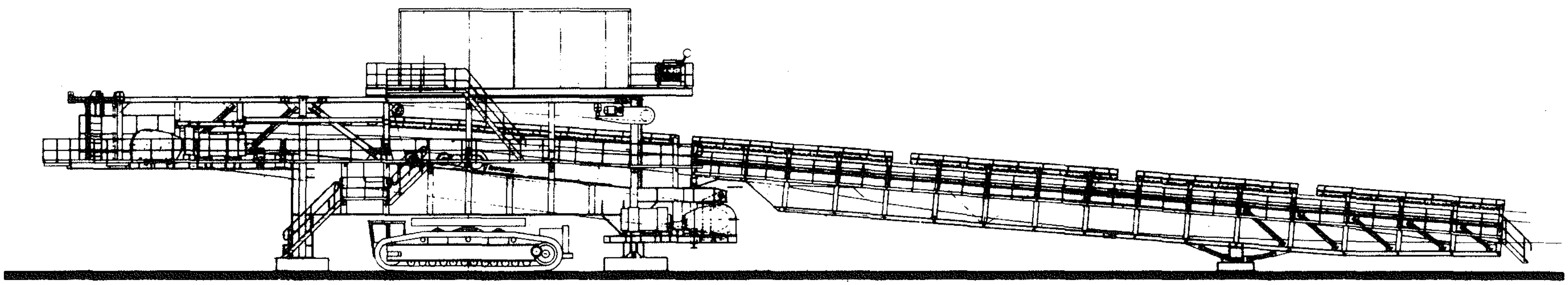

FIGURE 5.5-5

DRIVE STATION

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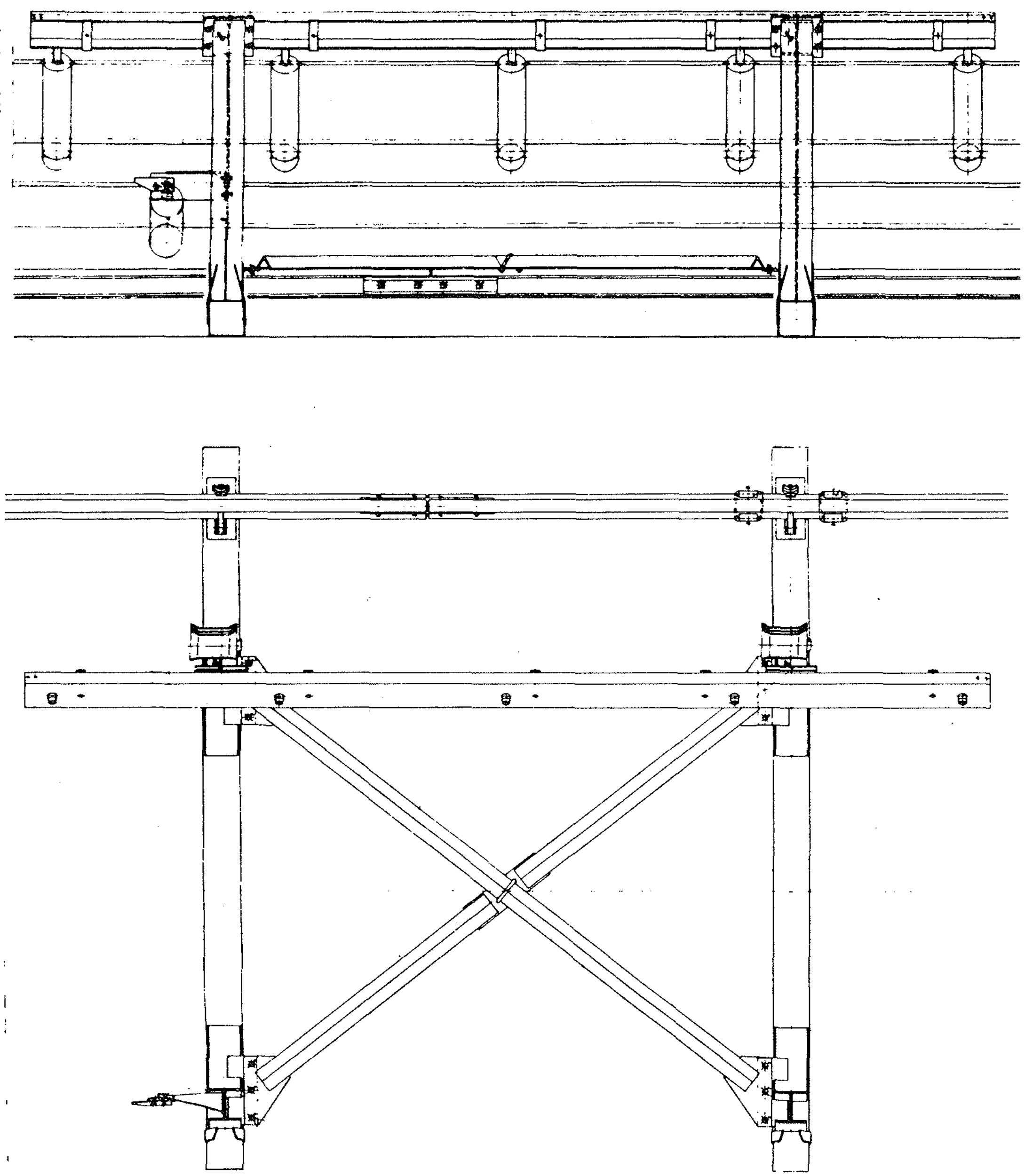

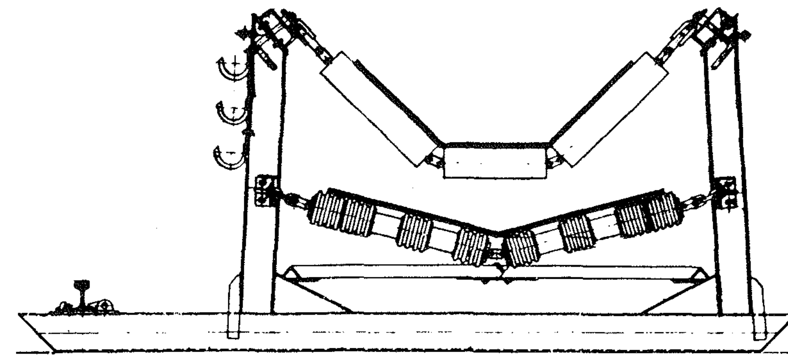

FIGURE 5.5-6

BELT CONVEYOR FRAME

(shiftable type)

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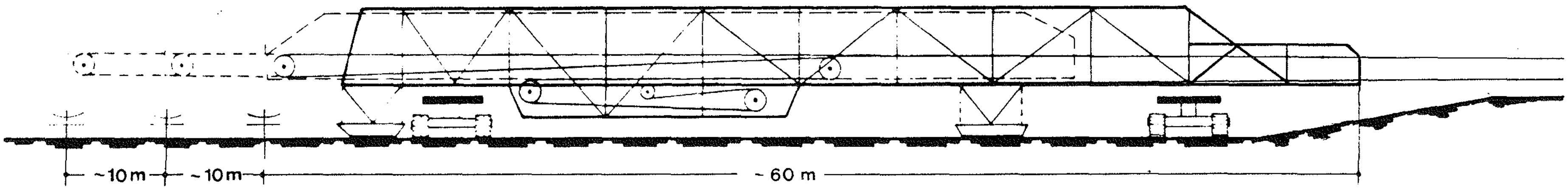

FIGURE 5.5-7

SHUNTING HEAD

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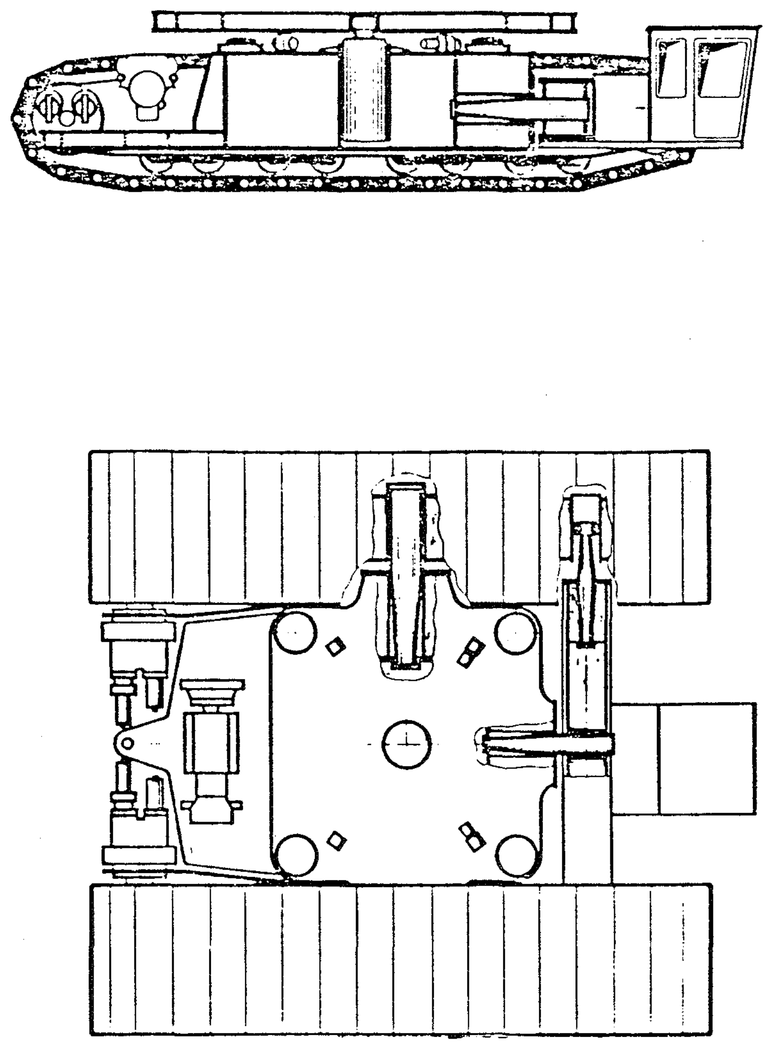

FIGURE 5.5-8

TRANSPORT CRAWLER

23-5-79

TABLE 5.5-1

Specifications: Materials Handling Equipment

Bucket Wheel Excavators, Draglines, and Dragline Hoppers

		Ore Body No. 2 (120,000 BPCD)				Ore Body No. 4 (60,000 BPCD)		Ore Body No. 1 (240,000 BPCD)	
		BWE/DL System		BWE System		BWE/DL System	BWE System	BWE System	
Description	Units	Min/Impr	Enhanced	Min/Impr	Enhanced	Min/Impr	Min/Impr	Minimum	Enhanced
Bucket Wheel Excavators									
Number of BWE per mine	-	1	1	3	3	1	3	6	6
Average Hourly Capacity	bank m^3/h	2,900	2,900	4,000	4,000	1,780	2,300	4,500	4,500
Nominal bucket capacity	m^3	2.0	2.0	4.5	4.5	1.4	2.3	4.5	4.5
Bucket wheel diameter	m	14.0	14.0	14.0	14.0	11.5	12.25	17.5	17.5
Number of buckets	-	14	14	10	10	9	10	11	11
Wheel rotary speed	RPM	3.57	3.57	3.85	3.85	5.28	4.33	3.93	3.93
Cutting height	m	32.0	32.0	32.0	32.0	20.0	30.0	30.0	30.0
Selective digging height	m	24.0	24.0	22.5	22.5	11.5	22.0	17.75	17.75
Boom length	m	36	36	36	36	36	36	36	36
Bridge length	m	85 9	85 9	85 9	85 9	85 9	85 9	85 9	85 9
Discharge boom length	m	15 2	15 2	15 2	15 2	15 2	15 2	15 2	15 2
Maximum Ground Slope									
-during digging	h:v	20:1	20:1	20:1	20:1	20:1	20:1	20:1	20:1
-during deadheading	h:v	18:1	18:1	18:1	18:1	18:1	18:1	18:1	18:1
Installed Power	kW	3,000	3,000	6,000	6,000	2,950	4,000	6,200	6,200
Service Weight	t	2,800	2,800	4,750	4,750	2,400	3,050	5,150	5,150
Draglines									
Number of D/L per mine	-	4	4	-	-	2	-	-	-
Average Hourly Capacity	bank m^3/h	2,800	2,800	-	-	2,500	-	-	-
Average swing angle	deg	45-50	45-50	-	-	45-50	-	-	-
Nominal bucket capacity	m^3	75	75	-	-	70	-	-	-
Maximum allowable load	kg	195,000	195,000	-	-	180,000	-	-	-
Boom length	m	92	92	-	-	92	-	-	-
Dumping radius	m	83	83	-	-	83	-	-	-
Maximum dumping height	m	40	40	-	-	40	-	-	-
Digging depth	m	45	45	-	-	45	-	-	-
Hoist motors, eight	kW	6,300	6,300	-	-	6,000	-	-	-
Drag motors, six	kW	4,700	4,700	-	-	4,600	-	-	-
Swing motors, four	kW	3,200	3,200	-	-	3,000	-	-	-
Propel motors, four	kW	2,000	2,000	-	-	2,000	-	-	-
AC driving motors	kW	7,700	7,700	-	-	7,500	-	-	-
Weight	kg	4,650,000	4,650,000	-	-	4,630,000	-	-	-

TABLE 5.5-1 (Continued)

Specifications: Materials Handling Equipment

Bucket Wheel Excavators, Draglines, and Dragline Hoppers

		Ore Body No. 2 (120,000 BPCD)				Ore Body No. 4 (60,000 BPCD)		Ore Body No. 1 (240,000 BPCD)	
		BWE/DL System		BWE System		BWE/DL System	BWE System	BWE System	
Description	Units	Min/Impr	Enhanced	Min/Impr	Enhanced	Min/Impr	Min/Impr	Minimum	Enhanced
Dragline Hoppers									
Number of D/L Hoppers per mine	-	4	4	-	-	2	-	-	-
Maximum throughput	bank m^3/h	6,000	6,000	-	-	5,600	-	-	-
Hopper top opening	m	18x18	18x18	-	-	18x18	-	-	-
Hopper height	m	17.5	17.5	-	-	17.5	-	-	-
Hopper volume	m^3	1,000	1,000	-	-	1,000	-	-	-
Apron feeder width	m	3.05	3.05	-	-	3.05	-	-	-
Apron feeder speed									
- maximum	m/min	22	22	-	-	20	-	-	-
- average	m/min	14	14	-	-	12	-	-	-
Empty weight	kg	1,905,000	1,905,000	-	-	1,905,000	-	-	-
Max. Payload	kg	2,504,000	2,504,000	-	-	2,504,000	-	-	-
Installed Power	kW	1,900	1,900	-	-	1,900	-	-	-

TABLE 5.5-2

Specifications: Materials Handling Equipment

Spreaders & Tripper Cars

Description	Units	Ore Body No. 2 (120,000 BPCD) BWE/DL System Min/Impr	Ore Body No. 2 (120,000 BPCD) BWE/DL System Enhanced	Ore Body No. 2 (120,000 BPCD) BWE System Min/Impr	Ore Body No. 2 (120,000 BPCD) BWE System Enhanced	Ore Body No. 4 (60,000 BPCD) BWE/DL System Min/Impr	Ore Body No. 4 (60,000 BPCD) BWE System Min/Impr	Ore Body No. 1 (240,000 BPCD) BWE System Minimum	Ore Body No. 1 (240,000 BPCD) BWE System Enhanced
Spreaders									
Number of spreaders per mine	-	1	2	1	2	1	1	4	6
Theoretical capacity	loose m^3/h	16,060	16,060	16,060	16,060	7,200	10,520	12,430	16,060
Length of discharge boom	m	60	60	60	60	60	60	60	60
Height of discharge boom	m	20	20	20	20	20	20	20	20
Discharge boom rotation	deg	210	210	210	210	210	210	210	210
Length of receiving boom	m	39	39	39	39	39	39	39	39
Conveyor belt width	mm	2,600	2,600	2,600	2,600	1,800	2,200	2,400	2,600
Maximum Ground Slope									
- during dumping	h:v	20:1	20:1	20:1	20:1	20:1	20:1	20:1	20:1
- during deadheading	h:v	18:1	18:1	18:1	18:1	18:1	18:1	18:1	18:1
Installed power	kW	3,000	3,000	3,000	3,000	2,000	2,600	2,850	3,000
Service Weight	t	2,000	2,000	2,000	2,000	1,370	1,700	1,850	2,000
Tripper Cars									
Number of tripper cars per mine	-	1	2	1	2	1	1	4	6
Length to discharge pulley	m	47	47	47	47	47	47	47	47
Boom length	m	14	14	14	14	14	14	14	14
Boom height									
- maximum	m	9	9	9	9	9	9	9	9
- minimum	m	6	6	6	6	6	6	6	6
Installed power (bench belt drive not included)	kW	650	650	650	650	450	500	600	650
Service Weight	t	640	640	640	640	400	450	600	640

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TABLE 5.5-3

Specifications: Materials Handling Equipment

Belt Conveyor Systems

Description	Units	Ore Body No. 2 (120,000 BPCD) BWE/DL System Min/Impr	Ore Body No. 2 (120,000 BPCD) BWE/DL System Enhanced	Ore Body No. 2 (120,000 BPCD) BWE System Min/Impr	Ore Body No. 2 (120,000 BPCD) BWE System Enhanced	Ore Body No. 4 (60,000 BPCD) BWE/DL System Min/Impr	Ore Body No. 4 (60,000 BPCD) BWE System Min/Impr	Ore Body No. 1 (240,000 BPCD) BWE System Minimum	Ore Body No. 1 (240,000 BPCD) BWE System Enhanced
Belt Conveyor Systems									
1. Top Bench Conveyors:									
Throughput	loose m^3/h	6,747	6,747	10,380	10,380	4,000	5,970	11,680	11,680
	t/h	11,212	11,212	17,250	17,250	6,650	8,600	16,820	16,820
Belt width	mm	1,800	1,800	2,200	2,200	1,600	1,800	2,400	2,400
Troughing	deg	45	45	45	45	45	45	45	45
Belt Speed	m/s	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2
2. Middle & Bottom Bench Conveyors:									
Throughput	loose m^3/h	5,641(2)	5,641(2)	10,380	10,380	3,560	5,970	11,680	11,680
	t/h	8,123	8,123	14,950	14,950	5,130	8,600	16,820	16,820
Belt width	mm	1,800	1,800	2,200	2,200	1,600	1,800	2,400	2,400
Troughing	deg	45	45	45	45	45	45	45	45
Belt speed	m/s	5.2	5.2	5.2	5.2	4.0	5.2	5.2	5.2
3. Plant Feed Conveyors:									
Throughput	loose m^3/h	11,282	11,282	10,380	10,380	3,560	5,970	11,680	11,680
	t/h	16,246	16,246	14,950	14,950	5,130	8,600	16,820	16,820
Belt width	mm	2,200	2,200	2,200	2,200	1,600	1,800	2,400	2,400
Troughing	deg	45	45	45	45	45	45	45	45
Belt speed	m/s	5.2	5.2	5.2	5.2	4.0	5.2	5.2	5.2
4. Dry Tailings Conveyors:									
Throughput	loose m^3/h	-	5,200	-	4,620	-	-	-	9,100
	t/h	-	8,320	-	7,400	-	-	-	14,560
Belt width	mm	-	1,800	-	2,600	-	-	-	2,400
Troughing	deg	-	45	-	45	-	-	-	45
Belt speed	m/s	-	5.2	-	5.2	-	-	-	5.2
5. Waste Dump Conveyors:									
Theoretical capacity	loose m^3/h	16,060	16,060	16,060	16,060	7,200	10,520	12,430	16,060
	t/h	24,620*	24,620*	25,120*	25,120*	11,260*	16,290*	20,130*	25,360*
Belt width	mm	2,600	2,600	2,600	2,600	1,800	2,200	2,400	2,600
Troughing	deg	45	45	45	45	45	45	45	45
Belt Speed	m/s	5.2	5.2	5.2	5.2	5.2	5.2	5.2	5.2

TABLE 5.5-3 (Continued)

Specifications: Materials Handling Equipment

Belt Conveyor Systems

		Ore Body No. 2 (120,000 BPCD)				Ore Body No. 4 (60,000 BPCD)		Ore Body No. 1 (240,000 BPCD)	
		BWE/DL System		BWE System		BWE/DL System	BWE System	BWE System	
Description	Units	Min/Impr	Enhanced	Min/Impr	Enhanced	Min/Impr	Min/Impr	Minimum	Enhanced
6. Overburden for Prepared Soil Conveyor:									
Throughput	loose m^3/h	-	6,747	-	8,960	-	-	-	10,080
	t/h	-	11,212	-	14,900	-	-	-	16,750
Belt width	mm	-	1,800	-	2,200	-	-	-	2,400
Troughing	deg	-	45	-	45	-	-	-	45
Belt speed	m/s	-	5.2	-	5.2	-	-	-	5.2
7. Prepared Soil Conveyor									
Throughput	loose m^3/h	-	420	-	440	-	-	-	660
	t/h	-	530	-	560	-	-	-	840
Belt width	mm	-	1,000	-	1,000	-	-	-	1,000
Troughing	deg	-	30	-	30	-	-	-	30
Belt speed	m/s	-	2.0	-	2.0	-	-	-	3.0

(2) Indicates two systems per bench.

* Note: tonnage will vary with ratio of OB:CR:DRY TAILINGS (where applicable)

References for Chapter 5.0

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6.0 COSTING METHODS

6.1 COSTING OBJECTIVES OF STUDY

a. General Costing Criteria

Operating costs and capital costs were determined for two mining methods (dragline and bucket wheel excavator mining), three associated levels of reclamation (Minimum, Improved, and Enhanced), and three mine sizes (60,000 BPCD, 120,000 BPCD, and 240,000 BPCD). By following costing guidelines rigorously, it was possible to compare mining and associated reclamation costs for the three synthetic crude-oil production cases.

A prime objective of the study was to determine which major categories of operations would be significantly affected by changes in mining method and choice of reclamation level, and so warrant detailed costing. Techman/RC determined that only three major operations would be significantly affected: mining, equipment maintenance, and planning operations. The above-mentioned categories are shown in Drawing No. BR 22900-16-00, Typical Departmental Chart. The Consultants considered the categories of "administration", "general services" and "ancillary services" not to have significant bearing on the net cost of mining and reclamation activities of the three synthetic crude oil production cases. Furthermore, although the capital and operating costs of the extraction plant vary as the quality and quantity of plant feed varies from mine to mine due to variations in geology, and somewhat due to variations caused by the selected mining method, no attempt is made to specify the added cost of bitumen extraction. The added cost of associated facilities such as the power plant is also excluded.

"Equipment maintenance" and "planning operations" must be matched with the requirements of "mine operations". Typical organization charts for "mine", "maintenance" and "planning" operations are illustrated in Drawing Nos. BR22900-17-00, BR22900-18-00 and BR22900-19-00.

The three major designated categories are each divided into six main cost centres, which are further subdivided into cost sub-centres. The main cost centres which determine the total cost of "mine operations" and the associated categories of "equipment maintenance" and "planning" are as follows:

1. Civil Construction Activities
2. Removal of Organic Materials and Soils
3. Overburden, Reject, and Oil Sands Handling
4. Tailings Disposal
5. Establishment of Ultimate Land Use Resources
6. Supervision and Technical Services

The division of each main cost centre into cost sub-centres is explained in detail in Section 6.2, Typical Oil Sands Mine Unit Costs. Each cost sub-centre described consists of an operating and a capital cost element.

A major concern in reporting costs is that the reported costs change with time. Consequently, tables summarizing unit capital and labour costs are included, and may be used to derive an index whereby the costs provided in the cost centres and the cost sub-centres can be updated. Hourly equipment operating rates are not published in this report since these are proprietary information.

b. Operating Costs

The determination of the costs of conducting mining, tailings disposal, and reclamation activities requires that hourly, monthly, or annual rates for activities be developed. The operating cost of any given activity consists of rates for the following prime accounts:

- i - operating salaries
- ii - operating labour
- iii - equipment operating costs
- iv - operating supplies and materials
- v - subcontracts

These accounts are further discussed below:

i. Operating salaries

The salary schedule for all staff personnel is listed in Table 6.1-1, Staff Salary Schedule. The listed annual salaries exclude the payroll burden for staff personnel, which is assumed to be 18% of the direct salary costs, and includes unemployment insurance, Canada Pension Plan, company pension plan, Worker's Compensation, health care, life insurance, and statutory holidays. Staff employees may receive other special benefits reflecting working conditions, relocation costs, etc. An average mean adjustment to the payroll burden for this type of cost is difficult to estimate and, therefore, was not included.

ii. Operating labour

The hourly labour rates are listed in Table 6.1-2, Hourly Labour Schedule. An analysis was made of the wage rate structure of G.C.O.S. Ltd. which has a union agreement with the Fort McMurray Independent Oil Workers (M.I.O.W.). Syncrude Canada Ltd. does not work under a union agreement, but the labour rates are similar to the rates of G.C.O.S. Ltd. In 1978, a new labour agreement was being negotiated between G.C.O.S. management and the union; consequently, the Consultants used the 1977 base rates as the labour benchmark.

The wage rate structure of G.C.O.S. Ltd., consisting of 15 different hourly base rates, was narrowed down to 4 hourly base rates (Group I to Group IV). In order to arrive at the 1978 base rates, the 1977 base rates were escalated by 6%. The escalation assumed is strictly for study purposes (all without prejudice).

The fringe benefits were assumed to be 24% of the hourly 1978 base rates, including unemployment insurance, Canada Pension Plan, company pension plan, vacation pay, statutory holidays, Worker's Compensation, medical plan, and life insurance. The following additional benefits were applied to the 1978 base rates: shift differential at 3.7% (4% for afternoon shift, 7% for night shift), overtime premium at 25% (time-and-three-quarters for Saturdays and Sundays), and dental plan at 1.3%. The total amount of benefits and burdens amounts to 54% of the 1978 base rates.

iii. Equipment operating cost

The equipment operating cost is a composite of the following items: overhaul labour, overhaul parts and overhaul major components, repair labour and repair parts, F.O.G. (fuel, oil, grease, belt wetting kerosene), and power. "Overhaul labour" and "overhaul parts" form a reserve fund for future major overhaul costs. "Overhaul major components" is a reserve fund for current replacement and repair costs such as tires (large trucks), undercarriages (tractors), buckets and ropes (draglines), and major components of bucket wheel excavators. Labour costs are included in overhaul labour. "Repair labour and repair parts" form a reserve fund for current running repair costs. The hourly or monthly rate of F.O.G. and power is meant to create a reserve fund for current fuel, oil, grease, and power expenses.

iv. Operating supplies and materials

Operating supplies and material include consumable materials such as:

- road building: culverts and pipes;
- road maintenance: sand, gravel, and calcium chloride;
- tailings dams: culverts, pipes, ties and welding supplies;
- equipment maintenance: mechanical tools, welding supplies, anti-freeze, bolts and plate.

v. Subcontracts

Subcontracts include rental and lease of buildings and equipment, and any work carried out by subcontractors. In this study, it was sufficient to provide a total sum for operating labour, equipment operating costs, and materials of the subcontract.

The rates of operating labour are based on those of G.C.O.S. and Syncrude Canada Ltd. The rates of overhaul labour and parts, overhaul major components, repair labour and parts, F.O.G., and power are based on the experience of Techman/RC. The F.O.G. rates are based on a price of $0.70/gallon for gasoline and $0.60/gallon for diesel fuel. Power cost is $0.015/ per kWh, with appropriate penalties for peak power demands of the large draglines.

Table 6.1-1

STAFF SALARY SCHEDULE

General Manager Mining	$ 55,000
Department Managers (Mine, Equipment Maint., Planning)	45,000
Superintendents	40,000
Senior Engineers	40,000
Senior Biologist (Specialist)	40,000
Engineering Supervisors	36,000
Dewatering Supervisor	36,000
Geological/Geotechnical Supervisors	36,000
Environmental Supervisor	36,000
Reclamation Supervisor	36,000
Specialist Engineers (for Maintenance Section)	33,000
Engineers (Cost Program, Mine, Reclamation)	30,000
Geologists	30,000
Biologists (Terrestrial, Aquatic)	30,000
Pedologist	30,000
Forest Ecologist	30,000
Shift General Foremen	30,000
Training Coordinator	28,000
Shift Foremen	25,000
Training Foremen	25,000
Scheduling Supervisors	25,000
Survey Supervisor	22,000
Drafting Supervisor	22,000
Schedulers	20,000
Instrumentmen	19,000
Draftsmen	19,000
Technologists (for Planning Section)	19,000
Technicians (for Maintenance Section)	19,000
Ore Samplers	19,000
Rodmen	16,000
Clerks (Typists, Stenos, Receptionists)	12,000

N.B. Payroll burdens of 18% to be applied.

Table 6.1-2

HOURLY LABOUR SCHEDULE

Group	Classification	Hourly Base Rate 1977	Hourly Base Rate 1978	Fringe Benefits	Shift Diff.	Overtime Premium	Total Burden	Hourly Labour Rate Calc.	Use
			6%	25.3%	3.7%	25%	54.0%		
I	Labourer	7.60	8.06	2.03	0.30	2.02	4.35	12.41	12.40
II	Apprentice Trade Helper Serviceman Equipment Operator*	8.64	9.16	2.32	0.34	2.29	4.95	14.11	14.10
III	Tradesman** Mine Control Operator Dragline Operator B.W.E. Operator Bridge Tender Hopper Operator Tripper Operator Stacker Operator Crane Operator Drill Operator Blaster	9.82	10.41	2.635	0.385	2.60	5.62	16.03	16.00
IV	Lead Hand	10.41	11.03	2.79	0.41	2.76	5.96	16.99	17.00

* Equipment Operator: operator of mining equipment and support equipment other than described in Group III.
** Tradesman: mechanic, welder, electrician, millwright.

c. Capital Costs

The initial capital costs are the purchase costs of the original units necessary for a specific mining, tailings disposal, or reclamation requirement.

The replacement capital costs are the purchase costs of the units replacing the original units as described above. The equipment life in years was determined by comparing the estimated operating hours per year versus the estimated life in hours of the particular unit. The estimated equipment life in hours is based on experience data of Techman/RC. Capital costs are determined by either introducing an initial capital cost and subsequent replacement costs, or by establishing a sinking fund. In the latter approach, the total capital cost is proportional to the total quantity of work carried out and the estimated average yearly depreciation of each item of machinery utilized, thus resulting in capital cost per cubic metre, per kilometre, per hectare, etc. as appropriate.

Quotations for mining and support equipment manufactured in North America were obtained by Techman Ltd.; quotations for the bucket wheel excavators and associated equipment were obtained by Rheinbraun - Consulting GmbH. Quotations include basic cost, option cost, freight cost, and erection cost.

Provincial Sales Tax is not applicable in the Province of Alberta. Federal Sales Tax was applied to such equipment items as pick-up trucks, crew cabs and vans. In this study, these vehicles have F.S.T. included in the purchasing price. Any applicable duty is also included in the price.

All quotations in German marks and American dollars were converted into Canadian dollars using commercial exchange rates as of July 1, 1978. Descriptions of mining equipment and support equipment, quotations, and quotation dates are listed in Table 6.1-3, Typical Equipment Costs.

Table 6.1-3

TYPICAL EQUIPMENT COSTS

	Description	Purchase Price	Quotation Date
Cat D6C	Tractor w/dozer and winch	$ 121,075	1/6/78
Cat D8K	Tractor w/dozer and winch	215,880	1/6/78
Cat D8K	Tractor w/dozer and ripper	219,950	1/6/78
Cat D9H	Tractor w/dozer and winch	284,270	1/6/78
Cat D9H	Tractor w/dozer and ripper	297,250	1/6/78
Cat 16 G	Grader w/ripper and front push plate	244,980	1/6/78
Cat 988	F.E.L. w/pallet fork	283,790	1/6/78
Cat 992 C	F.E.L. w/10 c.y rock bucket	604,685	1/6/78
Cat 245	Excavator (backhoe) w/3 c.y. bucket	398,400	1/6/78
Cat 594 H	Pipelayer	316,235	1/6/78
Cat 825 B	Compactor w/blade	219,990	1/6/78
Cat 777	End Dump Truck, 85 Ton	506,120	1/6/78
Cat 657 B	Push - Pull Scraper, 44 c.y.	530,455	2/6/78
Letourneau	L-800 F.E.L. w/15 c.y. rock bucket	585,825	5/6/78
Wabco 170 C	End Dump Truck, Haulpak 170 Ton	858,255	2/6/78
Ford	Pick-up 3/4 Ton 4 x 2	7,000	2/6/78
Ford	Pick-up 3/4 Ton 4 x 4	8,500	2/6/78
Vermeer	TS-44A Tree Spade w/serrated blades mounted on	12,000	Nov./78
Vermeer	M-485 Tractor w/front-end bucket	34,000	Nov./78
I.H. 986	Tractor w/three point hitch	30,000	Nov./78
Maletti	Rotary Tiller, 96" wide	4,150	Nov./78
M.F. 1805	Tractor w/three point hitch	40,000	Nov./78
SDP 18-24	Kello-Bilt Deep Plow, 30" wide	6,500	Nov./78

Table 6.1-3 (Continued)

TYPICAL EQUIPMENT COSTS

Description	Price*
Bucket Wheel Excavators	
For Overburden Only:	
1780 bank m^3/hr capacity	$ 18,055,000
2900 bank m^3/hr capacity	20,124,000
For Overburden and Oil Sand:	
2300 bank m^3/hr capacity	22,773,000
4000 bank m^3/hr capacity	35,423,000
4500 bank m^3/hr capacity	38,143,000
Draglines and Dragline Hopper	
75 m^3 Bucket Dragline	32,821,000
70 m^3 Bucket Dragline	32,070,000
1000 m^3 Dragline Hopper	10,006,000
Spreaders with Tripper Cars	
7,200 loose m^3/hr capacity	11,488,000
10,520 loose m^3/hr capacity	13,980,000
12,430 loose m^3/hr capacity	15,435,000
16,060 loose m^3/hr capacity	16,530,000

* Estimated capital costs in 1978 dollars, including freight and erection.

6.2 TYPICAL OIL SANDS MINE UNIT COSTS

In order to determine the net cost of reclamation as a function of major prime excavator employed, size of the mine and level of reclamation, costs must be determined for all the activities that will ultimately influence the cost of reclamation in an oil sands mine. Techman/RC have determined which activities are likely to influence reclamation costs, and have grouped these under six cost centres. Operating and capital costs were developed for each of these activities and are referred to as cost sub-centres.

Many of the costs developed remain constant regardless of prime excavators, mine size, or level of reclamation selected. Other costs vary, as different equipment is employed, leading, for example, to a reduction in unit operating cost because of economy of scale, or to an increase in unit operating cost because transport schedule requirements may result in excessive handling requirements over short peak periods. Such differences will be evident in the "comparative analysis" subsections of Chapters 7, 8 and 9 and sub-section 10.7.

The costs developed in this report consist of a summation of all operating and capital costs derived, as explained in the previous subsection 6.1. Operating costs are most often determined by multiplying calculated or estimated quantities by the appropriate unit operating costs and thus arriving at a total operating cost. For example, to obtain an operating cost for "Soil Spreading", applicable labour rates and equipment operating costs in units of dollars per cubic metre are multiplied by the volume (in cubic metres) of soil spread. The capital costs are sometimes determined using essentially the same approach. However, when the time and amount of a large capital purchase are known, this figure is introduced as a lump sum. Subsequent replacement costs are then also included as lump sums. Contract costs may be either calculated as a production price multiplied by quantity, or as an annual constant or variable lump sum. A brief explanation of each cost sub-centre as applied to all the mining options to be examined by this study follows in this subsection. Table 6.2-1, Listing of Developed Costs, specifies which cost sub-centres are costed, according to mining scheme, mine size, and level of reclamation.

COST CENTRE 1: Civil Construction-type Activities

Cost Sub-centre 1.1: Mine - Power Distribution and Control

The cost of power distribution is included, since a comparison of costs from mine to mine reveals added costs due to equipment used, size of mine, or level of reclamation. Power cables to the BWE's, draglines, spreaders, and conveyor drives are attached to the undercarriages of the conveyors. The construction of a supply line to the main substation is not included in this study. The power consumption is determined by the electrical requirements of the mining equipment, and is part of the operating costs of this equipment. The cost of the communication systems to control the operation of the mine is also included in this cost sub-centre.

Cost Sub-centre 1.2: Buildings

Differences in mining and reclamation methods will result in varying numbers of staff personnel required. The cost of providing office facilities per staff member decreases with a relative increase in mine size. Warehouse space may also show a decrease as lower unit inventories become acceptable with the increase in numbers of machines employed. Total maintenance floor area increases directly with increasing mine size, but varies only slightly between the dragline and bucket wheel excavator concepts. The variation is primarily determined by the size of the mobile equipment fleet. The increase or decrease in size of warehouse and maintenance buildings is assumed directly proportional to mine size and therefore is not costed. Only office space for staff personnel is costed in this sub-centre.

COST CENTRE 2: REMOVAL OF ORGANIC MATERIALS AND SOILS

Cost Sub-centre 2.1: Clearing

Brush and tree clearing, and burning or burial are necessary before muskeg removal or overburden removal begins. Clearing is also required on

Table 6.2-1
LISTING OF DEVELOPED COSTS

Mine Plan / Cost Centres	60,000 BPCD						120,000 BPCD						240,000 BPCD					
	Min.		Imp.		Enh.		Min.		Imp.		Enh.		Min.		Imp.		Enh.	
	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BW	D/L	BWE	D/L	BWE	D/L	BWE
Cost Centre 1: Civil Construction-Type Activities																		
1.1 Mine-Power Distribution & Control	X	X	X	X			X	X	X	X	X	X		X				X
1.2 Buildings	X	X	X	X			X	X	X	X	X	X		X				X
Cost Centre 2: Removal of Organic Materials & Soils																		
2.1 Clearing	X	X	X	X			X	X	X	X	X	X		X				X
2.2 Muskeg Dewatering	X	X	X	X			X	X	X	X	X	X		X				X
2.3 Muskeg Loading	X	X	X	X			X	X	X	X	X	X		X				X
2.4 Muskeg Hauling (incl. Road Maintenance)	X	X	X	X			X	X	X	X	X	X		X				X
2.5 Muskeg Placement	X	X	X	X			X	X	X	X	X	X		X				X
2.6 Muskeg Road Construction	X	X	X	X			X	X	X	X	X	X		X				X
Cost Centre 3: Overburden, Reject, Oil Sands Handling																		
3.1 Overburden BWE	X		X				X		X		X							
3.2 Oil Sands Draglines & Hoppers	X		X				X		X		X							
3.3 BWE (Overburden & Oil Sands)		X		X				X		X		X		X				X
3.4 Transport (All Conveyors)	X	X	X	X			X	X	X	X	X	X		X				X
3.5 Placement (Spreader)	X	X	X	X			X	X	X	X	X	X		X				X
3.6 Miscellaneous Equipment	X	X	X	X			X	X	X	X	X	X		X				X
Cost Centre 4: Tailings Disposal																		
4.1 Area Drainage	X	X	X	X			X	X	X	X	X	X		X				X
4.2 Clearing	X	X	X	X			X	X	X	X	X	X		X				X

Table 6.2-1
LISTING OF DEVELOPED COSTS

Cost Centres \ Mine Plan	60,000 BPCD						120,000 BPCD						240,000 BPCD					
	Min.		Imp.		Enh.		Min.		Imp.		Enh.		Min.		Imp.		Enh.	
	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BW	D/L	BWE	D/L	BWE	D/L	BWE
4.3 Construction of Starter Dams & Overburden Dams	X	X	X	X			X	X	X	X				X				
4.4 Piping of Wet Tailings or Conveying of Dry Tailings	X	X	X	X			X	X	X	X	X	X		X				X
4.5 Tailings Sand Placement into Dyke	X	X	X	X			X	X	X	X				X				
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	X	X	X	X			X	X	X	X	X	X		X				X
4.7 Recycling of Tailings Water	X	X	X	X			X	X	X	X				X				
4.8 Rehandling of Tailings Sludge	X	X					X							X				
4.9 Sludge Treatment			X	X					X	X								
4.10 Power Distribution	X	X	X	X			X	X	X	X				X				
4.11 Oversize Reject Disposal	X	X	X	X			X	X	X	X				X				
4.12 Oversize Reject Disposal Road Construction	X	X	X	X			X	X	X	X				X				
<u>Cost Centre 5</u>: Establishment of Ultimate Land Use Resources																		
5.1 Muskeg Rehandle Loading	X	X	X	X			X	X	X	X				X				
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	X	X	X	X			X	X	X	X				X				
5.3 Muskeg Rehandle Placement	X	X	X	X			X	X	X	X				X				
5.4 Muskeg Rehandle Road Construction	X	X	X	X			X	X	X	X				X				
5.5 Overburden Rehandle Loading	X	X	X	X			X	X	X	X				X				
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	X	X	X	X			X	X	X	X				X				
5.7 Overburden Rehandle Placement	X	X	X	X			X	X	X	X				X				

Table 6.2-1
LISTING OF DEVELOPED COSTS

Mine Plan / Cost Centres	60,000 BPCD						120,000 BPCD						240,000 BPCD					
	Min.		Imp.		Enh.		Min.		Imp.		Enh.		Min.		Imp.		Enh.	
	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BWE	D/L	BW	D/L	BWE	D/L	BWE	D/L	BWE
5.8 Overburden Rehandle Road Construction	X	X	X	X			X	X	X	X				X				
5.9 Muskeg Mining, Slurry Transport & Dewatering											X	X						X
5.10 Prepared Soil Manufacture	X	X	X	X			X	X	X	X	X	X		X				X
5.11 Prepared Soil Loading, F.E.L. & Trucks			X	X					X	X	X	X						X
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)			X	X					X	X	X	X						X
5.13 Prepared Soil Placement, Trucks			X	X					X	X	X	X						X
5.14 Prepared Soil Road Construction			X	X					X	X	X	X						X
5.15 Seed Bed Preparation, Maintenance	X	X	X	X			X	X	X	X	X	X		X				X
5.16 Specialized Land Use Requirements																		
Cost Centre 6: Supervision, Technical Services																		
6.1 Equipment Maintenance	X	X	X	X			X	X	X	X	X	X		X				X
6.2 Planning	X	X	X	X			X	X	X	X	X	X		X				X
6.3 Mining	X	X	X	X			X	X	X	X	X	X		X				X

areas without muskeg cover. D-9 dozers and operators, miscellaneous labour, and support vehicles such as personnel vehicles are included. Costs and revenues of timber harvesting and sale have not been estimated.

Cost Sub-centre 2.2: Muskeg Dewatering

Dewatering of muskeg for removal at the mine and plant site is costed here. Dewatering of muskeg at the out-of-pit tailings pond is costed elsewhere. Muskeg dewatering is achieved by establishing an array of drainage ditches. In continuous muskeg, an average ditch depth of 2 m and a ditch spacing of 100 metres is assumed. In discontinuous muskeg, an average ditch depth of 0.5 m and an average spacing of 30 m is assumed. Large backhoes or small draglines are used although ditches may be blasted in some situations.

Cost Sub-centre 2.3: Muskeg Loading

Digging and loading of muskeg at mine and plant site is costed here. In general, muskeg loading is done during the winter months using a large front-end loader assisted by a dozer. Whenever the materials handling schedule permits, the muskeg may be removed with a bucket wheel excavator as described in Subsection 5.4. The cost shown in the summary is based on the assumption that all muskeg is to be handled by F.E.L. and trucks.

Cost Sub-centre 2.4: Muskeg Hauling (incl. Road Maintenance)

Muskeg removed during the winter is hauled to reclamation or disposal sites using large off-highway end-dump mining trucks. Road maintenance is included on a per unit hauled basis. Whenever scheduling permits, muskeg removed during the summer by bucket wheel excavator is transported via the overburden belt conveyor system. The cost shown in the summary is based on the assumption that all muskeg is to be handled by front-end loader and trucks.

Cost Sub-centre 2.5: Muskeg Placement

Placement of muskeg into 4 m high muskeg dumps is costed here. It includes part of a truck cycle and dozers.

Cost Sub-centre 2.6: Muskeg Road Construction

Roads are required between the muskeg removal site and the final storage location or reclamation site in the case when direct muskeg transport is practical. The major trunk roads are constructed during the previous summer using overburden from the mine. An average road cross section of 15 m wide by 1 m deep is assumed for roads in muskeg removal areas, on tailings dykes, and on reclamation areas. Road construction materials are taken from borrow pits in the pit dumps or alongside the pits.

COST CENTRE 3: Overburden, Reject and Oil Sands Handling

The handling of major mass quantities such as overburden, top reject, centre reject, and oil sands is done with an integrated materials handling system consisting of machinery for excavation (draglines or bucket wheel excavators), machinery to transport (face conveyors, trunk conveyors, and transfer points), and machinery for placing materials (tripper cars and spreaders). For each material handled, various machinery is utilized, some for greater and some for lesser periods of time. Consequently, the handling cost of each type of material consists of a proportion of the capital costs and hourly operating costs of the components of the system being employed. The mine plan and manner of operation vary not only with the size of mine, but also according to the level of reclamation to be achieved. When the major materials handling costs have been determined, it is possible to estimate the costs attributable to reclamation demands by applying unit costs proportional to a planned materials handling scheme. Furthermore, it is possible to determine if the costs of reclamation activities vary with the scale of the oil sands mining operation.

Two basic equipment configurations are examined:

a. Three-bench mine utilizing both a bucket wheel excavator and draglines (referred to subsequently as the dragline mine plan):

Upper bench: BWE removes overburden, top reject, and occasionally pay zone
Middle bench: Dragline removes pay zone and centre reject
Lower bench: Dragline removes pay zone and centre reject

The middle and lower benches are approximately equal in height. In some cases, the height might be varied to allow the dragline on the middle bench to remove some top reject or even overburden in order to avoid under-utilization. Similarly, the overburden excavator may assist the draglines in meeting the designated crude oil production rate by removing some oil sands. The capacity of the two draglines is matched to the plant feed requirements; similarly, the BWE's capacity is matched to the overburden removal requirements.

b. Three-bench mine utilizing bucket wheel excavators only (referred to subsequently as BWE mine plan):

Upper bench: BWE removes overburden, top reject, and some pay zone
Middle bench: BWE removes pay zone, centre reject, and occasionally top reject or overburden
Lower bench: BWE removes pay zone and centre reject

In this case, the three excavators could have equal theoretical capacities and mine three benches of approximately equal height. Both the bench heights and the production required from each bench may be varied in order to achieve the designated crude bitumen production level. Alternately, the excavators may be sized as described for the dragline mine above where the upper bench BWE removes primarily overburden and top reject. Local troughs of oil sands in the pit floor that cannot be excavated by the BWE operating in the deep-cut mode would be excavated by a small dragline.

Defining the capital and operating cost of belt conveyors over the life of a mine is a rather complex problem. The capital and operating costs are dependent on many variables such as: belt width, belt length, amount and type of material moved, the height differential between head and tail pulleys, temperature, belt speed, requirements for belt wetting fluid, frequency of conveyor shifts and increases or reductions in length, etc. The mine conveyor system is composed of a number of belt conveyors of various lengths, belt widths and throughputs. For example in the 120,000 BPCD dragline scheme, there are 37 conveyors in year 23, but their length and number changes yearly.

Consequently, conveying costs for all materials are treated as a lump cost, and are derived by detailing costs at critical phases in the mine and extrapolating between these phases, keeping in mind the progressive change in conveyor layout during each year of operation.

An average system availability of 5000 hours was estimated. Further details on equipment capacities are tabulated in Subsection 5.5, and on system sizing for each mine plan in Chapter 7, 8 and 9. Further description of the defined cost sub-centres follows:

Cost Sub-centre 3.1: Overburden BWE

Overburden material is, in all cases, removed by a BWE. Clearing, muskeg dewatering, and muskeg removal will have occurred two or more years prior to overburden removal. (An exception may occur in areas where a muskeg layer has purposely been left behind to blend with underlying overburden; this cannot be planned without very detailed knowledge of field conditions and knowledge of actual progress of the excavators, and consequently, has not been deemed a standard operating alternative to be considered in the long term planning conducted in this study.)

Overburden and top reject drilling and blasting is not costed, since it does not vary significantly on a unit basis with size of the mine, or over the three suggested levels of reclamation. It is assumed to be either necessary or unnecessary, as the case may be, for both the dragline and the BWE mining schemes.

Included in the costs of overburden removal are the capital costs and the hourly operating costs of the BWE excavator. These costs vary at a given mine size, depending on whether the application is for a dragline mine or a bucket wheel mine, and also whether the BWE removes either overburden and top reject, or else overburden, top reject, and oil sands. The dragline mines usually employ a considerably smaller BWE. Overburden removal in the bucket wheel mining schemes is included in Cost Centre 3.3 together with other BWE's, since the type of material to be mined by the top BWE (i.e. usually includes some oil sands) is determined by the total mining depth. The capital and operating costs of the BWE are determined on the basis described in Section 6.1.

Cost Sub-centre 3.2: Oil Sands Draglines and Hoppers

Draglines are used in conjunction with suitable conveyor-loading hoppers. Hoppers are required in a multiple-bench dragline mine, since the utilization of an oil sands windrow to be rehandled by a bucket wheel reclaimer creates congestion and other operational difficulties in all but the most regularly-shaped ore bodies. The merits of such windrowing procedures do not appear decisive, since the same effect can be achieved by building extra capacity and flexibility into the dragline's conveyor system.

Since no dragline/hopper operation of the scale required by oil sands mines exists, the Consultants estimated costs of such a hopper based on preliminary in-house mechanical designs.

The dragline removes both oil sands and centre reject. In the case of the middle bench, the centre reject is cast into the hopper and transported to the disposal site via belt conveyors. Centre reject from the lower bench is backcast onto the pit floor except where dykes must be constructed.

It is assumed that, in the multiple bench mine, the blasting of oil sands will be standard procedure in the dragline and bucket wheel schemes. This is in contrast to a single bench dragline mining operation where the constraints imposed on the dragline by the high bench and the resultant slope stability problems completely rule out the blasting option.

The capital and operating costs of the draglines and hoppers are determined on the basis described in Section 6.1.

Cost Sub-centre 3.3: BWE (Overburden and Oil Sands)

Mine plans featuring only BWE's may employ BWE's which are of equal sizes, and are thus interchangeable between all benches; or which may be sized as in the case of the dragline plans described above. The choice depends on the relative advantage of a mine operating identical excavators, thus utilizing one conveyor size throughout, as compared to a plan having one smaller overburden BWE and two larger oil sands BWE's and thus at least two major conveyor sizes. The reach of the BWE (i.e., digging height) is also significant, since it has a direct bearing on the weight and cost of the excavator. The choice with respect to sizing requires rather detailed simulation and scheduling of the ore body under consideration. In this study the BWE's are identically sized for a particular mine size, but the scheduled production for each unit is varied to achieve a designated combined production target.

Cost Sub-centre 3.4: Transport (All Conveyors)

A detailed analysis was made of capital and operating costs of conveyors at critical points in the life of each mine. Costs between these critical points were estimated in terms of the annual materials transport requirements over the life of the mine. As was the case with the excavators, unit costs were developed which could be applied proportionally to the annual quantities of materials moved. Annual variation in capital cost (additions and replacements) were given consideration. Conveyor requirements are detailed in the sequential drawings included in Chapters 7, 8 and 9. An estimate of the relative unit cost of transporting overburden, top reject, centre reject, pay zone, and dry tailings can be obtained by comparing quantities of materials moved and the percentages of material moved over specified conveyor lengths at the critical years.

At the Enhanced Level of Reclamation, dry tailings sand is transported by conveyors. The costs of the conveyors and shunting heads located at the plant that are required to load dry tailings sand onto the two conveyors leading to the spreaders are not included, nor is the cost of an emergency tailings and dumping system that may be required in the event of the shut-down of both main conveyors. The associated reclaim system would likely consist of front-end loaders loading trucks or loading directly onto the conveyor. The estimating of the capacities and costs of such an emergency system is dependent on the surge capacity built into the extraction plant.

Cost Sub-centre 3.5: Placement (Spreader)

Overburden reject materials and dry tailings may be deposited within the mined-out pits or into waste dumps outside of the pits. In some BWE plans, most of the overburden, top reject, and centre reject are placed within the pits by a spreader, and later covered with tailings. In other plans, the overburden is initially placed outside of the pit as an outside dump. When additional room for in-pit tailings ponds is no longer needed, the remaining overburden is placed inside the mined-out pits.

In the dragline mine plans, centre reject from the lower bench is backcast into the mine, thus avoiding transfer via conveyor to the spreader. The remaining overburden waste is placed into outside dumps until further room for in-pit tailings ponds is not required.

At the Minimum and Improved Levels of Reclamation, the spreader can be used to selectively bury undesirable materials, to build embankments to serve as in-pit tailings dykes, and to temporarily stockpile overburden for use in the manufacturing of prepared soil. Where the spreader dumps overburden onto the pit floor prior to the construction of in-pit tailings ponds, the cost of transporting selected overburden from inside the pit to a temporary outside stockpile is included in this cost sub-centre. Costs for the spreader are determined in the same manner as for excavators.

At the Enhanced Level of Reclamation, the spreader will place the dry tailings sand as well as overburden materials.

It is likely that the costs of mine dewatering will be the greatest at the Enhanced Level since recharge into the high backfill embankments may cause instability at the toes of slopes. However, since the extent of this problem cannot be adequately described at this time, the additional dewatering costs have not been estimated. It should be noted that a somewhat similar recharge problem could develop whenever a sand dyke or overburden dyke is constructed, or overburden is backfilled into the pit.

Cost Sub-centre 3.6: Miscellaneous Equipment

In addition to the basic bucket wheel excavators, draglines, and conveyors, smaller mobile equipment is needed to assist in or perform specialized functions. Such equipment includes dozers, pipelayers for conveyor shifting, special crawler units for shifting drive stations, front-end loaders with specially-fitted forks for cleaning under conveyors, belt wetting equipment, etc. The number of units to be applied depends on the volume transported by conveyor, the frequency and rate of conveyor shifting, and the length of installed conveyor. The construction cost of in-pit service roadways and ramps is absorbed in this cost sub-centre as part of the operating cost of miscellaneous equipment.

COST CENTRE 4: Tailings Disposal

Cost Sub-centre 4.1: Area Drainage

Major ditching for lowering of the surface water table prior to constructing the starter dams is costed. All other types of surface water control are site-specific and unavoidable in the course of developing a mine. The cost of such surface water control is not included, since a comparison of these costs from mine to mine would not reveal added cost of reclamation due to equipment used, size of the mine or level of reclamation.

Cost Sub-Centre 4.2: Clearing

Brush and tree clearing, and burning or burial are necessary ahead of tailings pond dyke construction and the tailings pond operating level. Clearing is estimated on a per hectare basis.

Cost Sub-centre 4.3: Construction of Starter Dams and Overburden Dams

Starter dykes and overburden dams both consist of overburden material. Starter dykes are required for out-of-pit and in-pit dykes, and are constructed by conventional earth moving equipment. Overburden dams are in-pit structures built by spreaders, dozers, scrapers, and compactors. The overburden material from a spreader is rehandled by scrapers and compactors in lifts. In some cases overburden is trucked from a previously consructed outside waste dump. The overburden dams are sometimes constructed in lieu of the conventional tailings sand dykes. A typical composite cost, which includes the cost of excavation, drains, filters, placement, compaction, slope protection, etc., is used for this type of earth construction project.

Cost Sub-centre 4.4: Piping of Wet Tailings

The cost of pumping a tailings slurry varies according to quantity of tailings, length of line, and hydraulic head maintained in pumping to each disposal pond. Pipeline construction, rotation, and maintenance costs were determined as a cost per metre length per cubic metre of tailings transported. The capital and operating costs for the pumping and booster stations were determined for each tailings pond for all the mine plans. Slurry pipelines were sized to meet the various pumping requirements, and to minimize the use of booster pumps. The length of the tailings pipelines in use varies throughout the life of the operation, and can be determined from piping-layout sketches. In this way, unit costs were developed for each typical pumping situation throughout the life of the operation.

Cost Sub-centre 4.5: Tailings Sand Placement into Dyke

The placement of sand into dykes involves activities in which the sand is allowed to settle in the specially-constructed cells, while the sludge separates and flows into the pond. Tailings pipes are periodically relocated as cells are progressively constructed to raise the elevation of the dyke. Mobile equipment is used to relocate pipe, to construct cells, to compact the dyke, to haul in filter and drain materials and to construct operational roadways. The costs were estimated based on graduant sand filters, but the use of coke may be feasible in some situations. The cost sub-centre includes the cost of assembling and disassembling the pipelines.

Cost Sub-centre 4.6: Tailings Overboarding and Sanding-in

When tailings sand is not needed to construct sand dykes, or when a pond is to be "sanded-in" as part of the reclamation scheme, the slurry is spigotted or "overboarded" into the pond and allowed to form a sand beach. The sludge separates from the sand and moves into the liquid portion of the pond. Tailings pipes are strategically positioned to allow a controlled formation of beach behind the dyke. The cost sub-centre includes the relocating, assembling, and disassembling the pipelines.

Cost Sub-centre 4.7: Recycling of Tailings Water

This cost is determined solely because a comparison between the wet, deoperating costs for various situations encountered over the life of the mines. An additional barge and pipeline for sludge pond water recycle is also included (not applicable to Improved and Enhanced Level schemes).

Cost Sub-centre 4.8: Rehandling of Tailings Sludge

In some operating concepts the tailings sludge is pumped from one pond to another to reduce final pond surface area or to relocate the wet pond. The capital and operating costs of the pump barge and pipelines for the various operating situations were determined for each typical situation throughout the life of the mines.

Cost Sub-centre 4.9: Sludge Treatment

For the Improved Level of Reclamation, the sludge is removed from the operating pond and sent to a sludge treatment plant for recovery of bitumen and further dewatering of sludge, thus reducing the ultimate sludge volume. Barge-mounted pumps are used to recover sludge from the active tailings pond.

Cost Sub-centre 4.10: Power Distribution

Power lines must be constructed along roadways between the power source and the pump barges located in the tailings ponds. The cost of providing this utility will vary with the number of ponds employed and with the location of the ponds. In the dry tailings method this cost is entirely eliminated. Power consumption is costed as part of equipment operating costs as explained earlier.

Cost Sub-centre 4.11: Oversize Reject Disposal

Oversize reject is produced by plants using a wet-process technique. In future plants utilizing a dry-process technique, no separate reject stream will result or, at worst, very little since it will likely be possible to dispose of the reject on conveyors along with the dry tailings. When a plant produces wet tailings, the oversize reject is transported by off-highway truck to a disposal site. The size of the truck fleet was determined by yearly oversize reject quantities. The fleet may have to be increased if daily or hourly surges exceed its capacity.

Cost Sub-centre 4.12: Oversize Reject Disposal Road Construction

The haul distance to the reject pile remains constant when the reject is disposed of in an out-of-pit dump. If trucked into the pit as backfill, the distance varies with the progress of the mine.

COST CENTRE 5: Establishment of Ultimate Land Use Resources

Only the major activities related to reclamation have been estimated in this cost centre. Many minor activities could be identified but these are generally too site specific for use in a meaningful comparative cost analysis. Also, a distinction is made between land reclamation activities and pollution control activities. The operation of sedimentation ponds, temporary ditching, pond seepage recycle, pest control, wildlife protection, maintenance of service access for pollution control activities, monitoring for compliance with regulatory standards and many similar activities are not reclamation costs and therefore are not included in Cost Centre 5. Research costs may, in a sense, be classified as a reclamation cost but in this study are considered as an overhead item. The cost of long-term maintenance required beyond the five years allowed to prepared the mine for abandonment is also not included.

Cost Sub-centre 5.1: Muskeg Rehandle Loading

At the Minimum Level of Reclamation, a portion of the muskeg removed in the mine is ultimately used for reclamation. Such muskeg is stored in strategically located dumps and rehandled as required. The materials not required for reclamation are placed into permanent waste dumps along the perimeter of the mine. Some earth work (cut or fills) may be required to facilitate both the temporary and permanent storage procedure.

At the Improved Level of Reclamation, twice the quantity of muskeg is needed for reclamation. Layered dumps (one part muskeg to two parts overburden) are strategically located and constructed during the winter months. A dozer working on this dump helps to assure that the layering is maintained sufficiently thin. Some mixing is accomplished by blading the materials after dumping. Muskeg not required for reclamation is placed into permanent waste dumps along the perimeter of the mine.

Cost Sub-centre 5.2: Muskeg Rehandle Hauling (incl. Road Maintenance)

The large end-dump trucks transport muskeg from the muskeg storage areas to the reclamation sites. Road maintenance is included on a per cubic metre hauled basis.

Cost Sub-centre 5.3: Muskeg Rehandle Placement

After transport the muskeg is dumped and dozers are employed to spread the dumped material. The cost includes the dumping portion of truck haul cycles.

Cost Sub-centre 5.4: Muskeg Rehandle Road Construction

Off-highway-type roads are constructed between the muskeg storage areas and the reclamation sites. A branched network of roads allows for an even distribution of the muskeg on the reclamation sites and is schematically illustrated in Figure 6.2-1 and 6.2-2.

Cost Sub-centre 5.5: Overburden Rehandle Loading

Stockpiled overburden from an outside dump or a temporary out-of-pit stockpile is reloaded with a front-end loader into large end-dump mining trucks in the Minimum and Improved Levels of Reclamation. In the Enhanced Levels, this sub-centre shows the cost of front-end loaders loading overburden onto a conveyor belt after the overburden bucket wheel excavator has fiished its stripping operation.

Cost Sub-centre 5.6: Overburden Rehandle Hauling (incl. Road Maintenance)

The large end-dump trucks transport overburden from the overburden storage areas to the reclamation sites. Road maintenance is included on a per cubic metre hauled basis.

Cost Sub-centre 5.7: Overburden Rehandle Placement

After transport, the overburden is dumped and dozers are employed to spread the dumped material.

Cost Sub-centre 5.8: Overburden Rehandle Road Construction

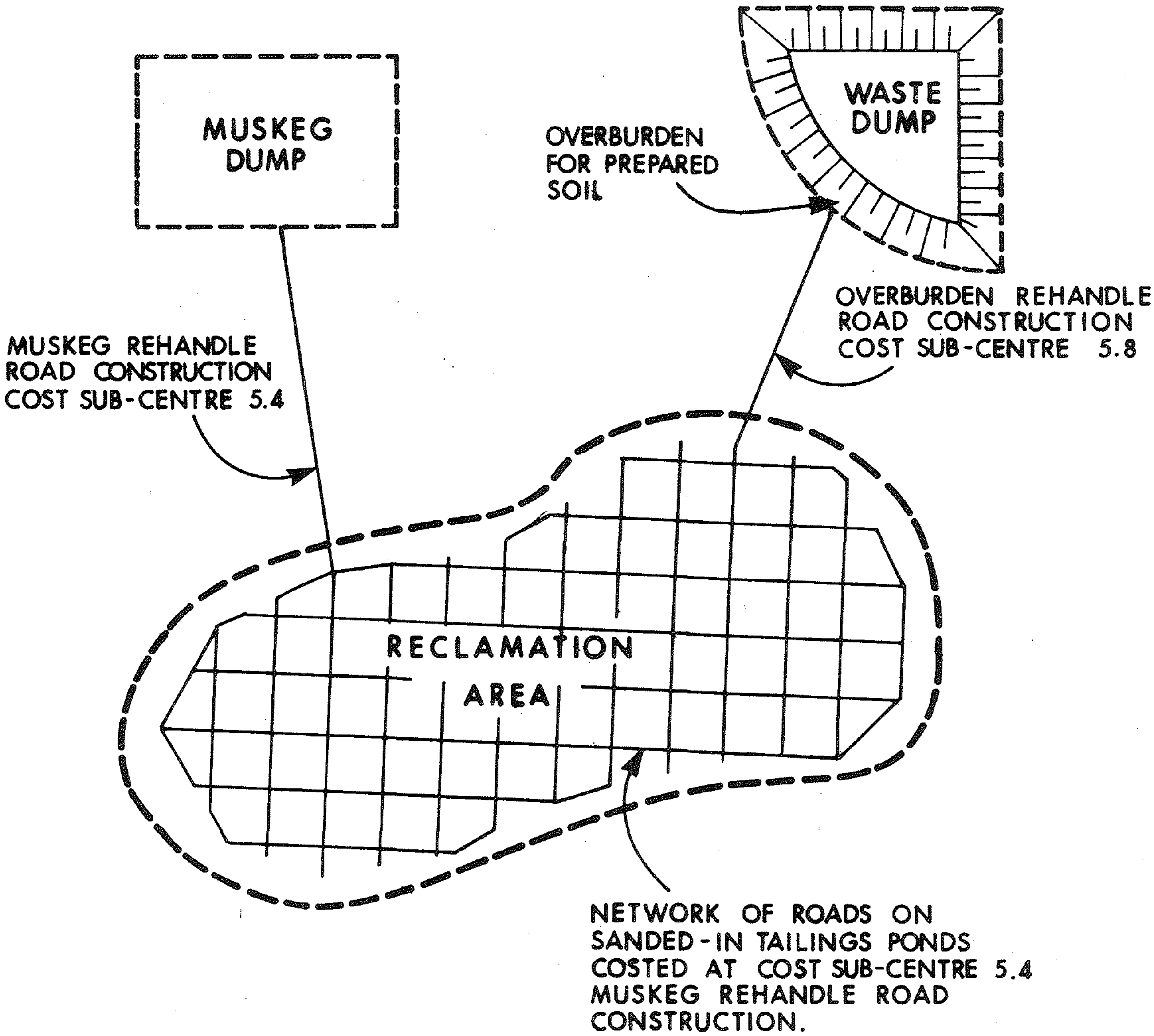
Roads are constructed between the overburden storage areas and the reclamation sites. A branched network of roads costed in sub-centre 5.4, also allows for an even distribution of the overburden on the reclamation sites. The system is illustrated schematically in Figure 6.2-1 and 6.2-2.

Cost Sub-centre 5.9: Muskeg Mining, Slurry Transport, and Dewatering

At the Enhanced Level of Reclamation, muskeg is mined hydraulically and transported as a slurry to a centrally-located dewatering plant. Dewatered muskeg is fed onto a feed belt supplying both muskeg and overburden to a stacker used to form a layered stockpile of materials. Muskeg mining will occur from May to October. This cost sub-centre includes delivery of dewatered muskeg onto the stacker feed conveyor.

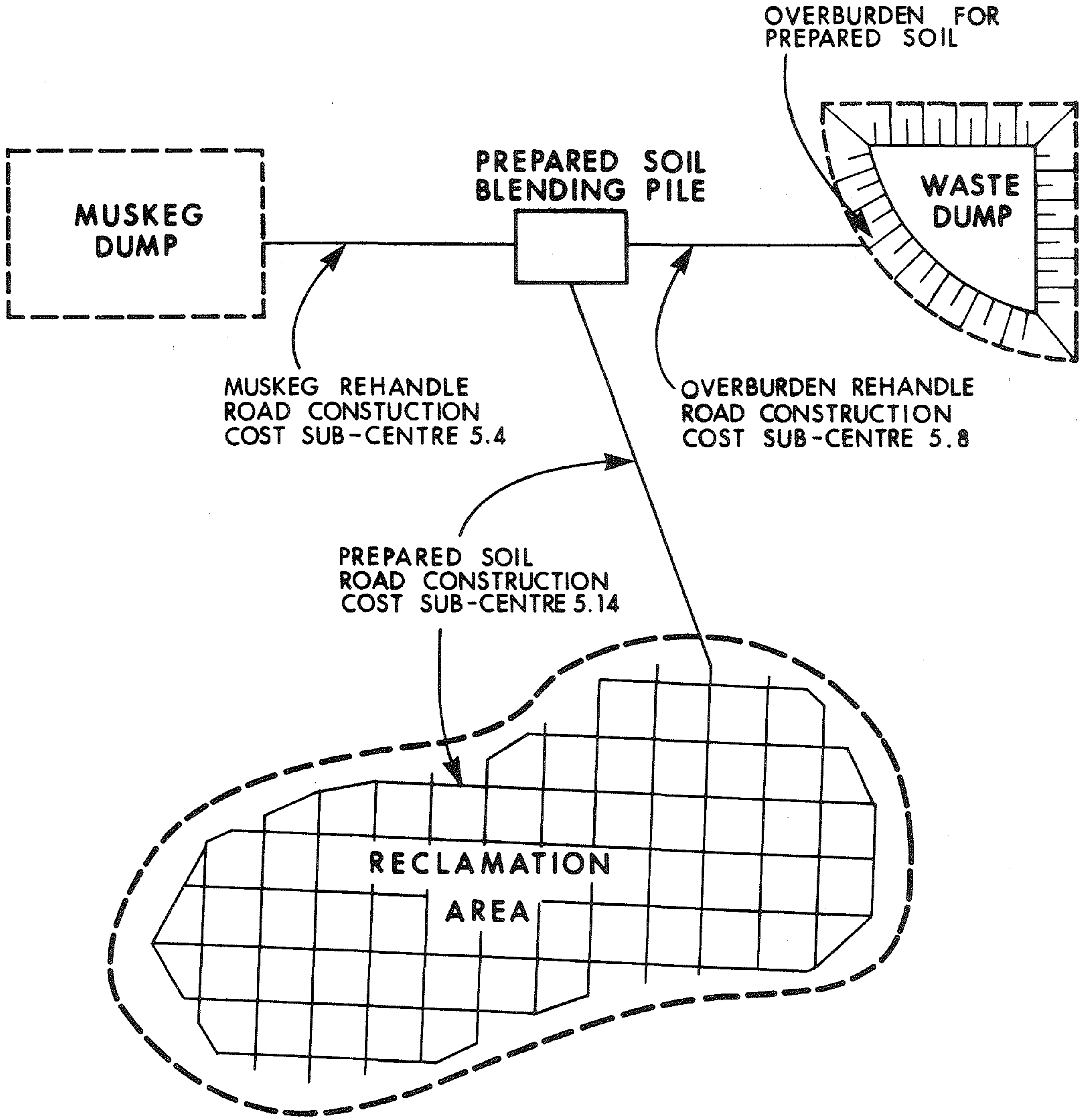
Cost Sub-centre 5.10: Prepared Soil Manufacture

The detailed procedures for obtaining prepared soil are described in Section 5.4. At the Minimum Level of Reclamation, muskeg and suitable overburden are roughly spread by dozer and then mixed with tailings sand by deep plowing to form a prepared soil layer of 0.6 m. At the Improved Level, the cost of manufacturing a prepared soil is accounted for by the



ROAD CONSTRUCTION COSTS ALLOCATION FOR MINIMUM LEVEL OF RECLAMATION

FIGURE 6.2-1



ROAD CONSTRUCTION COSTS ALLOCATION FOR IMPROVED LEVEL OF RECLAMATION

FIGURE 6.2-2

D10 dozer used to blade the layered muskeg and overburden from the layered stockpile. At the Enhanced Level prepared soil is manufactured as a bucket wheel reclaimer operates at the prepared soil (layered) stockpile. The entire cost of the stacker-bucket wheel reclaimer-conveyor operation is included. The cost of a conveyor delivering prepared soil to the reclamation site, and a small stacker stockpiling it, are also included in this sub-centre.

Field costs related to prepared soil manufacture are not encountered for either the Improved or the Enhanced Levels of Reclamation. Spreading in the field at all levels of reclamation is included under placement.

Cost Sub-centre 5.11: Prepared Soil Loading, F.E.L. and Trucks

Prepared soil is loaded only at the Improved and the Enhanced Levels of Reclamation. This is done by front-end loader in both cases, but the sources are different. At the Improved Level, a D10-dozer prepared mixture of muskeg and overburden are loaded, while at the Enhanced Level, prepared soil stacked at the terminus of the prepared soil conveyor is loaded.

Cost Sub-centre 5.12: Prepared Soil Transport, Trucks

Prepared soil is transported by off-highway trucks to reclamation sites in the Improved and Enhanced Levels of Reclamation.

Cost Sub-centre 5.13: Prepared Soil Placement, Trucks

At the Improved and Enhanced Levels of Reclamation, this sub-centre includes the dumping of the soil by off-highway trucks, and short distance relocation and spreading of dumped loads by dozers or scrapers.

Cost Sub-centre 5.14: Prepared Soil Road Construction

At the Minimum Level of Reclamation, muskeg and overburden, the components of prepared soil, are trucked along specially-constructed roads from the stockpiles of muskeg and the stockpiles of overburden to their final position on the reclamation site as shown schematically in Figure 6.2-1. These roads are costed in sub-centre 5.4 and 5.8. No transportation of finished prepared soil takes place in the Minimum Level. Some areas are reached by travelling on frozen ground or sand. At the Improved Level of Reclamation, a network of roads is required between the layered stockpiles and the reclamation sites (see Figure 6.2-2). At the Enhanced Level, a road distribution network exists to transport in-field stockpiled prepared soil from the terminus of the conveyor to the final reclamation sites.

Cost Sub-centre 5.15: Seed Bed Preparation, Planting and Maintenance

Rototilling of the prepared soil surface prior to planting is required at all Levels of Reclamation. The blending of overburden, muskeg and sand is part of prepared soil manufacture and is included in Cost Sub-Centre 5.10. Seeding and fertilizing are carried out by helicopter, preferably in the spring.

During the first year of reclamation, the seeding is followed up by another fertilization (also by helicopter) prior to tilling during the summer or after tilling in the fall. During the following years, two maintenance fertilizations are carried out annually. Since it is not known when fertilization can be stopped (in 10, 15, 25 or more years), the costs for fertilization are included as additional costs which are accumulated for each hectare to be reclaimed. Maintenance fertilization is costed only to Year 30, but must be continued for all, or at least part of the areas reclaimed. If all areas must continue to be fertilized for a period extending beyond the years since the first hectare is reclaimed, fertilization would continue at the cost incurred in Year 30, as shown on the computer print-out.

An allowance has been made for maintenance work required to repair damaged areas on a yearly basis. The cost includes transplanting of native shrubs and trees, as well as minor planting of seedlings on slopes to promote the stability of these slopes and serve as shelter belts. This basic type of revegetation is costed for all Levels of Reclamation.

Cost Sub-centre 5.16: Specialized Land Use Requirements

A variety of specialized land uses are possible for each mine plan. The amount of specialization determined on a per hectare basis or as a lump sum will vary and be greatly influenced by local differences existing with each mine or mine plan, and anticipated mine use. For example, the ratio of revegetation for erosion control, compared to noncommercial and to commercial forest is not determinable at the level of mine planning conducted in this study. Consequently, only typical per hectare costs representative of such land uses are suggested. Once an estimate of the relative ratios of land uses is made, the additional per barrel costs can be calculated by multiplying hectares by unit costs of each type of land use, and dividing the summation of costs by the total barrels for the given mine plan. At the Improved and Enhanced Levels of Reclamation, the reclamation consists primarily of planting trees and shrubs, necessitating the construction of greenhouses and the operation of a tree nursery. The unit cost will vary with the species selected. An estimate of the cost can be obtained by multiplying the area reclaimed each year by the cost per hectare of planting and maintaining the selected species. Typical costs for commercial and non-commercial forest planting are $4,000 and $3,000 per hectare respectively. This includes the costs of plants, labour, equipment and specialized sowing (on strips) for erosion control.

Many other specialized land uses are possible but none of these are costed. The list of possibilities includes the development of recreation area for camping, fishing, hunting, hiking, boating, etc. Although the filling of the mined-out end pit is assumed in all mine plans, the cost of pumping water into the lake as well as grading of beach areas is not included in any cost centre.

COST CENTRE 6: Supervision and Technical Services

Cost Sub-centre 6.1: Equipment Maintenance

An average rate of remuneration of staff was determined according to mining method, level of reclamation and mine size. Variations in the staff work force, primarily foremen, were estimated. With some adjustment, the same organizational chart (Drawing No. BR22900-18-00) can be applied to all options considered in this study. Hourly maintenance labour costs are included in the equipment unit operating costs, and consequently, no manning charts of hourly personnel were used in developing maintenance costs.

Cost Sub-centre 6.2: Planning

"Planning operations" is considered to be the least variable of the three major operations considered. Nonetheless, the total number of staff required decreases at the 60,000 BPCD production rate, and increases at the 240,000 BPCD production rate, relative to those of the 120,000 BPCD rate. No variation in staff with respect to level of reclamation can be justified. Average rates of remuneration and an estimate of the staff labour force were used to determine costs. The organizational chart (Drawing No. BR22900-19-00) remains almost identical for all options considered by this study.

Cost Sub-centre 6.3: Mining

The "mine operations" chart (Drawing No. BR22900-17-00) depicts both the supervisory staff and hourly labour force required. Again, the same is applicable for all three mine sizes, although the number of on-site individuals required varies. Average rates of remuneration and an estimate of the staff labour force were used to determine costs. Foremen are considered as salaried personnel. The hourly labour costs are part of the unit operating costs of other cost sub-centres already described in this section. Consequently, no manning charts of hourly personnel were developed.

7.0 CONCEPTS AND COSTS OF DEVELOPMENT AND RECLAMATION OF 120,000 BPCD OIL SANDS MINE

7.1 OVERVIEW OF DEVELOPMENT PLANS

Six detailed mine plans have been developed for Ore Body No. 2: a dragline mine scheme and a bucket wheel excavator mine scheme for each of the three levels of reclamation. This chapter must be read in conjunction with those 120,000 BPCD mine plans which are presented in Volume II.

The efficiency of oil sands removal compared to that of tailings disposal is the major factor influencing the layout of the mine. The relationship between oil sands transport costs and tailings disposal costs therefore has received special consideration in this study.

A governing objective for tailings disposal is that the areal extent of out-of-pit disturbances such as the out-of-pit tailings ponds and waste dumps should be minimized. The proximity of neighbouring ore bodies to the southwest necessitates that the outside tailings pond be situated north of these ore bodies but not so far north that the pond would begin to interfere with ore bodies located northwest of Ore Body No. 2 (see chapter 2.0 and Drawing No. F 22910-02-00, Ore Bodies No. 1 and 2 Under Changing Economic Conditions). The presence of a large area of continuous muskeg northwest of the ore body also influences siting of the pond.

A further consideration has been given to the time-value of oil sands resources. Ideally, mining should begin in an area where the oil saturation percentages drop rapidly along the boundary of the ore body. A more efficient utilization of a finite oil sands reserve is obtained by scheduling the areas with gradually dropping saturation percentages for mining some time in the future, when currently uneconomic ore may become economically mineable. Adjoining the northern boundary of Ore Body No. 2 are large areas which fall into this category. The greater the average depth of mined-out pit the more desirable the pit becomes for in-pit tailings disposal (especially the disposal of sludge) because the surface area of the pond is minimized. Accordingly, Ore Body No. 2 is opened at the south end and developed in a northwardly-advancing direction. The positioning of in-pit dykes is determined both by the shape of the mine and the mining schedule. It is advantageous to locate dykes in the "necks" of an ore body. The number, as well as the spacing, of a series of in-pit dykes is also critical. Each time a dyke position and pond elevation are chosen, the mining and tailings mass balance must be examined for the entire life of the mine to determine if the new scheme is feasible. Small changes in the position of dykes, size of pond, and the ratio of in-pit storage to out-of-pit storage may have either detrimental or beneficial effects on the entire tailings disposal scheme.

Tailings disposal must also be integrated with the rate of advance of the mine. This is particularly important when the development plan features a series of wet ponds. Oil sands removal from the bottom bench must be advanced beyond the point where the next dyke is to be located, and in the meantime, the newly-created void cannot be used for storage of tailings. The tailings generated from the oil sands mined in the current pit must be distributed amongst the previous ponds or contained entirely by the previous tailings pond. Ore Body No. 2 consists of a series of four relatively small pits, of which three at most may be used for disposal of wet tailings. It is not practical to further subdivide these pits into intermediate tailings ponds. This constraint is equally applicable if the plan involves an ore body group consisting of four completely unconnected ore bodies.

A three bench mining system imposes more severe time constraints on the tailings disposal scheme than does a single bench mine. Since an in-pit tailings disposal scheduling is governed by the advance of the bottom bench, a three bench mine must store an extra volume of tailings in the out-of-pit tailings pond. This extra volume of tailings is produced from oil sands mined at the upper two benches, which are a certain distance ahead of the bottom bench.

The dragline development scheme at the Minimum Level is based on a "Concept 5" tailings disposal scheme. A "Concept 2" tailings disposal scheme is used in the bucket wheel plans. The general characteristics of these concepts are described in Subsection 5.3.2., Tailings Disposal at the Minimum Level of Reclamation.

At the Minimum Level of Reclamation, the dragline plans include an out-of-pit tailings pond that is operated in a conventional manner for nine years. Tailings are then diverted to the first mined-out pit for the purpose of constructing the first in-pit dyke. In the tenth year, sludge pumping from the out-of-pit pond to the in-pit sludge pond is started. The out-of- pit pond is sanded-in progressively. In the meantime, the first mined-out area for tailings disposal becomes available (Pit 2). This pond is also progressively sanded-in and the sludge pumped to the sludge pond. Before the first in-pit tailings pond (Pit 2) is completely full, another mined-out area is available for tailings disposal, and so the second in-pit tailings pond may be started (Pit 3). As has already been mentioned, the availability of suitable in-pit ponds is critically hinged to the time at which oil sands removal from the bottom bench has advanced beyond sites suitable for the location of dykes.

The bucket wheel excavator mine at the Minimum Level requires nearly the same size of out-of-pit tailings pond, but differs substantially in-pit. Because of the slewing style of development, only two in-pit ponds are possible. The location of the first in-pit tailings pond coincides with the location of the sludge pond in the dragline plan. The lower portions of both in-pit dykes are constructed with overburden, and the upper portions with tailings sand. With a slewing development plan, the oil sands mining machinery is spread over a much larger area of the mine at any given time; thus the opportunity for the development of in-pit ponds is greatly reduced. This is a drawback of the slewing system that occurs when the ore body consists of a series of connected pits. Such disadvantages must be weighed against the flexibility afforded the mining operation. Fewer conveyors and less reconstruction of conveyor systems are possible.

In the bucket wheel plan, almost all the overburden and centre reject are returned to the pit prior to the construction of in-pit ponds. This has the effect of elevating the surface of the pond and may, under certain hydrological conditions, be very advantageous. Compared to the dragline plan where little or no backfill is placed under the ponds, considerably more dyke must be built around the perimeter of the mined-out pits. The more extensive dyke construction is avoided in the dragline plans by designing much larger out-of-pit dumps, and when further in-pit tailings storage capacity is not needed, overburden is diverted into the mined-out pit. Once completely developed, the bucket wheel mine has three large wet tailings pond surfaces, compared to only one wet sludge pond surface in the dragline plan. This is largely by choice. It does not represent an advantage or disadvantage to either mining method, but rather, illustrates the advantages and disadvantages of two different tailings disposal concepts (see Subsection 5.3, Tailings Disposal Techniques).

At the Improved Level, the tailings disposal operation includes a sludge dewatering scheme. However, the schedule and rate of dewatering are substantially different in the dragline and the bucket wheel plans. The most significant difference is that the sludge pond of the dragline scheme is operational earlier in the life of the mine. In the dragline mine, a sludge dewatering period is planned, beginning in the sixth year of operation and continuing to just after shutdown of the mine. In contrast, the sludge dewatering rate in the bucket wheel mine is at a higher rate but the period is shorter, beginning in the seventeenth year.

In the Improved Level dragline plan, the first in-pit pond is formed by constructing a long earth-filled dyke across the widest portion of the first mined-out pit. This is certainly a disadvantage for sludge disposal, and illustrates the impact of mine configuration on the disposal of extraction plant tailings. A dyke constructed of tailings sand at this position is impossible, as water from the tailings slurry used for dyke construction would dilute dewatered sludge and defeat the very purpose of dewatering the sludge. The second in-pit dyke is located in the second neck of the ore body. No other in-pit tailings ponds are required. The dyke is constructed from overburden, with only a small sand dyke at the toe of the dyke to provide time for the backfill system to be started in-pit and supply material for the overburden portion of the dyke. Pit 2 and half of Pit 1 are sanded-in full. The third pit is backfilled with overburden. The surface elevation of the in-pit pond is somewhat higher than that used at the Minimum Level.

In the Improved Level dragline plan, the backfilling of the pit with overburden can begin much sooner than at the Minimum Level. This results in considerably less out-of-pit overburden dump construction. Nonetheless, the final void remaining is still larger than that occurring in the dragline plan at the Minimum Level of Reclamation.

Major differences between the bucket wheel plan at the Improved Level, as compared to that at the Minimum Level, also exist. The primary reason is that, in essence, a completely new tailings disposal scheme is employed: a modified "Concept 4" tailings disposal scheme. Rather than pumping all the tailings sludge from all the tailings ponds into the final mined-out pit, the sludge is thickened and pumped into a pit formed just prior to the final pit.

The out-of-pit tailings pond remains in shape as at the Minimum Level. The first in-pit tailings pond is formed in the same location as the first in-pit pond at the Minimum Level, but with one major difference: all sludge is removed to the sludge pond (Pond B) and the void is gradually filled with tailings sand. The sludge pond is formed by constructing an overburden dyke at the second neck in the ore body and at the north and west edges of the second pit. Sludge from the out-of-pit tailings pond and from the in-pit tailings pond are pumped into the in-pit sludge pond. The bowl-shaped tailings pond is reclaimed, both inside and outside.

The final pit remaining in all schemes is to be developed as a lake. Should a multiple-mine development plan be possible, the remaining empty pits could be used for the tailings ponds, sludge ponds, or make-up water reservoirs for the adjoining mines.

The out-of-pit tailings pond is rectangular or somewhat oval in shape. A sand dyke is required around the entire perimeter of the pond, and in this respect the pond differs from the out-of-pit ponds proposed for Ore Bodies No. 1 and No. 4.

The dragline and bucket wheel mine plans for the Enhanced Level of Reclamation are very similar to each other. Dry tailings sand, overburden, centre reject, and top reject are backfilled into the mined-out area of the pit by means of two spreaders. Overburden and oil sands removal remain as in the previous plans.

All the backfilled materials, including tailings sand, are conveyed into the empty pits. The backfilling operation is started as soon as sufficient room exists in the pit to lay out a suitable arrangement of conveyors and spreaders. Prior to this time material is placed into outside dumps. The backfilling of the mine in both the dragline and bucket wheel plans occurs as close to the toe of the lowest oil sands mining bench as is operationally allowable. The lake is created in the final mined-out void. Three choices for backfilling exist when the creation of a water body is not a desirable form of land use:

1. The out-of-pit overburden dump created at the start of mining could be removed and backfilled at a unit cost similar to the previous overburden removal;
2. A portion of the backfilled pit might be redistributed, lowering the backfill elevation in a portion of the mine; or
3. The final pit could be filled with material from a neighbouring mine.

All the dragline mine plans utilize a parallel benching system, in which the working face of the overburden bench and both oil sands benches are parallel to each other, and are always directed from west to east. Since mining progresses from south to north, any oil sands to be mined north of any pit boundary requires a box cut. The ore body discussed in this section requires about 10 km of box cuts. The face conveyors must be shifted after every two dragline passes are completed (approximately each 60 m). In the bucket wheel excavator scheme, face conveyors require somewhat less frequent shifting (approximately every 80 m), taking advantage of the conveyor bridge. Over a period of time the absolute distance between the excavators on each bench increases or decreases, depending primarily on the relative depth of material on each bench. The overall rate of advance is regulated by the combined production of plant feed oil sands from the two lower benches. The overburden BWE is not scheduled to remove oil sands on a regular basis.

The bucket wheel mine plans utilize a slewing benching system. In Ore Body No. 2, this system is considerably more efficient than the parallel system. In total, fewer conveyors are required, and less reconstruction of conveyor transfer and distribution points occurs. The total mining depth is divided into three benches of approximately equal height, with one BWE on each bench. This means that the top BWE removes overburden and mines some oil sands for most of the time. The middle bench and bottom bench BWE's mine oil sand and centre reject. It is possible that, in certain parts of the mine, the middle bench BWE may dig some overburden where the overburden is deeper than one-third of the total mining depth.

The short radius of the slewing system in Ore Body No. 2 is not amenable to dragline mining because the pass width of a dragline can only be varied to a limited extent; thus, the repeated operation of a dragline at the apex of the slewing system tends to make the overall scheme rather inefficient.

In a slewing system the benches are rotated around pivot points. Wider passes and more frequent cuts at the face are made by the bucket wheel excavator at the extreme ends of the benches. Thinner and fewer cuts are made near the pivot points. The benches swing around the pivot points; hence the term "slewing". Once the conveyor on the uppermost bench (overburden bench) has swung to the boundary of the mine, the conveyor is kinked at a suitable point along the belt to form a new centre of rotation. When the next lower bench is advanced to a position where it is parallel to the bench above, it too is kinked. The same procedure is followed with the lowest bench. Ultimately an array of pie-shaped areas with varying radii are combined to form a sequence of pits.

The slewing technique is well illustrated in the 120,000 BPCD bucket wheel mine plans. Similar techniques are used for the overburden backfilling operation at the Minimum and Improved Levels, and for backfilling the overburden and dry tailings at the Enhanced Level.

Drawings showing mine layouts have been made at points in the development of the mine where the method of operation remains similar for some period of time. Other drawings, such as those depicting overall material distribution and reclamation, are applicable over the entire life of the mine. The time intervals between the stages selected for drawings have been studied to detect changes in quantities and costs. The arrangement of conveyors can be seen on each of the staged development drawings. The face conveyors (those seen crossing the ore body) also indicate the position of the working face on each bench.

The major operational differences between the six mine plans can be seen on the six sets of drawings provided in Volume II. Mass balance schedules are provided for overall mining, tailings disposal, and reclamation in Subsections 7.2 and 7.3. Schedules for various other items are provided in the computer-printed cost summaries. Cost estimates for selected operating activities for a period of 35 years are provided and include five years of preproduction and five years for deactivation. Summary cost comparisons are made in Subsection 7.4

The drawings indicate somewhat differing overall shapes for the dragline and bucket wheel mines. This is to be expected, because the dragline mine uses a parallel mode of operation, whereas the bucket wheel mine uses a slewing type of operation. Another cause for some of the difference in the plans (for the 120,000 BPCD operations only) is that the mass balances in the dragline mine are based on somewhat updated data (the result of progressive development of mine simulation computer programs). This does not introduce significant errors when comparing the costs of component activities if they are compared on a per unit basis.

7.2 MINE PLANS EMPLOYING DRAGLINES

The three dragline mine plans have been developed using identical mining schedules. This means that the rate of removal of muskeg, overburden, top reject, and oil sands is identical for a mine operated at the Minimum, Improved, and Enhanced Levels of Reclamation. In fact, the positions of the major excavating machinery are identical in all three cases. If oil sands production were to vary, the relative positions of the excavators would change.

The face conveyors run in an east-west direction. At transfer points the excavated materials are dumped from the face conveyors onto trunk conveyors. The trunk conveyors lead to the conveyor distribution point. From here, overburden, top reject, and centre reject are directed either to outside waste dumps or into the mine as backfill. Plant feed grade oil sands are directed to the extraction plant.

The direction that the material is moved is indicated by an arrowhead symbol. A solid arrow implies that only oil sands are moved. An arrow that is half-solid implies that overburden and reject as well as oil sands are transported. Clear arrows indicate that only overburden and reject are transported on the conveyor. When a change in conveyor direction occurs, the conveyor must be terminated and another conveyor started. These transfer points are shown as clear dots.

Two face conveyors are required for each bench. One runs the entire length of the face while the other runs only halfway. Two draglines per bench are required but each machine works only half the bench length. Each face conveyor leads into the distribution point, and two plant feed conveyors move oil sands from the conveyor distribution point to the extraction plant. The use of two conveyors improves the overall availability of the system as compared to that of a system using only one.

Each dragline loads oil sands into a large hopper positioned close to the face conveyor. The hopper, in turn, feeds onto the bench conveyor via a much smaller hopper located on the face conveyor. The hopper and dragline move in unison alongside the conveyor. Whenever possible, centre reject from the lowest bench is cast onto the pit floor rather than removed by conveyor. The dragline on the second bench must be operated in such a way that the number of interruptions in plant feed stream is minimized. This is achieved by exposing as much centre reject surface as possible before switching from oil sands excavating to reject excavating.

Overburden is dug on the upper bench using a BWE. A single face conveyor is required. If necessary, oil sands may also be excavated by this BWE when one of the draglines is inoperative for an extended period of time, or when the extraction plant is in peak operating condition, allowing a sustained period of above-average production.

At the Minimum and Improved Levels one overburden conveyor leads from the distribution point. This conveyor is sized to transport overburden excavated by the BWE as well as the reject excavated by the draglines. Overburden for prepared soil manufacture is removed from the waste dump or backfill by trucks or scrapers. At the Enhanced Level two conveyors transport overburden, reject and dry tailings to two spreaders. Two dry tailings conveyors equipped with shunting heads feed onto the two dumping side conveyors. Overburden for prepared soil manufacture is handled by a short conveyor leading from the distribution point to the prepared soil blending yard. When the distribution point is too far from the prepared soil blending yard, one of the plant feed conveyors is used for about an hour per day (during summer) to transport the overburden required for prepared soil.

Since each dragline must have its own conveyor to allow it to selectively mine oil sands and centre reject independently of the other dragline on the same bench, a duplication of the conveyor systems between the draglines and the distribution point results. A considerably larger total length of the mine conveyor system is required for the two draglines per bench than would be required if only one prime excavator per bench were used. For example, compare this to the Ore Body 4 design (60,000 BPCD), which employs one dragline per bench. In the early years, additional conveyors are required because the trunk conveyors must be back-tracked to the conveyor distribution point. In due course, the mining faces advance to a position north of the plant site, and at this time the backtracking is eliminated.

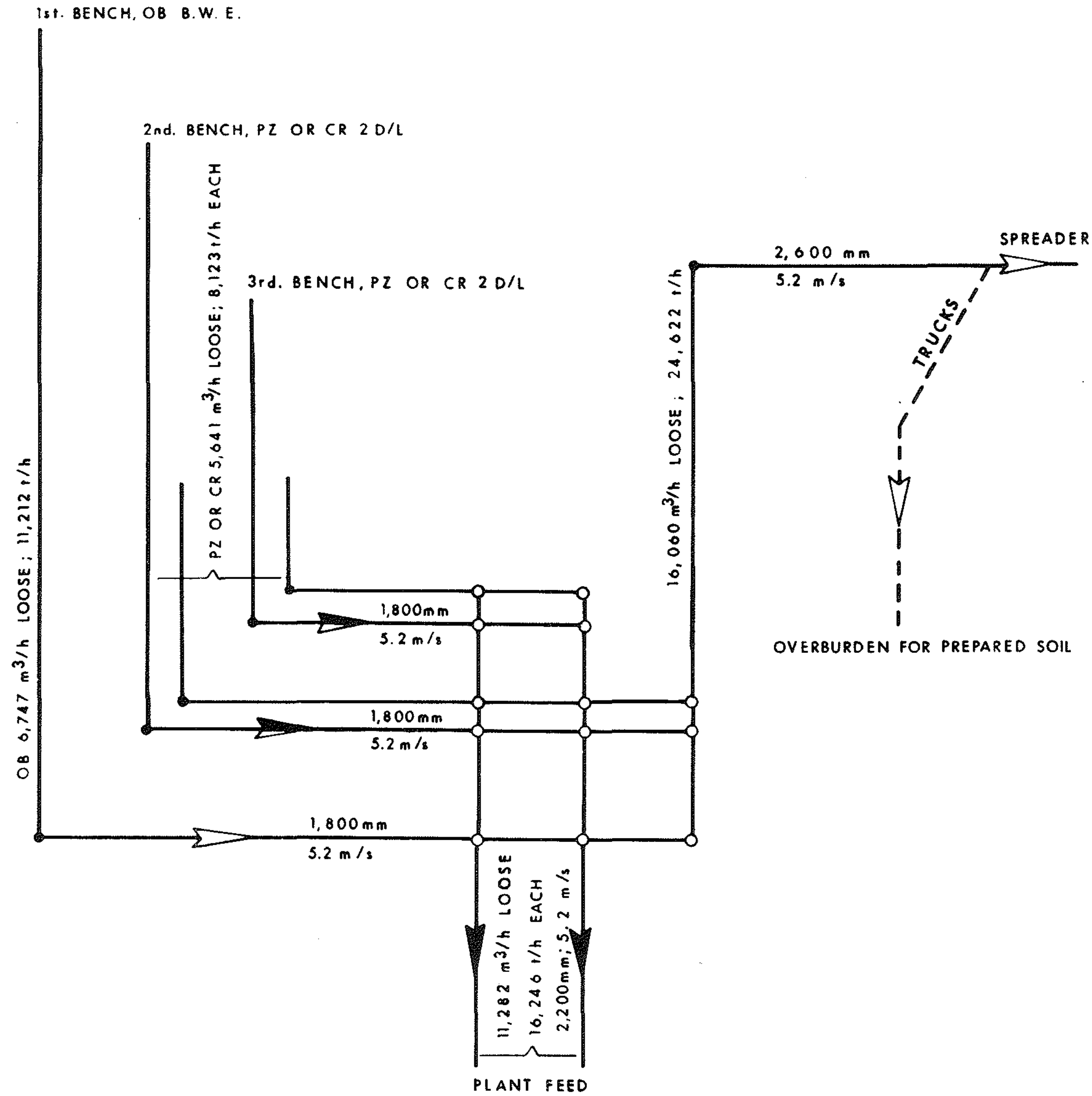
The parallel system minimizes the total area that is in a state of active mining at any given time. The two oil sands faces can be much closer together, thus allowing the construction of in-pit dykes to occur sooner than is possible with a slewing system.

The production schedules for the three mines are identical. Annual quantities for overburden, reject, oil sands, bitumen, and crude oil are provided in Table 7.2-1. The sizing of the materials handling system is schematically illustrated in Figures 7.2-1 and 7.2-2. The production of the BWE, draglines, spreaders, and belts is based on 5,000 operating hours annually, out of a total of 8,760 calendar hours. Details regarding the mine layout, tailings disposal, and reclamation for each mine plan follow.

TABLE 7.2-1

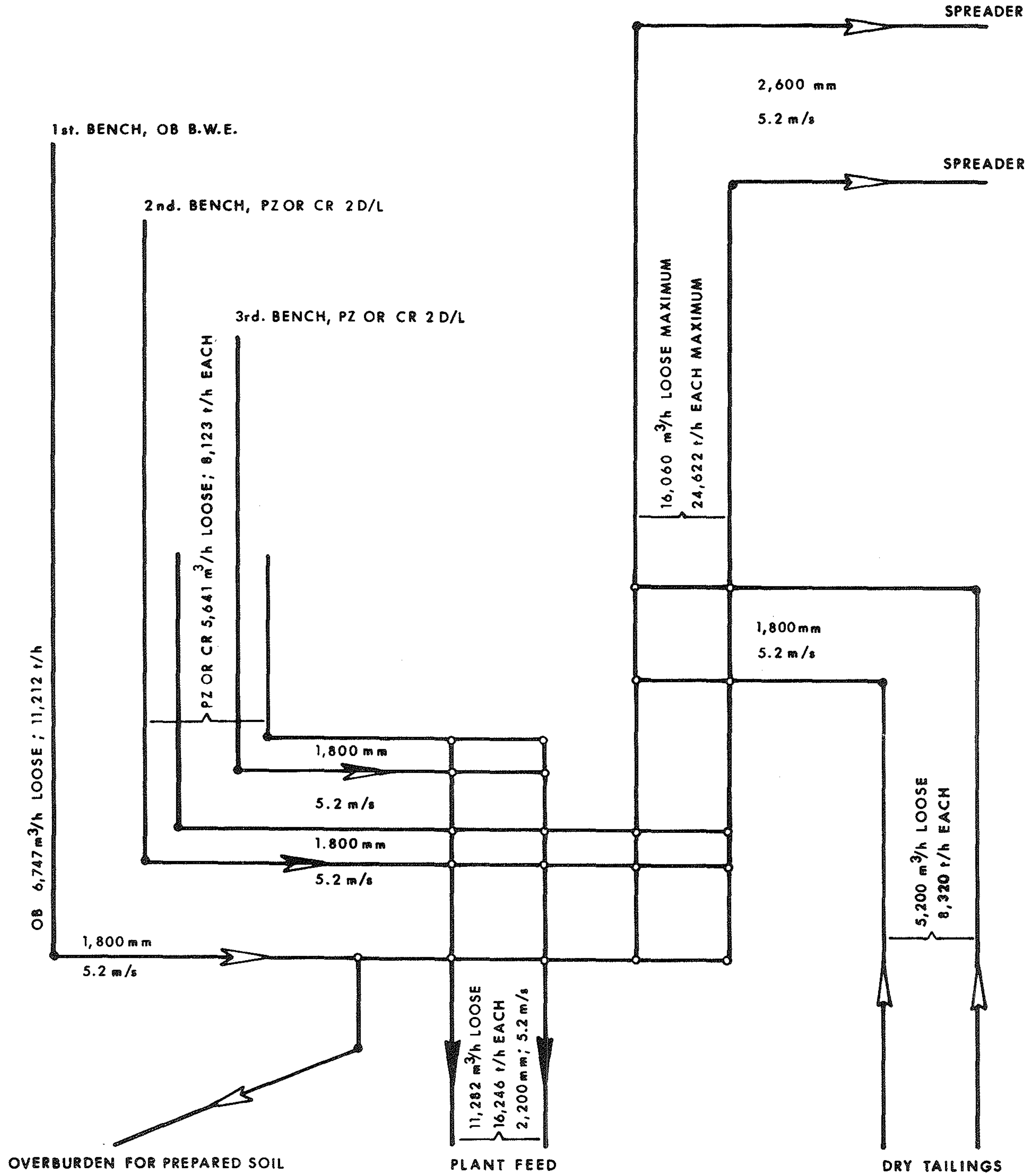
Ore Body No.2 , 120,000 B.P.C.D., -Production Schedule, 1 Bucket Wheel Excavator and 4 Draglines

Year	Top Bench Waste bm^3 x 10^6	Top Bench Feed bm^3 x 10^6	Top Bench Total bm^3 x 10^6	Middle Bench Waste bm^3 x 10^6	Middle Bench Feed bm^3 x 10^6	Middle Bench Total bm^3 x 10^6	Bottom Bench Backcast Waste bm^3 x 10^6	Bottom Bench Feed bm^3 x 10^6	Bottom Bench Total bm^3 x 10^6	Mine Waste bm^3 x 10^6	Mine Feed bm^3 x 10^6	Mine Total bm^3 x 10^6	Mine Bitumen tonnes x 10^6	Mine Crude barrels x 10^6
- 2	14.150		14.150	–	–	–	–	–	–	14.150	–	14.150	–	–
- 1	14.150		14.150	–	–	–	–	–	–	14.150	–	14.150	–	–
1	14.150		14.150	3.568	23.476	27.044	1.895	7.894	9.789	19.613	31.370	50.983	6.272	28.834
2	14.150		14.150	2.031	18.646	20.677	2.980	19.393	22.373	19.161	38.039	57.200	7.106	36.726
3	14.150		14.150	3.018	21.325	24.343	3.251	22.023	25.274	20.419	43.348	63.767	9.115	43.479
4	14.150		14.150	4.685	20.452	25.137	4.100	21.319	25.419	22.935	41.771	64.706	9.368	43.800
5	14.150		14.150	7.362	19.011	26.373	6.464	19.872	26.336	27.976	38.883	66.859	9.109	43.800
6	14.150		14.150	7.319	18.904	26.223	8.549	18.819	27.368	30.018	37.723	67.741	8.649	43.800
7	14.150		14.150	4.461	20.543	25.004	7.619	19.325	26.944	26.230	39.868	66.098	9.090	43.800
8	14.150		14.150	2.239	21.097	23.336	4.200	21.505	25.705	20.589	42.602	63.191	9.246	43.800
9	14.150		14.150	2.519	19.481	22.000	2.870	20.992	23.862	19.539	40.473	60.012	8.815	43.800
10	14.150		14.150	4.840	18.608	23.448	4.032	18.830	22.862	23.022	47.438	60.460	9.317	43.800
11	14.150		14.150	1.874	17.753	19.627	5.502	18.493	23.995	21.526	36.246	57.772	8.896	43.800
12	14.150		14.150	1.751	18.301	20.052	1.679	18.079	19.758	17.580	36.380	53.960	8.638	43.800
13	14.150		14.150	7.405	20.483	27.888	4.166	19.424	23.590	25.721	39.907	65.628	9.098	43.800
14	14.150		14.150	4.325	20.047	24.372	8.313	20.272	28.585	26.788	40.319	67.107	8.972	43.800
15	14.150		14.150	3.158	20.675	23.833	4.429	20.273	24.702	21.737	40.948	62.685	8.940	43.800
16	14.150		14.150	3.304	23.998	27.302	4.281	21.376	25.657	21.735	45.374	67.109	9.469	43.800
17	14.150		14.150	2.958	25.285	28.243	3.902	25.155	29.057	21.010	50.440	71.450	9.698	43.800
18	14.150		14.150	3.100	17.605	20.705	4.340	22.847	27.187	21.590	40.452	62.042	9.267	43.800
19	14.150		14.150	2.227	17.365	19.592	3.188	16.942	20.130	19.565	34.307	53.872	9.095	43.800
20	12.750		12.750	3.178	20.445	23.623	2.210	18.254	20.464	18.138	38.699	56.837	8.907	43.800
21	7.250		7.250	2.730	24.632	27.362	4.119	22.068	26.187	14.099	46.700	60.799	9.574	43.800
22	7.250		7.250	0.971	23.888	24.859	3.128	25.538	28.666	11.349	49.426	60.775	9.471	43.800
23	7.250		7.250	0.532	21.620	22.152	1.024	24.191	25.215	8.806	45.811	54.617	9.092	43.800
24	2.188		2.188	0.532	16.726	17.258	0.876	23.958	24.834	3.596	40.684	44.280	8.218	38.812
25	–		–	–	–	–	0.339	6.313	6.652	0.339	6.313	6.652	1.221	5.478
Total	333.838		333.838	80.087	490.366	570.453	97.456	493.155	590.611	511.381	983.521	1,494.902	214.643	1,029.329



MATERIALS HANDLING SYSTEM-120,000 BPCD
1 B.W.E. & 4 DRAGLINES-MINIMUM AND IMPROVED LEVEL

FIGURE 7.2-1



MATERIALS HANDLING SYSTEM - 120,000 BPCD
1 B.W.E. AND 4 DRAGLINES - ENHANCED LEVEL

FIGURE 7.2-2

7.2.1 MINIMUM (WET) LEVEL OF RECLAMATION

The major aspects of the development of the 120,000 BPCD mine at the Minimum Level are depicted by five drawings, accompanied by tailings disposal and reclamation schedules (See Tables 7.2.1-1 and 7.2.1-2). The mining schedule, which is common to the plans for all three levels of reclamation, has been illustrated previously in Table 7.2-1. A drawing-by-drawing discussion follows:

Mining and Tailings Disposal - Year 3
(Techman Drawing No. D22918-20-00)

By Year 3 all the working faces of the mine are fully developed. Overburden removal has advanced beyond the oil sands removal benches by over a kilometer. Overburden and reject are placed by spreader into outside dumps south of the mine. Muskeg is dewatered, excavated, and placed into muskeg dumps prior to overburden removal. The out-of-pit tailings pond is fully operational with water being recycled to the plant. An oversize reject dump has been started northeast of the plant site.

Mining and Tailings Disposal - Year 15
(Techman Drawing No. D22918-21-00)

The mining faces have advanced to a position north of the plant site. The waste dump south of the mine has been completed. Waste materials are being placed into the combined waste and reject dump northeast of the plant site. The outside tailings pond has been sanded-in and the active tailings pond is now located in the second mined-out pit. Sludge from the out-of-pit tailings pond has been pumped into the sludge pond, and currently the sludge from the active pond is being pumped into the sludge pond. A sand dyke has been constructed in the first neck of the ore body to form the sludge pond. Another sand dyke is in the process of being constructed in the second neck of the ore body to form the first in-pit tailings pond. Reclamation of the outside tailings pond dyke slope commenced in Year 11. The southerly-located outside waste dump was reclaimed in Years 13 and 14.

Mining and Tailings Disposal - Year 22
(Techman Drawing No.D22918-22-00)

The mining face is now in the extreme north of the ore body. Muskeg removal has been completed and overburden removal is to be completed within the next year. A second in-pit tailings pond is in operation. Low sand dykes have been constructed prior to Year 22 along the western mine boundary to contain sludge and tailings sand to the specified elevation. The waste dump northeast of the plant site is not required as overburden and reject can now be backfilled into the mine. The reclamation of the northerly waste dump was completed in Year 18 and 19, and sanded-in portions of the tailings pond in Year 15 - 17. The reclamation of the first in-pit tailings pond is underway.

Material Distribution Plan
(Techman Drawing No. D22918-23-00)

This plan shows the types of materials that must be surfaced with prepared soil. The sludge pond surface is currently considered unreclaimable. Waste dumps are surfaced with acceptable overburden materials by the spreader during the construction of the dumps and therefore require only the application of muskeg, whereas tailings sand surfaces require the application of both overburden and muskeg to form a prepared soil surface. Muskeg dumps require no surfacing. The end-pit is filled with water to form a fresh water lake. All exposed pit walls (primarily along the eastern pit boundaries) require application of prepared soil. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-24-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurred are shown. Only the sludge pond remains wet and unreclaimable. Plant species are selected according to the reclamation objectives for the Minimum Level as described in Chapter 4.0. Table 7.2.1-2 is a year-by-year summary of prepared soil manufacture and placement.

Ore Body No. 2, 120,000 B.P.C.D.- 1 B.W.E. & 4 Draglines
TAILINGS SCHEDULE
FOR MINIMUM LEVEL OF RECLAMATION

TABLE 7.2.1-1

Outline of Tailings Disposal Scheme:

- Years 1-15 Tailings to out-of-Pit Pond, until Full of Sand.
- Years 10-15 Sludge Removed from out-of-Pit Pond to in-Pit Sludge Pond (Pit 1).
- Years 15-20 Tailings to First in-Pit Tailings Pond (Pit 2).
- Years 17-20 Sludge Removed from First in-Pit Tailings Pond to Sludge Pond.
- Years 21-25 Tailings to Second in-Pit Tailings Pond (Pit 3).
- Years 23-25 Sludge Removed from Second in-Pit Tailings Pond to Sludge Pond.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge Rehandle Volume [$m^3 \times 10^6$]
1	71.037	26.759	12.357	31.922	15.000	16.922	0
2	86.141	32.448	14.984	38.709	11.500	27.209	0
3	98.163	36.976	17.075	44.111	11.000	33.111	0
4	94.590	35.631	16.454	42.506	7.700	34.806	0
5	88.052	33.168	15.316	39.568	9.100	30.468	0
6	85.424	32.178	14.859	38.387	6.200	32.187	0
7	90.281	34.007	15.704	40.570	4.500	36.070	0
8	96.471	36.339	16.781	43.351	2.800	40.551	0
9	91.931	34.523	16.223	41.185	4.820	36.365	0
10	84.778	31.935	14.747	38.097	14.590	23.507	39.747
11	82.078	30.917	14.277	36.883	14.590	22.293	39.277
12	82.382	31.032	14.330	37.020	0	37.020	39.330
13	90.370	34.041	15.719	40.610	0	40.610	40.770
14	91.301	34.392	15.881	41.028	0	41.028	40.882
15	92.728	34.929	16.130	41.669	8.710	32.959	24.105
16	102.751	38.704	17.873	46.173	8.710	37.463	0
17	114.221	43.025	19.868	51.328	0	51.328	24.155
18	91.603	34.505	15.934	41.164	0	41.164	22.610
19	77.689	29.264	13.514	34.911	4.500	30.411	20.189
20	87.633	33.010	15.243	39.380	2.500	36.880	21.919
21	105.751	39.835	18.395	47.522	2.000	45.522	1.254
22	112.268	42.160	19.812	50.296	0	50.296	0
23	103.739	39.077	18.045	46.617	0	46.617	27.804
24	92.128	34.703	16.025	41.400	0	41.400	37.701
25	14.296	5.385	2.487	6.424	0	6,424	8.339
	2,227.306	838.943	388.033	1000.831	128.220	872 611	388.033

Ore Body No. 2, 120,000 B.P.C.D. - 1 B.W.E. & 4 Draglines
SCHEDULE FOR MINIMUM LEVEL OF RECLAMATION

TABLE No. 7.2.1-2

Soil Composition:
0.20m Muskeg
0.20m Overburden
0.20m Sand (where applicable)

Soil Manufacture:
Layer of muskeg and overburden (where required) are spread onto area to be reclaimed and plowed 0.6 m deep.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1						
2						
3						
4						
5						
6						
7						
8						
9						
10						
11	1,480	888	296	4.9	296	2.2
12	1,480	888	296	5.0	296	2.2
13	2,138	1,283	428	3.9	856	-
14	2,138	1,283	428	3.9	856	-
15	3,333	2,000	667	1.0	667	5.6
16	3,333	2,000	667	1.0	667	5.6
17	3,333	2,000	667	1.0	667	5.6
18	1,400	840	280	2.6	560	-
19	1,400	840	280	2.6	560	-
20	1,317	790	263	2.2	263	3.0
21	1,317	790	263	2.2	263	3.0
22	1,317	790	263	1.5	263	3.0
23	1,317	790	263	1.5	263	3.0
24	878	527	176	2.8	352	-
25	878	527	176	2.8	352	-
26	1,547	928	309	2.2	569	0.8
27	1,547	928	309	1.9	569	0.8
28	1,547	928	309	1.9	569	0.8
29	2,040	1,224	408	2.3	460	0.6
30	3,000	1,800	600	2.3	600	1.2
	36,740	22,044	7,348		9,948	

7.2.2 IMPROVED (DEWATERED) LEVEL OF RECLAMATION

The major aspects of the development of the 120,000 BPCD mine at the Improved Level are depicted by three drawings accompanied by tailings disposal and reclamation schedules (See Table 7.2.2-1 and 7.2.2-2). The mining schedule, which is common to the plans for all three levels of reclamation, has been illustrated previously in Table 7.2-1. A drawing-by-drawing discussion follows:

Mining and Tailings Disposal - Year 15
(Techman Drawing No. D22918-25-00)

The advance of the mining faces in the Improved Level is identical to that occurring at the Minimum Level, and consequently the Minimum Level drawings should be examined for this aspect of the development plan. The tailings disposal is noticeably different, and is illustrated in the drawing showing the status of development in Year 15. The operation of the sludge pond began in Year 6 when sludge was transferred into it from a sludge treatment plant designed to partially dewater the tailings pond sludge. A dyke constructed completely of overburden separates the sludge pond from the in-pit tailings pond. The in-pit tailings pond became operational in Year 14. The dyke located at the second neck in the ore body has been constructed from compacted tailings sand on the pond side (west side), and from spreader-backfilled overburden on the other side. The outside waste dumps were reclaimed in Years 11 to 15.

Material Distribution Plan
(Techman Drawing No. D22916-26-00)

This plan shows the types of materials that must be surfaced with prepared soil. The sludge pond surface is currently considered unreclaimable. Waste dumps are surfaced with acceptable overburden materials by the spreader during the construction of the dumps and therefore require only the application of muskeg, whereas tailings sand surfaces require the application of both overburden and muskeg to form a prepared soil surface. Muskeg dumps require no surfacing. The end-pit is filled with water to form a fresh water lake. All exposed pit walls require application of prepared soil. Compared to the plan at the Minimum Level of Reclamation, the end-pit lake is larger, the volume of outside dumps less, and the total area unreclaimable 50 % less (sludge pond). Refer to Section 10-6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. 22980-27-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurred are shown. Only the sludge pond remains wet and unreclaimable. Plant species are selected according to the reclamation objectives for the Improved Level as described in Chapter 4.0. Table 7.2.2-2 is a year-by-year summary of prepared soil manufacture and placement.

Ore Body No. 2, 120,000 B.P.C.D. - 1 B.W.E. & 4 Draglines
TAILINGS SCHEDULE
FOR IMPROVED LEVEL OF RECLAMATION
TABLE 7.2.2-1

Outline of Tailings Disposal Scheme:

- Years 1 - 14 Tailings to out-of-Pit Pond, until Full of Sand.
- Years 6 - 14 Sludge from out-of-Pit Pond, Treated (50% of Water and Most of Remaining Bitumen Removed). Treated Sludge to in-Pit Sludge Pond (Southern Half of Pit 1).
- Years 14-25 Tailings to in-Pit Tailings Pond.
- Years 17-25 Sludge from in-Pit Tailings Pond Treated. Treated Sludge to in-Pit Sludge Pond.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge into Treatment [$m^3 \times 10^6$]
1	17.037	26.759	12.357	31.922	15.000	16.922	0
2	86.141	32.448	14.984	38.709	11.500	27.209	0
3	98.163	36.976	17.075	44.111	11.000	33.111	0
4	94.590	35.631	16.454	42.506	7.700	34.806	0
5	88.052	33.168	15.316	39.568	9.100	30.468	0
6	85.424	32.178	14.859	38.387	6.200	32.187	24.012
7	90.281	34.007	15.704	40.570	4.500	36.070	24.012
8	96.471	36.339	16.781	43.351	2.800	40.551	24.012
9	91.931	34.523	16.223	41.185	0.700	40.485	24.012
10	84.778	31.935	14.747	38.097	0	38.097	24.012
11	82.078	30.917	14.277	36.883	0	36.883	24.012
12	82.382	31.032	14.330	37.020	0	37.020	24.012
13	90.370	34.041	15.719	40.610	0	40.610	24.012
14	91.301	34.392	15.881	41.028	1.000	40.028	19.690
15	92.728	34.929	16.130	41.669	0	41.669	0
16	102.751	38.704	17.873	46.173	5.740	40.433	0
17	114.221	43.025	19.868	51.328	3.820	47.508	21.312
18	91.603	34.505	15.934	41.164	0	41.164	21.312
19	77.689	29.264	13.514	34.911	0	34.911	21.312
20	87.633	33.010	15.243	39.380	0	39.380	21.312
21	105.751	39.835	18.395	47.522	0	47.522	21.312
22	112.268	42.160	19.812	50.296	0	50.296	21.312
23	103.739	39.077	18.045	46.617	0	46.617	21.312
24	92.128	34.703	16.025	41.400	0	41.400	21.312
25	14.296	5.385	2.487	6.424	0	6.424	5.750
	2,173.806	838.943	388.033	1000.831	79.060	918.771	388.032

Ore Body No. 2, 120,000 B.P.C.D.- 1 B.W.E. & 4 Draglines
SCHEDULE FOR IMPROVED LEVEL OF RECLAMATION
TABLE No. 7.2.2-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Alternating layers of muskeg and overburden are scraped at a sloping face of pile by dozers.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1							
2							
3							
4							
5							
6							
7							
8							
9							
10							
11	970	970	0.8	323	2.0	647	0.8
12	970	970	0.8	323	2.0	647	0.8
13	970	970	0.8	323	2.0	647	0.8
14	865	865	0.7	288	1.0	577	0.5
15	865	865	0.7	288	1.0	577	0.5
16	1,620	1,620	5.0	540	1.8	1,080	0.5
17	1,620	1,620	5.0	540	1.8	1,080	0.5
18	1,620	1,620	5.0	540	1.8	1,080	0.5
19	1,620	1,620	6.0	540	1.8	1,080	0.5
20	1,620	1,620	6.0	540	1.8	1,080	0.5
21	1,620	1,620	6.0	540	1.8	1,080	0.5
22	1,620	1,620	7.0	540	1.8	1,080	0.5
23	1,620	1,620	7.0	540	1.8	1,080	0.5
24	2,100	2,100	1.0	700	2.0	1,400	0.5
25	2,100	2,100	2.6	700	2.0	1,400	0.5
26	2,200	2,200	2.0	740	3.0	1,480	2.8
27	2,200	2,200	2.0	740	3.0	1,480	2.8
28	2,200	2,200	3.0	740	3.0	1,480	2.8
29	2,200	2,200	4.5	740	3.0	1,480	2.8
30	3,000	3,000	1.0	1,000	3.1	2,000	3.0
	33,600	33,600		11,225		22,455	

7.2.3 ENHANCED (DRY) LEVEL OF RECLAMATION

The major aspects of the development of the 120,000 BPCD mine at the Enhanced Level are depicted by three drawings, accompanied by tailings disposal and reclamation schedules (See Table 7.2.3-1 and 7.2.3-2). The mining schedule, which is common to the plans at all three levels of reclamation, has been illustrated previously in Table 7.2-1. A drawing-by-drawing discussion follows:

Mining and Tailings Disposal - Year 15
(Techman Drawing No. D22918-28-00)

At the Enhanced Level, the mining face advance is identical to that occurring at the Minimum and Improved Levels. Since the tailings stream is dry, the mine is backfilled with tailings sand, overburden, and reject by means of two spreaders operating on two benches. No out-of-pit tailings pond is required, but an outside waste dump is needed while the pit is opened and sufficient room for backfilling is developed. This waste dump is reclaimed in Years 2 and 3. Reclamation progressively follows the backfilling operation, and remains approximately 1 km from the upper face. By Year 15, about fifty percent of the muskeg required for prepared soil manufacture has been mined. The prepared soil blending yard is located north of the plant site. A 1,000 mm wide belt conveyor extends from the blending yard to the prepared soil field stockpile. This conveyor changes in length, depending on the progress of the backfilling operation. The muskeg stripped by the mining operation is placed into permanent disposal dumps. By Year 15, the muskeg stripping operation has advanced into the last quarter of the pit, just north of the overburden removal conveyor.

Material Distribution Plan
(Techman Drawing No. D22916-29-00)

This plan shows the types of materials that must be surfaced with prepared soil. The entire mine is reclaimable. In addition to a small, final end-pit lake, a residual lake is formed by the removal of muskeg from the muskeg mine. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-30-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurs are shown. No unreclaimed areas remain. Plant species are selected according to the reclamation objectives for the Enhanced Level described in Chapter 4.0.

Ore Body No. 2, 120,000 B.P.C.D.- 1 B.W.E. & 4 Draglines
TAILINGS SCHEDULE
FOR ENHANCED LEVEL OF RECLAMATION
TABLE 7.2.3-1

Outline of Tailings Disposal Scheme:
-Dry Tailings Conveyed with Overburden and Center Reject for First 1.25 Years to out-of-Pit Waste Dump, then into Mined- out Pit.

YEAR	Volume of Dry Tailings Produced [$m^3 \times 10^6$]	Dry Tailings Conveying Distance * [m]
1	33.001	9.6
2	40.017	12.6
3	45.602	13.3
4	43.943	14.1
5	40.905	15.4
6	39.685	16.3
7	41.941	11.8
8	44.817	11.7
9	42.578	11.5
10	39.385	6.8
11	38.131	10.3
12	38.272	11.1
13	41.982	13.7
14	42.416	14.1
15	43.077	8.6
16	47.733	9.5
17	53.063	9.8
18	42.556	14.0
19	36.091	15.9
20	40.711	16.7
21	49.128	17.1
22	51.996	19.6
23	48.193	21.7
24	42.800	22.7
25	6.641	23.9
	1,034.664	

*NOTE:
These are Total Lengths of Two Conveyor Systems from Plant to Two Spreaders. At the Same Time Overburden is Transported Via These Conveyors.

Ore Body No.2, 120,000 B.P.C.D. - 1 B.W.E. & 4 Draglines
SCHEDULE FOR ENHANCED LEVEL OF RECLAMATION

TABLE No. 7.2.3-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Stacker deposits layers of muskeg and overburden into piles. Components are mixed by bucket wheel reclaimer.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by conveyors) [km]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by pipeline) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by conveyors) [km]
1	0	0	0	0	0	0	0	0
2	866	866	4.5	1.8	289	8.3	577	8.3
3	866	866	4.5	1.8	289	8.3	577	8.2
4	933	933	4.5	2.3	311	8.3	622	8.1
5	933	933	4.5	2.3	311	8.3	622	7.1
6	933	933	4.5	2.3	311	8.3	622	6.7
7	933	933	4.5	2.3	311	8.3	622	7.0
8	933	933	1.8	2.0	311	8.3	622	7.1
9	933	933	1.8	2.0	311	8.3	622	6.5
10	933	933	1.8	2.0	311	8.3	622	7.2
11	933	933	1.8	2.0	311	8.3	622	7.7
12	933	933	1.8	2.0	311	8.3	622	7.9
13	1,197	1,197	0	2.0	399	8.3	798	8.5
14	1,197	1,197	0	2.0	399	8.3	798	7.1
15	1,197	1,197	0	2.0	399	8.3	798	7.4
16	1,197	1,197	0	2.0	399	8.3	798	7.7
17	1,000	1,000	0	1.9	333	8.3	667	7.9
18	1,000	1,000	0	1.9	333	8.3	667	12.1
19	1,000	1,000	2.5	0.8	333	8.3	667	8.3
20	1,000	1,000	2.5	0.8	333	8.3	667	9.3
21	1,000	1,000	2.5	0.8	333	8.3	667	9.8
22	955	955	2.5	2.3	318	8.3	637	10.4
23	955	955	2.5	2.3	318	8.3	637	10.8
24	1,377	1,377	4.5	1.5	459	8.3	918	2.1
25	1,174	1,174	4.5	1.9	391	8.3	783	2.1
26	1,174	1,174	4.5	1.9	391	8.3	783	2.1
27	1,000	1,000	3.0	0	333	8.3	667	2.1
28	1,000	1,000	3.0	0	333	8.3	667	2.1
29	1,000	1,000	3.0	0	333	8.3	667	2.1
30	0	0	0	0	0	0	0	0
	28,552	28,552			9,514		19,038	

7.3 MINE PLANS EMPLOYING BUCKET WHEEL EXCAVATORS

The entire production of the mine is achieved with three BWE's operating on three benches of approximately equal height. The bucket wheel excavator on the first bench typically removes all the overburden, and also a certain quantity of oil sands, on a scheduled basis. The excavators on the second and third benches load oil sands and centre reject. The three excavators are identical in size, creating opportunities for interchanging the BWE's between benches, and for standardization.

A slewing mining system allows the mine to be operated with only one complete reconstruction of the conveyor distribution point. The first distribution point is situated just west of the plant site. At the Minimum and Improved Levels, the relocated distribution point is constructed near the southern end of the final mined-out void. At the Enhanced Level, the reconstructed distribution point is located north of and immediately adjacent the plant site. This distribution point is justified in that large volumes of tailings sand as well as overburden and reject are handled by conveyors, and backtracking large quantities of sand is not advisable. Overburden and reject are placed at the bottom of the pit to reduce the length of conveyors at the Minimum and Improved Levels. The same principle applies at the Enhanced Level except that backfill dump heights are much higher.

A single spreader dumps the overburden onto the pit floor as backfill. As the backfill conveyor approaches the alignment of the in-pit dyke, the spreader separates material suitable for dyke construction by dumping overburden into a stockpile, using a high dump mode of operation. This material is carried into the dyke construction zone by trucks or scrapers, and is compacted. As the backfill conveyor passes over the dyke base, the spreader again places selected overburden in a temporary stockpile for rehandle into the dyke with trucks or scrapers. In this fashion the starter dyke is constructed to the required elevation. The remainder of the dyke is constructed with hydraulically-placed and compacted tailings sand.

At the Improved Level the technique is similar, except that the second in-pit dyke (the sludge pond dyke) is constructed completely of overburden. This dyke cannot be constructed of tailings sand since the extra tailings water and untreated sludge would affect the limited storage capacity of the second pit for thickened sludge, as well as adding unwanted tailings water to again dilute the treated sludge.

The inboard side of the second dyke is riprapped for protection against erosion by waves. The slope of this dyke must be designed to assure overall stability of the dyke even when submerged and hence may have to be shallower than dyke slopes which are covered by tailings sand. Once the spreading operation has proceeded beyond the second dyke all the overburden and centre reject are dumped onto the pit floor. This portion of the backfilled material is submerged in the final lake.

The backfilling operation at the Enhanced Level requires two spreaders, since the volume to be handled is more than at the Minimum and Improved Levels. A short conveyor between the distribution point and the blending yard is required for the transport of overburden needed in the manufacture of prepared soil. This conveyor has the same capacity as the face conveyor being loaded by the BWE. When the distribution point is too far from the prepared soil blending yard, one of the plant feed conveyors is utilized as described in section 7.2.

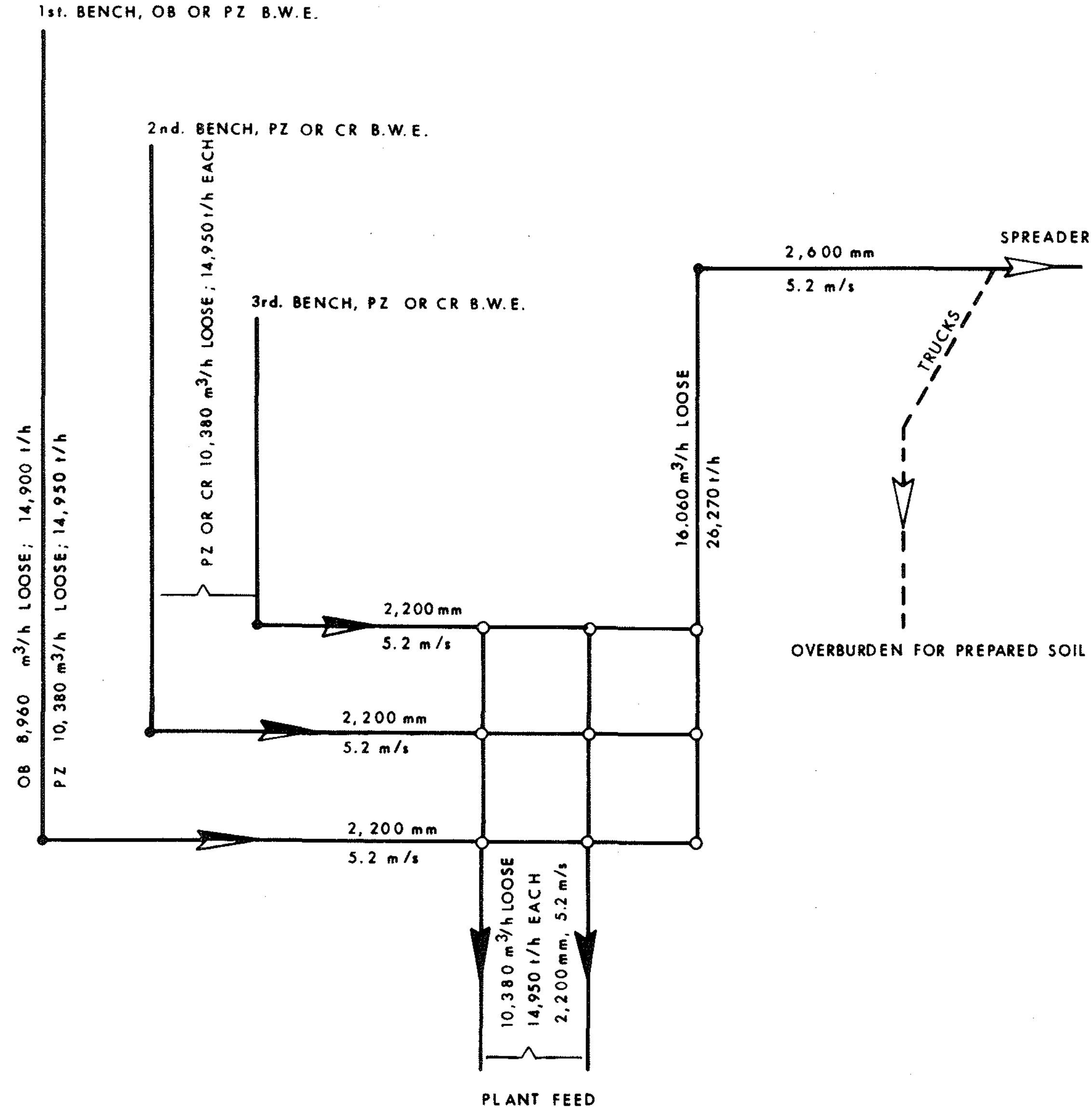
The spreader operating on the upper bench operates most of the time in the "low dump" mode and a minor portion of the time in the "high dump" mode. The spreader on the lower bench is shown as operating only in the "low dump" mode, but flexibility exists to vary the ratio of material placed in "high dump" as compared to that placed in "low dump". The ratio need not be the same for both spreaders. It is preferable to use the low dump whenever possible, the limiting factor being the stability of the pile on which the spreader operates. This is in the order of about 40 m for this study. The characteristics of the dry tailings sand, the overburden, and the various sequences of material determine the long term dumping method.

The quantities and schedules of overburden, reject, oil sand, bitumen, and crude oil are identical for the three mine plans. Table 7.3-1 shows the mining mass balance applicable for the 120,000 BPCD bucket wheel options. The sizing of the material handling system is schematically illustrated in Figures 7-3.1 and 7-3.2. The production of the BWE's, spreaders and conveyors is based on 5,000 operating hours annually out of a total of 8,760 calendar hours. Details regarding mine layout, tailings disposal and reclamation for each mine follow.

TABLE 7.3-1

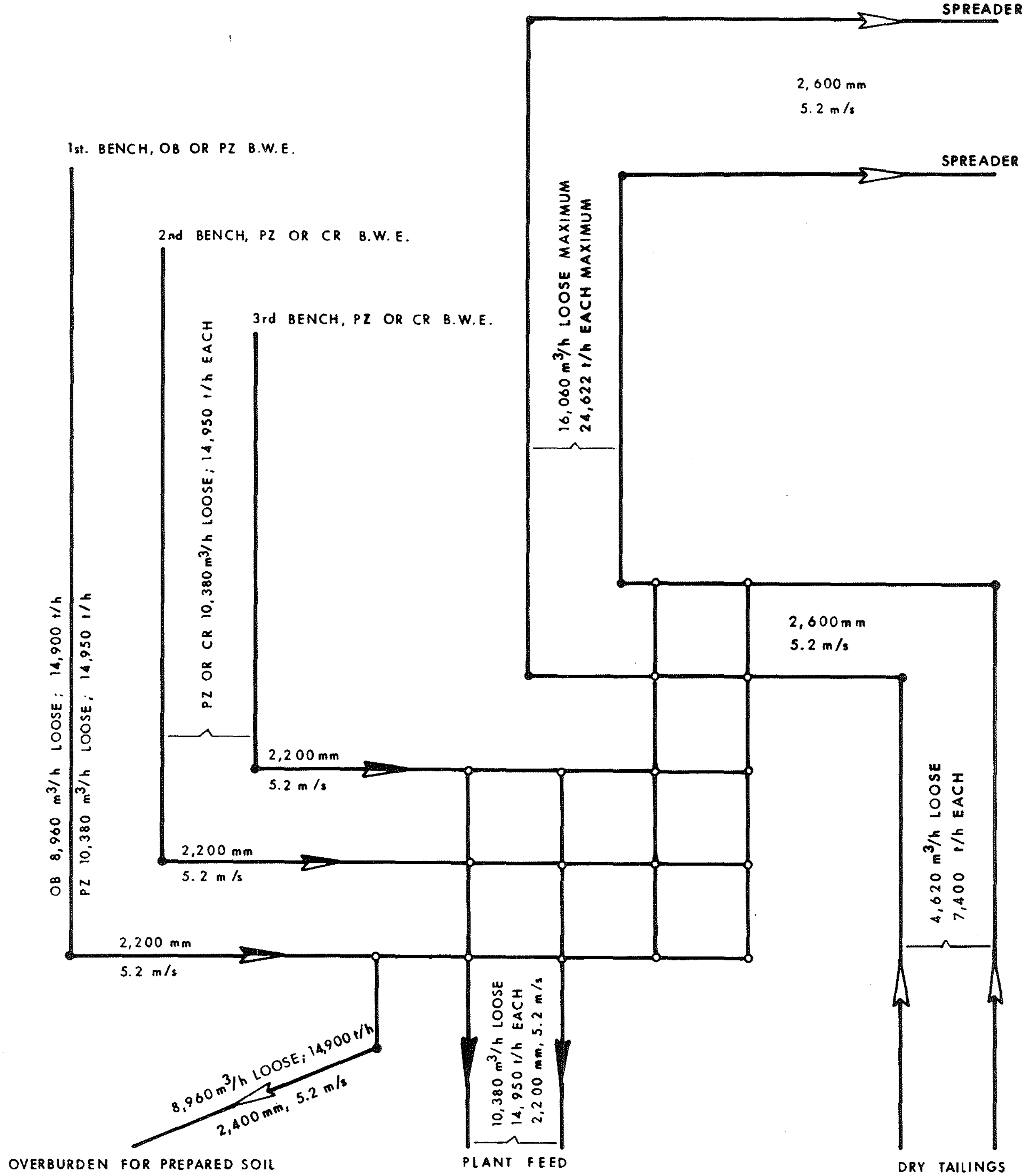
Ore Body No.2, 120,000 B.P.C.D., - Production Schedule, 3 Bucket Wheel Excavators Scheme -

Year	Top Bench			Middle Bench			Bottom Bench			Mine				
	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Bitumen tonnes x 10^6	Crude barrels x 10^6
-2	11.100	-	11.100	-	-	-	-	-	-	11.100	-	11.100	-	-
-1	11.600	-	11.600	-	-	-	-	-	-	11.600	-	11.600	-	-
1	5.297	6.486	11.783	2.351	12.149	14.500	1.249	6.265	7.514	8.897	24.900	33.797	5.375	25.865
2	12.000	-	12.000	2.594	15.294	17.888	3.806	15.906	19.712	18.400	31.200	49.600	7.116	34.242
3	9.000	5.000	14.000	3.899	16.101	20.000	4.207	15.799	20.000	17.100	36.900	54.000	8.371	40.285
4	10.000	6.200	16.200	4.053	15.947	20.000	3.547	16.453	20.000	17.600	38.600	56.200	8.811	42.401
5	11.000	6.900	17.900	4.580	15.420	20.000	4.020	15.980	20.000	19.600	38.300	57.900	8.367	40.263
6	10.000	11.500	21.500	5.996	14.004	20.000	7.004	12.996	20.000	23.000	38.500	61.500	8.156	39.249
7	7.000	12.200	19.200	5.576	14.424	20.000	9.524	10.476	20.000	22.100	37.100	59.200	8.075	38.859
8	10.000	10.000	20.000	4.277	15.723	20.000	8.023	11.977	20.000	22.300	37.700	60.000	8.417	40.505
9	11.700	8.400	20.100	4.954	15.046	20.000	5.646	14.354	20.000	22.300	37.800	60.100	8.553	41.158
10	18.100	2.000	20.100	3.055	16.945	20.000	2.545	17.455	20.000	23.700	36.400	60.100	8.345	40.160
11	19.200	1.100	20.300	1.575	18.425	20.000	4.625	15.375	20.000	25.400	34.900	60.300	8.274	39.817
12	11.000	0.100	11.100	2.859	17.141	20.000	2.747	17.259	20.000	16.600	34.500	51.100	8.476	40.790
13	11.000	1.200	12.200	5.056	14.944	20.000	2.844	17.156	20.000	18.900	33.300	52.200	8.208	39.499
14	11.000	6.200	17.200	4.243	15.757	20.000	8.154	11.843	20.000	23.400	33.800	57.200	8.087	38.920
15	11.000	4.400	15.400	2.580	17.420	20.000	3.620	16.380	20.000	17.200	38.200	55.400	8.689	41.815
16	11.000	9.100	20.100	3.049	16.951	20.000	3.951	16.049	20.000	18.000	42.100	60.100	9.450	45.475
17	7.000	11.500	18.500	3.234	16.766	20.000	4.266	15.734	20.000	14.500	44.000	58.500	10.132	48.757
18	11.000	9.300	20.300	3.000	17.000	20.000	4.200	15.800	20.000	18.200	42.100	60.300	9.382	45.151
19	14.000	2.600	16.600	3.002	16.998	20.000	4.298	15.702	20.000	21.300	35.300	56.600	7.379	45.511
20	20.000	-	20.000	4.011	15.989	20.000	2.789	16.611	19.400	26.800	32.600	59.400	7.174	34.523
21	15.000	4.300	19.300	3.189	16.811	20.000	4.811	15.189	20.000	23.000	36.300	59.300	8.947	43.058
22	9.000	5.400	14.400	1.611	18.389	20.000	5.189	14.811	20.000	15.800	38.600	54.400	9.553	45.972
23	6.000	5.900	11.900	2.051	17.949	20.000	3.949	16.051	20.000	12.000	39.900	51.900	8.748	42.099
24	2.000	4.900	6.900	1.889	18.111	20.000	3.111	16.889	20.000	7.000	39.900	46.900	8.261	39.754
25	1.000	-	1.000	0.550	4.575	5.125	1.650	13.725	15.375	3.200	18.300	21.500	3.847	18.515
Total	285.997	134.686	420.683	80.234	394.279	477.513	109.766	372.235	482.001	478.997	901.200	1,380.197	204.193	982.643



MATERIALS HANDLING SYSTEM-120,000 BPCD
3 B.W.E.- MINIMUM AND IMPROVED LEVEL

FIGURE 7.3-1



MATERIALS HANDLING SYSTEM - 120,000 BPCD
3 B.W.E. - ENHANCED LEVEL

FIGURE 7.3-2

7.3.1 MINIMUM (WET) LEVEL OF RECLAMATION

The major aspects of development of the 120,000 BPCD mine at the Minimum Level are depicted by nine drawings accompanied by tailings disposal and reclamation schedules (see Table 7.3.1-1 and 7.3.1-2). The mining schedule, which is common to the plans for all three levels of reclamation, has been illustrated previously in Table 7.3-1. A drawing-by-drawing discussion follows:

General Mine Layout
(Techman Drawing No. D22910-31-00)

The location and ultimate size of the pit, out-of-pit tailings pond, outside dumps, and plant site are shown for the plans at the Minimum, Improved, and Enhanced Levels of Reclamation. The pit remains identical in all cases, but the disturbance out-of-pit varies.

Mine Opening - Year 1
(Techman Drawing No. D22918-32-00)

Oil sands mining begins in Year 1. Sufficient overburden has been removed to allow the progressive installation of two additional oil sands bench conveyors. The drawing shows only the overburden and oil sands conveyor just prior to the start of the first oil sands mining bench. An outside waste dump has been constructed just south of the out-of-pit tailings pond site. Some of the overburden may be used in the construction of the starter dykes.

Mining and Tailings Disposal - Year 7
(Techman Drawing No. D22918-33-00)

By Year 7, the backfilling of the mine with overburden and reject waste is well under way, and the outside waste dump has been reclaimed. The mining faces are fully developed and are followed closely by the backfilling operation. The first bench leads the lower two benches by a maximum of one-and-one-half kilometres. Overburden for prepared soil manufacture is being selectively separated from the backfill by the spreader operating within the pit. The in-pit stockpiled materials must be hauled to the stockpiles located outside of the pit boundary for later use in reclamation activities.

Mining and Tailings Disposal - Year 12
(Techman Drawing No. D22918-34-00)

By Year 12, the first in-pit starter dyke has been constructed and dyke building with compacted sand has begun. The out-of-pit tailings pond dyke has reached ultimate crest height and the pond has been filled to capacity. Tailings are now being pumped into the first in-pit tailings pond. The reclamation of the out-of-pit tailings pond dyke slopes has been completed and the reclamation of the sand beaches in the pond is under way.

Mining and Tailings Disposal - Year 15
(Techman Drawing No. D22918-35-00)

In this drawing the mine is shown developing into the northern half of the mine. The uppermost bench is well advanced, while the lower bench is still operating within the first half of the mine. The middle bench is shown in a transitory phase in which it is finalizing removal of middle bench oil sands in the southern half of the mine. When the bench slews to the pit boundary just north of the plant site, the conveyor system will be relocated to allow re-entry into the distribution point from the north. The reclamation of the out-of-pit tailings pond is completed in Year 15.

Mining and Tailings Disposal - Year 19
(Techman Drawing No. D22918-36-00)

The mining faces have now advanced into the northern quarter of the mine. The conveyor distribution point is located in its final position, allowing the face conveyors to be simply slewed until completion of the mine. The in-pit tailings dyke has been constructed for the second in-pit tailings pond. Reclamation of the first in-pit pond has started.

Ore Body No. 2, 120,000 B.P.C.D. - 3 Bucket Wheel Excavators

TAILINGS SCHEDULE FOR MINIMUM LEVEL OF RECLAMATION

TABLE 7.3.1-1

Outline of Tailings Disposal Scheme

- Years 1-11 Tailings to Out-of-Pit Pond
- Years 12-20 Tailings to First in-Pit-Pond
- Years 21-25 Tailings to Second in-Pit Pond

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]
1	56.360	21.230	9.800	25.330	12.660	12.670
2	70.660	26.620	12.290	31.750	15.870	15.880
3	83.650	31.510	14.550	37.590	11.280	26.310
4	87.430	32.930	15.210	39.290	11.780	27.510
5	86.810	32.700	15.100	39.010	9.750	29.260
6	87.280	32.880	15.180	39.220	8.240	30.980
7	84.040	31.660	14.620	37.760	7.170	30.590
8	85.420	32.180	14.860	38.380	6.140	32.240
9	85.640	32.260	14.900	38.480	4.230	34.250
10	82.480	31.070	14.350	37.060	2.220	34.840
11	79.070	29.790	13.750	35.530	0.710	34.820
12	78.200	29.460	13.600	35.140	11.900	23.240
13	75.520	28.450	13.140	33.930	11.370	22.560
14	76.480	28.810	13.300	34.370	10.780	23.590
15	86.600	32.620	15.060	38.920	3.060	35.860
16	95.390	35.930	16.590	42.870	2.580	40.290
17	99.530	37.490	17.310	44.730	2.080	42.650
18	95.300	35.900	16.580	42.820	1.410	41.410
19	80.020	30.140	13.920	35.960	0.840	35.120
20	73.770	27.790	12.830	33.150	0.270	32.880
21	82.200	30.960	14.300	36.940	11.330	25.610
22	87.340	32.900	15.190	39.250	10.120	29.130
23	90.180	33.970	15.690	40.520	2.410	38.110
24	90.370	34.040	15.720	40.610	1.190	39.420
25	41.500	15.630	7.220	18.650	0	18.650
	2,041.240	768.920	355.060	917.260	159.390	757.870

Ore Body No. 2, 120,000 B.P.C.D.- 3 Bucket Wheel Excavators
SCHEDULE FOR MINIMUM LEVEL OF RECLAMATION
TABLE No. 7.3.1 - 2

Soil Composition:
0.20m Muskeg
0.20m Overburden
0.20m Sand (where applicable)

Soil Manufacture:
Layer of muskeg and overburden (where required) are spread onto area to be reclaimed and plowed 0.6 m deep.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1	183	110	37	9.2	37	7.4
2	183	110	37	8.7	37	7.4
3	183	110	37	8.0	37	7.4
4	183	110	37	11.5	37	7.4
5	183	110	37	12.3	37	7.4
6	183	110	37	12.3	37	7.4
7	183	110	37	6.7	37	7.4
8	183	110	37	6.3	37	7.4
9	183	110	37	6.1	37	7.4
10	183	110	37	6.2	37	7.4
11	182	109	36	5.8	36	7.4
12	870	522	174	6.0	174	7.7
13	870	522	174	6.3	174	7.7
14	870	522	174	10.7	174	7.7
15	870	522	174	8.4	174	7.7
16	0	0	0	0	0	0
17	0	0	0	0	0	0
18	50	30	10	0.8	0	0
19	300	180	60	6.3	0	0
20	50	30	10	1.5	0	0
21	790	474	158	6.4	158	9.7
22	700	420	140	9.9	140	7.8
23	700	420	140	6.7	140	5.7
24	700	420	140	1.6	140	3.5
25	700	420	140	1.8	140	2.8
26	1,610	966	322	2.8	322	3.1
27	1,610	966	322	3.3	322	11.8
28	1,800	1,080	360	1.2	360	11.8
29	1,800	1,080	360	1.9	360	9.6
30	0	0	0	0	0	0
	16,302	9,783	3,264		3,184	

Mining and Tailings Disposal - Year 23
(Techman Drawing No. D22918-37-00)

Only two years of mining remain and consequently the benches have been slewed almost into final position. All but the "wet" portion of the first in-pit tailings pond has been reclaimed. The second in-pit pond is still operational and will not be reclaimed until after completion of mining.

Material Distribution Plan
(Techman Drawing No. D22916-38-00)

This plan shows the types of material that must be surfaced with prepared soil. Wet tailings pond surfaces are currently considered unreclaimable. Waste dumps are surfaced with acceptable overburden materials by the spreader during the construction of the dumps, and therefore require only the application of muskeg, whereas tailings sand beaches require the application of both overburden and muskeg to form a prepared soil surface. The in-pit waste dump in the northern half of the ore body will be submerged by the final end-pit lake. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-39-00)

The extent of surface reclamation is shown in this drawing. Only the wet pond surfaces remain unreclaimed. A schedule for reclamation activities on a year-by-year basis is provided in Table 7.3.1-2. Plant species are selected according to the reclamation objectives for the Minimum Level as described in Chapter 4.0.

7.3.2 IMPROVED (DEWATERED) LEVEL OF RECLAMATION

The major aspects for the development of the 120,000 BPCD mine at the Improved Level are depicted by four drawings, accompanied by tailings disposal and reclamation schedules (See Tables 7.3.2-1 and 7.3.2-2). The mining schedule, which is common to the plans for all three levels of reclamation, has been illustrated previously in Table 7.3-1. A drawing-by-drawing discussion follows:

General Mine Layout
(Techman Drawing No. D22910-31-00)

The general layout of the mine at the Improved Level differs only with respect to the outside waste dump situated immediately south of the tailings pond. This waste dump is largely eliminated at the Minimum Level. Although the pit boundary remains the same, the internal arrangement of tailings and sludge ponds differs with respect to the plan at the Minimum Level.

Note: This drawing precedes the detailed plans for the Minimum Level of Reclamation - 3 BWE Mining Scheme.

Mining and Tailings Disposal - Year 19
(Techman Drawing No. D22918-40-00)

The mining sequence has been previously illustrated in the plans prepared for the Minimum Level. At the Improved Level, one out-of-pit and one in-pit tailings pond are utilized. The second in-pit pond stores partially dewatered sludge. The sludge treatment plant became operational in Year 17. Reclamation of the dyke slopes began as early as Year 4. By Year 19, the dyke slope (out-of-pit and in-pit) reclamation has been completed. Beach reclamation of the first in-pit pond began in Year 16. After the sludge has been removed from the out-of-pit tailings pond, reclamation of the inside beach slopes will be started in Year 21.

Ore Body No.2, 120,000 B.P.C.D.-3 Bucket Wheel Excavators
TAILINGS SCHEDULE
FOR IMPROVED LEVEL OF RECLAMATION
TABLE 7.3.2-1

Outline of Tailings Disposal Scheme:

- Years 1 - 11 Tailings to out-of-Pit Tailings Pond.
- Years 12-25 Tailings to First in-Pit Pond.
- Years 17-25 Simultaneous Pumping of Sludge from out-of-Pit Pond and First in-Pit Pond to Treatment Plant (50% of Water and Most of Remaining Bitumen Removed) Treated Sludge to Second in-Pit Pond.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge into Treatment [$m^3 \times 10^6$]
1	56.360	21.230	9.800	25.330	12.660	12.670	0
2	70.660	26.620	12.290	31.750	15.870	15.880	0
3	83.650	31.510	14.550	37.590	11.280	26.310	0
4	87.430	32.930	15.210	39.290	11.780	27.510	0
5	86.810	32.700	15.100	39.010	9.750	29.260	0
6	87.280	32.880	15.180	39.220	8.240	30.980	0
7	84.040	31.660	14.620	37.760	7.170	30.590	0
8	85.420	32.180	14.860	38.380	6.140	32.240	0
9	85.640	32.260	14.900	38.480	4.230	34.250	0
10	82.480	31.070	14.350	37.060	2.220	34.840	0
11	79.070	29.790	13.750	35.530	0.710	34.820	0
12	78.200	29.460	13.600	35.140	5.270	29.870	0
13	75.520	28.450	13.140	33.930	11.250	22.680	0
14	76.480	28.810	13.300	34.370	7.460	26.910	0
15	86.600	32.620	15.060	38.920	3.800	35.120	0
16	95.390	35.930	16.590	42.870	0	42.870	0
17	99.530	37.490	17.310	44.730	0	44.730	18.070
18	95.300	35.900	16.580	42.820	0	42.820	45.174
19	80.020	30.140	13.920	35.960	0	35.960	45.174
20	73.770	27.790	12.830	33.150	0	33.150	45.174
21	82.200	30.960	14.300	36.940	0	36.940	45.174
22	87.340	32.900	15.190	39.250	0	39.250	45.174
23	90.180	33.970	15.690	40.520	0	40.520	45.174
24	90.370	34.040	15.720	40.610	0	40.610	45.174
25	41.500	15.630	7.220	18.650	0	18.650	20.772
	2,041.240	768.920	355.060	917.260	117.830	799.430	355.060

Ore Body No. 2, 120,000 B.P.C.D.- 3 Bucket Wheel Excavators
SCHEDULE FOR IMPROVED LEVEL OF RECLAMATION
TABLE No. 7.3.2-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Alternating layers of muskeg and overburden are scraped at a sloping face of pile by dozers.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1							
2							
3							
4	325	325	6.7	108	0.9	217	1.0
5	325	325	6.7	108	0.9	217	1.0
6	325	325	6.7	108	0.9	217	1.0
7	325	325	6.7	108	0.9	217	1.0
8	325	325	6.7	108	0.9	217	1.0
9	325	325	6.7	108	0.9	217	1.0
10	325	325	6.7	108	0.9	217	1.0
11	325	325	6.7	108	0.9	217	1.0
12	605	605	6.7	202	0.9	403	1.0
13	605	605	6.7	202	0.9	403	1.0
14	605	605	6.7	202	0.9	403	1.0
15	913	913	3.0	304	0.8	609	0.8
16	1,115	1,115	3.0	372	0.8	743	0.8
17	1,115	1,115	3.0	372	0.8	743	0.8
18	1,115	1,115	3.0	372	0.8	743	0.8
19	1,115	1,115	3.0	372	0.8	743	0.8
20	1,115	1,115	3.0	372	0.8	743	0.8
21	1,319	1,319	4.5	440	0.9	879	0.9
22	1,319	1,319	5.5	440	0.9	879	0.9
23	1,319	1,319	5.5	440	0.9	879	0.9
24	1,319	1,319	5.5	440	0.9	879	0.9
25	1,378	1,378	5.6	459	0.9	919	0.9
26	1,378	1,378	5.6	459	0.9	919	0.9
27	1,865	1,865	6.7	622	0.9	1,243	1.0
28	1,865	1,865	5.4	622	0.8	1,243	1.0
29	1,809	1,809	2.9	603	0.8	1,206	0.6
30	1,809	1,809	2.7	603	0.8	1,206	0.6
	26,283	26,283		8,762		17,521	

Material Distribution Plan
(Techman Drawing No. D22916-21-00)

This plan shows the types of materials that must be surfaced with prepared soil. The out-of-pit tailings pond consists primarily of sloped sand surfaces. Some rehandle of tailings sand within the pond may be required to cover surfaces that are soft or toxic. Much of this type of remedial work may be done during the winter when the surface is frozen. The lake formed in the final end-pit will submerge the in-pit dump. The sludge pond surface is considered unreclaimable. All exposed pitwalls require application of prepared soil. Refer to Section 10.6, Table 10-6.1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-42-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurred are shown. Only the sludge pond remains wet and unreclaimable. Plant species are selected according to the reclamation objectives for the Improved Level as described in Chapter 4.0. Table 7.3.2-2 is a year-by-year summary of prepared soil manufacture and placement.

7.3.3 ENHANCED (DRY) LEVEL OF RECLAMATION

The major aspects of the development of the 120,000 BPCD mine at the Enhanced Level are depicted by eight drawings, accompanied by tailings disposal and reclamation schedules (See Tables 7.3.3-1 and 7.3.3-2). The mining schedule, which is common to the plans for all three levels of reclamation, has been illustrated previously in Table 7.3-1. A drawing-by-drawing discussion follows:

General Mine Layout
(Techman Drawing No. D22910-31-00)

The outside waste dump is situated in approximately the same location as the tailings pond for the Minimum and Improved Levels of Reclamation. The dump is located slightly more southward and a little closer to the pit. The pit outline remains as before, but the manner of development of the mining faces differs.

Mine Opening - Year 1
(Techman Drawing No. D22918-43-00)

The mine has been opened by developing a preproduction overburden bench prior to Year 1, with a slewing conveyor system utilized to deposit the waste into the outside dump. By the start of Year 1, the pit is sufficiently developed to allow the installation of the belt conveyor on the middle bench. The muskeg mine has been started.

Mining and Tailings Disposal - Year 7
(Techman Drawing No. D22918-44-00)

By Year 7, all the operating faces in the mine are fully developed. The outside dump is almost complete. Reclamation activities follow closely behind the spreader operating on the upper bench. Three bucket wheel excavators and two spreaders are in use.

Ore Body No. 2, 120,000 B.P.C.D. - 3 Bucket Wheel Excavators
TAILINGS SCHEDULE
FOR ENHANCED LEVEL OF RECLAMATION
TABLE 7.3.3-1

Outline of Tailings Disposal Scheme:
- Dry Tailings Conveyed with Overburden and Center Reject for First 8 Years to out-of-Pit Waste Dump, then into Mined-out Pit.

YEAR	Volume of Dry Tailings Produced [$m^3 \times 10^6$]	Dry Tailings Conveying Distance * [m]
1	26.186	4.9
2	32.827	11.9
3	38.857	11.7
4	40.617	11.7
5	40.329	11.9
6	40.549	11.9
7	39.042	11.9
8	39.681	11.9
9	39.782	7.4
10	38.316	7.4
11	36.736	7.9
12	36.333	7.9
13	35.082	7.6
14	35.528	7.6
15	40.235	4.7
16	44.316	6.1
17	46.240	6.1
18	44.270	7.1
19	37.173	4.7
20	34.269	7.9
21	38.189	9.2
22	40.580	10.7
23	41.896	11.1
24	41.986	11.9
25	19.281	11.9
	948.300	

* NOTE:
These are Total Lengths of Two Conveyor Systems from Plant to Two Spreaders. At the Same Time Overburden is Transported Via These Conveyors.

Ore Body No. 2, 120,000 B.P.C.D.- 3 Bucket Wheel Excavators
SCHEDULE FOR ENHANCED LEVEL OF RECLAMATION
TABLE No. 7.3.3-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Stacker deposits layers of Muskeg and overburden into piles. Components are mixed by bucket wheel reclaimer.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by conveyors) [km]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by pipeline) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by conveyors) [km]
1	1,142	1,142	4.5	2.0	381	8.3	761	4.6
2	1,142	1,142	4.5	1.8	381	8.3	761	5.2
3	1,142	1,142	4.5	1.6	381	8.3	761	5.2
4	1,142	1,142	4.5	1.1	381	8.3	761	5.2
5	1,142	1,142	4.5	1.1	381	8.3	761	5.1
6	1,142	1,142	4.5	0.9	381	8.3	761	4.9
7	1,142	1,142	4.5	0.9	381	8.3	761	4.6
8	1,142	1,142	4.5	0.8	381	8.3	761	3.3
9	1,142	1,142	4.5	0.8	381	8.3	761	2.6
10	1,142	1,142	4.5	1.1	381	8.3	761	2.6
11	1,142	1,142	4.5	1.2	381	8.3	761	3.3
12	1,142	1,142	4.5	1.4	381	8.3	761	3.9
13	1,142	1,142	4.5	1.3	381	8.3	761	3.9
14	1,142	1,142	4.5	1.5	381	8.3	761	5.3
15	1,142	1,142	1.5	0.9	381	8.3	761	3.3
16	1,142	1,142	1.5	0.9	381	8.3	761	3.4
17	1,142	1,142	1.5	0.8	381	8.3	761	3.3
18	1,142	1,142	1.5	0.8	381	8.3	761	3.4
19	1,142	1,142	1.5	0.9	381	8.3	761	1.9
20	1,142	1,142	1.5	0.8	381	8.3	761	2.5
21	1,142	1,142	1.5	1.0	381	8.3	761	2.5
22	1,142	1,142	3.0	0.4	381	8.3	761	3.4
23	1,142	1,142	3.0	0.4	381	8.3	761	3.9
24	1,142	1,142	4.0	0.3	381	8.3	761	4.1
25	1,142	1,142	4.0	2.2	381	8.3	761	1.4
26	1,140	1,140	4.0	2.2	380	8.3	760	1.4
27	1,140	1,140	1.5	0.8	380	8.3	760	1.4
28	1,140	1,140	1.5	1.2	380	8.3	760	1.4
29	1,140	1,140	1.5	0.7	380	8.3	760	1.4
30	0	0	0	0	0	0	0	0
	33,110	33,110			11,045		22,065	

Mining and Tailings Disposal - Year 12
(Techman Drawing No. D22918-45-00)

By Year 12, the outside waste dump has been completed and backfilling of the pit has begun. Backfilling has progressed sufficiently to allow about two years of reclamation activities to take place within the pit. The in-pit waste dump is slightly elevated above the surrounding landscape.

Mining and Tailings Disposal - Year 15
(Techman Drawing No. D22918-46-00)

The advance of faces in the mine to Year 15 is identical to that in the plans for the Minimum and Improved Levels of Reclamation, except that all the backfill is dry. In this drawing it is easily seen that the overburden and tailings conveyors are common. A short set of linking conveyors provides the connection between these conveyors and the main conveyor distribution point. Reclamation is very current and the hydraulic muskeg mine is nearly fifty percent developed.

Mining and Tailings Disposal - Year 21
(Techman Drawing No. 22918-47-00)

The conveyor distribution point has been relocated to a position north of the plant site. The connections between the main distribution point and the overburden tailings conveyors are similar to those used at the first distribution point. The northern half of the pit is being back-filled. A large trench-like trough is being created so that the face conveyors for the spreader can connect with the distribution point. The trench is filled with dry tailings sand and reject as the spreader retreats in the trench in the final years of mining. The backfilled trench is to be one of the last areas to be reclaimed. The trunk conveyors connecting the mining face conveyors with the distribution point are situated outside of the pit boundary. Reclamation is current and within a few hundred metres of the dumping faces.

Material Distribution Plan
(Techman Drawing No. D2916-48-00)

This plan shows the types of materials that must be surfaced with prepared soil, which in this case is a blend of overburden, reject, and dry tailings sand. The burial of undesirable overburden, reject, and tailings products ensures that the amounts of prepared soil required for reclamation are minimized. All dump surfaces are mildly sloped to provide drainage. The final end-pit and the muskeg mine are to be filled with water to form lakes. No unreclaimable areas exist in this plan. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-49-00)

The surfaces to be reclaimed as well as the period during which reclamation occurred are shown. Table 7.3.3-2 is a year-by-year schedule of the manufacture and placement of prepared soil. Plant species are selected according to the reclamation objectives for the Enhanced Level of Reclamation as described in Chapter 4.0.

7.4 COST SUMMARIES FOR 120,000 BPCD MINE PLANS

Cost summaries are provided for each of the mine plans detailed in this chapter. Rather than following immediately behind the description of the mine plan, the summary tables are grouped at this point in the report for ease of reading and comparison.

Six tables (Tables 7.4-1 to 7.4-6) summarize the quantities, unit costs and $/bbl costs of both capital and operating items. Further details for each cost summary are provided on an annual basis in Volume III. A comparison between the mine costed in this and other chapters of this report follows in Section 10.7.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), DRAGLINE SCHEME, MINIMUM LEVEL

TABLE 7.4-1

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0588
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0280	0.0280	-	-	-	0.0212	0.0212	0.0492
1.2 Buildings	6,349.0 persons		1,000.00 $/person	0.0062	-	0.0062	234.0 persons	15,000.00 $/person	0.0034	-	0.0034	0.0096
COST CENTRE 2: Removal of Organic Materials & Soils												0.0475
2.1 Clearing	3,748.7 hectares	-	1,266.82 $/ha	0.0046	-	0.0046	3,748.7 hectares	196.85 $/ha	0.0007	-	0.0007	0.0053
2.2 Muskeg Dewatering	917.8 1,000 bank m³	-	1,850.00 $/1,000 bank m³	0.0016	-	0.0016	917.8 1,000 bank m³	300.00 $/1,000 bank m³	0.0003	-	0.0003	0.0019
2.3 Muskeg Loading	17,149.3 1,000 bank m³	-	851.40 $/1,000 bank m³	0.0142	-	0.0142	17,149.3 1,000 bank m³	207.00 $/1,000 bank m³	0.0034	-	0.0034	0.0176
2.4 Muskeg Hauling (Including Road Maintenance)	17,149.3 1,000 bank m³	2.3 km	250.80 $/1,000 bank m³xkm	0.0096	-	0.0096	39,395.9 1,000 bank m³xkm	48.70 $/1,000 bank m³xkm	0.0019	-	0.0019	0.0115
2.5 Muskeg Placement	17,149.3 1,000 bank m³	-	479.40 $/1,000 bank m³	0.0080	-	0.0080	17,149.3 1,000 bank m³	34.60 $/1,000 bank m³	0.0006	-	0.0006	0.0086
2.6 Muskeg Road Construction	64.3 km	-	33,825.11 $/km	0.0021	-	0.0021	64.3 km	6,915.90 $/km	0.0004	-	0.0004	0.0025
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.3396
3.1 Overburden B.W.E.	333,838.0 1,000 bank m³	-	177.72 $/1,000 bank m³	0.0576	-	0.0576	-	-	-	0.0196	0.0196	0.0772
3.2 Oil Sands Draglines & Hoppers	1,162,065.0 1,000 bank m³	-	201.46 $/1,000 bank m³	0.2274	-	0.2274	-	-	-	0.1668	0.1668	0.3943
3.3 B.W.E. (Overburden & Oil Sands)	- 1,000 bank m³	-	- $/1,000 bank m³	-	-	-	-	-	-	-	-	-
3.4 Transport (All Conveyors)	1,397,446.0 1,000 bank m³	28,345.2 m	0.01 $/1,000 bank m³xkm	0.4550	-	0.4550	4[illegible],610.0 m	4,108.57 $/m	0.1621	-	0.1621	0.6171
3.5 Placement (Spreaders)	413,925.0 1,000 bank m³	-	103.48 $/1,000 bank m³	0.0416	-	0.0416	-	-	-	0.0161	0.0161	0.0577
3.6 Miscellaneous Equipment	-	-	-	-	0.1562	0.1562	-	-	-	0.0372	0.0372	0.1934
COST CENTRE 4: Tailings Disposal												0.4465
4.1 Area Drainage	163.0 1,000 bank m³	-	1,239.20 $/1,000 bank m³	0.0002	-	0.0002	163.0 1,000 bank m³	300.95 $/1,000 bank m³	0.00005	-	0.00005	0.0002
4.2 Clearing	1,375.0 hectares	-	1,266.82 $/ha	0.0017	-	0.0017	1,375.0 hectares	196.85 $/ha	0.0003	-	0.0003	0.0020
4.3 Construction of Starter Dams & Overburden Dams	21,820.0 1,000 bank m³	-	1,950.00 $/1,000 bank m³	0.0413	-	0.1413	- 1,000 bank m³	- $/1,000 bank m³	-	-	-	0.0413
4.4 Piping of Tailings or Conveying of Dry Tailings	2,227,806.0 1,000 m³	-	69.15 $/1,000 m³	0.1497	-	0.1497	-	-	-	0.0328	0.0328	0.1824
4.5 Tailings Sand Placement into Dyke	128,220.0 1,000 m³	-	123.56 $/1,000 m³	0.0154	-	0.0154	-	-	-	0.0216	0.0216	0.0370
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	872,611.0 1,000 m³	-	67.89 $/1,000 m³	0.0576	-	0.0576	-	-	-	-	-	0.0576
4.7 Recycling of Tailings Water	-	-	-	-	0.0255	0.0255	-	-	-	0.0095	0.0095	0.0350
4.8 Rehandling of Tailings Sludge	388,032.0 1,000 m³	-	31.90 $/1,000 m³	0.0120	-	0.0120	-	-	-	0.0094	0.0094	0.0215

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), DRAGLINE SCHEME, MINIMUM LEVEL (Continued)

TABLE 7.4-1 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-		15.0 km	25,000.00 $/km	0.0004	0.0005	0.0009	0.0009
4.11 Oversize Reject Disposal	56,059.0 1,000 loose m^3	-	1,066.63 $/1,000 loose m^3	0.0581	-	0.0581	56,059.0 1,000 loose m^3	191.69 $/1,000 loose m^3	0.0104	-	0.0104	0.0685
4.12 Oversize Reject Disposal Road Construction	3.0 km	-	33,825.11 $/km	0.0001	-	0.0001	3.0 km	6,915.90 $/km	0.00002	-	0.00002	0.0001
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.0501
5.1 Muskeg Rehandle Loading	4,755.0 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0028	-	0.0028	4,755.0 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0007	-	0.0007	0.0035
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	4,755.0 1,000 bank m^3	2.5 km	224.53 $/1,000 bank m^3xkm	0.0026	-	0.0026	11,898.6 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0005	-	0.0005	0.0031
5.3 Muskeg Rehandle Placement	4,755.0 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0008	0.0021	0.0029	4,755.0 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0006	-	0.0006	0.0035
5.4 Muskeg Rehandle Road Construction	176.2 km	-	33,825.11 $/km	0.0058	-	0.0058	176.2 km	6,915.90 $/km	0.0012	-	0.0012	0.0070
5.5 Overburden Rehandle Loading	6,412.0 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0034	-	0.0034	6,412.0 1,000 bank m^3	131.53 $1,000 bank m^3	0.0008	-	0.0008	0.0042
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	6,412.0 1,000 bank m^3	2.81 km	178.56 $/1,000 bank m^3xkm	0.0031	-	0.0031	18,025.6 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0006	-	0.0006	0.0037
5.7 Overburden Rehandle Placement	6,412.0 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0009	-	0.0009	6,412.0 1,000 bank m^3	22.92 $1,000 bank m^3	0.0001	-	0.0001	0.0010
5.8 Overburden Rehandle Road Construction	13.4 km	-	33,825.11 $/km	0.0004	-	0.0004	13.4 km	6,915.90 $/km	0.0001	-	0.0001	0.0005
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	22,044.0 1,000 bank m^3	-	27.85 $/1,000 bank m^3	0.0006	-	0.0006	22,044.0 1,000 bank m^3	4.36 $/1,000 bank m^3	0.0001	-	0.0001	0.0007
5.11 Prepared Soil Loading, F.E.L. & Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.13 Prepared Soil Placement, Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.14 Prepared Soil Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.15 Seed Bed Preparation, Maintenance	3,674.0 hectares	-	768.05 $/ha	0.0027	0.0201	0.0228	- hectares	- $/ha	-	-	-	0.0228
COST CENTRE 6: Supervision, Technical Services												0.1870
6.1 Equipment Maintenance (Staff only)	2,383.0 persons	-	29,970.00 $/person	0.0694	-	0.0694	-	-	-	-	-	0.0694
6.2 Planning (Staff only)	2,170.0 persons	-	29,605.00 $/person	0.0624	-	0.0624	-	-	-	-	-	0.0624
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.0552	-	0.0552	-	-	-	-	-	0.0552
TOTAL COSTS						1.6061					0.5233	2.1294

NOTE: Refer to Chapter 6 for Cost Centre Description.

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COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), DRAGLINE SCHEME, IMPROVED LEVEL

TABLE 7.4-2

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0588
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0280	0.0280	-	-	-	0.0212	0.0212	0.0492
1.2 Buildings	6,349.0 persons	-	1,000.00 $/person	0.0062	-	0.0062	234.0 persons	15,000.00 $/person	0.0034	-	0.0034	0.0096
COST CENTRE 2: Removal of Organic Materials & Soils												0.0455
2.1 Clearing	3,748.7 hectares	-	1,266.82 $/ha	0.0046	-	0.0046	3,748.7 hectares	196.85 $/ha	0.0007	-	0.0007	0.0053
2.2 Muskeg Dewatering	917.8 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0016	-	0.0016	917.8 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0003	-	0.0003	0.0019
2.3 Muskeg Loading	17,149.3 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0142	-	0.0142	17,149.3 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0034	-	0.0034	0.0176
2.4 Muskeg Hauling (Including Road Maintenance)	17,149.3 1,000 bank m^3	1.89 km	250.80 $/1,000 bank m^3xkm	0.0079	-	0.0079	32,429.4 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0015	-	0.0015	0.0094
2.5 Muskeg Placement	17,149.3 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0080	-	0.0080	17,149.3 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0006	-	0.0006	0.0086
2.6 Muskeg Road Construction	66.3 km	-	33,825.11 $/km	0.0022	-	0.0022	66.3 km	6,915.90 $/km	0.0004	-	0.0004	0.0026
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.3434
3.1 Overburden B.W.E.	333,838.0 1,000 bank m^3	-	177.72 $/1,000 bank m^3	0.0576	-	0.0576	-	-	-	0.0196	0.0196	0.0772
3.2 Oil Sands Draglines & Hoppers	1,162,065.0 1,000 bank m^3	-	201.46 $/1,000 bank m^3	0.2274	-	0.2274	-	-	-	0.1668	0.1668	0.3943
3.3 B.W.E. (Overburden & Oil Sands)	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	
3.4 Transport (All Conveyors)	1,397,446.0 1,000 bank m^3	28,859.3 m	0.01 $/1,000 bank m^3xkm	0.4447	-	0.4447	43,660.0 m	4,152.39 $/m	0.1761	-	0.1761	0.6209
3.5 Placement (Spreaders)	413,925.0 1,000 bank m^3	-	103.48 $/1,000 bank m^3	0.0416	-	0.0416	-	-	-	0.0161	0.0161	0.0577
3.6 Miscellaneous Equipment	-	-	-	-	0.1562	0.1562	-	-	-	0.0372	0.0372	0.1934
COST CENTRE 4: Tailings Disposal												1.2945
4.1 Area Drainage	163.0 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	163.0 1,000 bank m^3	300.95 $/1,000 bank m^3	0.00005	-	0.00005	0.0002
4.2 Clearing	1,375.0 hectares	-	1,266.82 $/ha	0.0017	-	0.0017	1,375.0 hectares	196.85 $/ha	0.0003	-	0.0003	0.0020
4.3 Construction of Starter Dams & Overburden Dams	129,475.0 1,000 bank m^3	-	1,406.39 $/1,000 bank m^3	0.1769	-	0.1769	107,655.0 1,000 bank m^3	286.06 $/1,000 bank m^3	0.0299	-	0.0299	0.2068
4.4 Piping of Tailings or Conveying of Dry Tailings	2,227,806.0 1,000 m^3	-	75.73 $/1,000 m^3	0.1639	-	0.1639	-	-	-	0.0331	0.0331	0.1970
4.5 Tailings Sand Placement into Dyke	79,060.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0095	-	0.0095	-	-	-	0.0243	0.0243	0.0338
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	921,771.0 1,000 m^3	-	67.89 $/1,000 m^3	0.0608	-	0.0608	-	-	-	-	-	0.0608
4.7 Recycling of Tailings Water	-	-	-	-	0.0251	-	-	-	-	0.0081	0.0081	0.0333
4.8 Rehandling of Tailings Sludge	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), DRAGLINE SCHEME, IMPROVED LEVEL (Continued)

TABLE 7.4-2 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	388,032.0 1,000 m^3	-	1,717.42 $/1,000 m^3	0.6474	-	0.6474	-	-	-	0.0438	0.0438	0.6912
4.10 Power Distribution	-	-	-	-	-	-	14.6 km	25,000.00 $/km	0.0004	0.0003	0.0008	0.0007
4.11 Oversize Reject Disposal	56,059.0 1,000 loose m^3	-	1,066.63 $/1,000 loose m^3	0.0581	-	0.0581	56,059.0 1,000 loose m^3	191.69 $/1,000 loose m^3	0.0104	-	0.0104	0.0685
4.12 Oversize Reject Disposal Road Construction	3.0 km	-	33,825.11 $/km	0.0001	-	0.0001	3.0 km	6,915.90 $/km	0.00002	-	0.00002	0.0001
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.1273
5.1 Muskeg Rehandle Loading	11,226.2 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0067	-	0.0067	11,226.2 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0016	-	0.0015	0.0083
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	11,226.2 1,000 bank m^3	2.23 km	224.53 $/1,000 bank m^{3}xkm	0.0055	-	0.0055	25,072.4 1,000 bank m^{3}xkm	42.43 $/1,000 bank m^{3}xkm	0.0010	-	0.0010	0.0065
5.3 Muskeg Rehandle Placement	11,226.2 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0018	-	0.0018	11,226.2 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0003	-	0.0003	0.0021
5.4 Muskeg Rehandle Road Construction	17.6 km	-	33,825.11 $/km	0.0006	-	0.0006	17.6 km	5,915.90 $/km	0.0001	-	0.0001	0.0007
5.5 Overburden Rehandle Loading	22,454.1 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0118	-	0.0118	22,454.1 1,000 bank m^3	131.53 $1,000 bank m^3	0.0029	-	0.0029	0.0147
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	22,454.1 1,000 bank m^3	1.35 km	178.56 $/1,000 bank m^{3}xkm	0.0053	-	0.0053	30,425.2 1,000 bank m^{3}xkm	34.00 $/1,000 bank m^{3}xkm	0.0010	-	0.0010	0.0063
5.7 Overburden Rehandle Placement	22,454.1 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0031	-	0.0031	22,454.1 1,000 bank m^3	22.92 $1,000 bank m^3	0.0005	-	0.0005	0.0036
5.8 Overburden Rehandle Road Construction	9.0 km	-	33,825.11 $/km	0.0003	-	0.0003	9.0 km	5,915.90 $/km	0.0001	-	0.0001	0.0004
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	33,600.0 1,000 bank m^3	-	211.57 $/1,000 bank m^3	0.0069	-	0.0069	33,600.0 1,000 bank m^3	47.66 $/1,000 bank m^3	0.0016	-	0.0016	0.0085
5.11 Prepared Soil Loading, F.E.L. & Trucks	33,600.0 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0140	-	0.0140	[illegible] 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0038	-	0.0038	0.0178
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	33,600.0 1,000 bank m^3	3.44 km	212.40 $/1,000 bank m^{3}xkm	0.0238	-	0.0238	115,539.0 1,000 bank m^{3}xkm	34.30 $/1,000 bank m^{3}xkm	0.0039	-	0.0039	0.0277
5.13 Prepared Soil Placement, Trucks	33,600.0 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0049	-	0.0049	33,600.0 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0008	-	0.0008	0.0057
5.14 Prepared Soil Road Construction	199.0 km	-	33,825.11 $/km	0.0002	-	0.0065	199.0 km	6,519.90 $/km	0.0013	-	0.0013	0.0079
5.15 Seed Bed Preparation, Maintenance	3,360.0 hectares	-	768.05 $/ha	0.0025	0.0148	0.0173	- hectares	- $/ha	-	-	-	0.0173
COST CENTRE 6: Supervision, Technical Services												0.1870
6.1 Equipment Maintenance (Staff Only)	2,383.0 persons	-	29,970.00 $/person	0.0694	-	0.0694	-	-	-	-	-	0.0694
6.2 Planning (Staff Only)	2,170.0 persons	-	29,605.00 $/person	0.0624	-	0.0624	-	-	-	-	-	0.0624
6.3 Mining (Staff Only)	1,891.0 persons	-	30,065.00 $/person	0.0552	-	0.0552	-	-	-	-	-	0.0552
TOTAL COSTS						2.4396					0.6169	3.0565

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), DRAGLINE SCHEME, ENHANCED LEVEL

TABLE 7.4-3

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0699
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0314	0.0314	-	-	-	0.0287	0.0287	0.0601
1.2 Buildings	6,464.0 persons	-	1,000.00 $/person	0.0063	-	0.0063	238.0 persons	15,000.00 $/person	0.0035	-	0.0035	0.0097
COST CENTRE 2: Removal of Organic Material & Soils												0.0455
2.1 Clearing	3,748.7 hectares	-	1,266.82 $/ha	0.0046	-	0.0046	3,748.7 hectares	196.85 $/ha	0.0007	-	0.0007	0.0053
2.2 Muskeg Dewatering	917.8 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0016	-	0.0016	917.8 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0003	-	0.0003	0.0019
2.3 Muskeg Loading	17,149.3 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0142	-	0.0142	17,149.3 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0034	-	0.0034	0.0176
2.4 Muskeg Hauling (Including Road Maintenance)	17,149.3 1,000 bank m^3	1.89 km	250.80 $/1,000 bank m^3xkm	0.0079	-	0.0079	32,429.4 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0015	-	0.0015	0.0094
2.5 Muskeg Placement	17,149.3 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0080	-	0.0080	17,149.3 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0006	-	0.0006	0.0086
2.6 Muskeg Road Construction	66.3 km	-	33,825.11 $/km	0.0022	-	0.0022	66.3 km	6,915.90 $/km	0.0004	-	0.0004	0.0026
COST CENTRE 3: Overburden, Reject, Oil Sands Handling and Tailings Disposal												1.6506
3.1 Overburden B.W.E.	333,838.0 1,000 bank m^3	-	177.72 $/1,000 bank m^3	0.0576	-	0.0576	-	-	-	0.0196	0.0196	0.0772
3.2 Oil Sands Draglines & Hoppers	1,162,065.0 1,000 bank m^3	-	201.46 $/1,000 bank m^3	0.2274	-	0.2274	-	-	-	0.1668	0.1668	0.3943
3.3 B.W.E. (Overburden & Oil Sands)	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.4 Transport (All Conveyors)	2,427,530.0 1,000 bank m^3	39,528.65 m	0.01 $/1,000 bank m^3xkm	0.6021	-	0.6021	61,100.0 m	4,424.17 $/m	0.2626	-	0.2626	0.8647
3.5 Placement (Spreaders)	1,448,589.0 1,000 bank m^3	-	56.80 $/1,000 bank m^3	0.0799	-	0.0799	-	-	-	0.0321	0.0321	0.1121
3.6 Miscellaneous Equipment	-	-	-	-	0.1641	0.1641	-	-	-	-	0.0383	0.2024
COST CENTRE 4: Tailings Disposal - Included in Cost Centre 3												
4.1 Area Drainage	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
4.2 Clearing	- hectares	-	- $/ha	-	-	-	- hectares	- $/ha	-	-	-	-
4.3 Construction of Starter Dams & Overburden Dams	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
4.4 Piping of Tailings or Conveying of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.5 Tailings Sand Placement into Dyke	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.7 Recycling of Tailings Water	-	-	-	-	-	-	-	-	-	-	-	-
4.8 Rehandling of Tailings Sludge	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCO), DRAGLINE SCHEME, ENHANCED LEVEL (Continued)

TABLE 7.4-3 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	- km	- $/km	-	-	-	-
4.11 Oversize Reject Disposal	- 1,000 loose m^3	-	- $/1,000 loose m^3	-	-	-	- 1,000 loose m^3	- $/1,000 loose m^3	-	-	-	-
4.12 Oversize Reject Disposal Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.1712
5.1 Muskeg Rehandle Loading	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.3 Muskeg Rehandle Placement	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.4 Muskeg Rehandle Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.5 Overburden Rehandle Loading	5,120.2 1,000 bank m^3	-	377.35 $/1,000 bank m^3	0.0019	-	0.0019	5,120.2 1,000 bank m^3	42.35 $1,000 bank m^3	0.0002	-	0.0002	0.0021
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.7 Overburden Rehandle Placement	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $1,000 bank m^3	-	-	-	-
5.8 Overburden Rehandle Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	
5.9 Muskeg Mining, Slurry Transport and Dewatering	9,514.0 1,000 bank m^3	-	2,944.51 $/1,000 bank m^3	0.0272	-	0.0272	- 1,000 bank m^3	- $/1,000 bank m^3	-	0.0100	0.0100	0.0373
5.10 Prepared Soil Manufacture	28,552.0 1,000 bank m^3	-	1,590.18 $/1,000 bank m^3	0.0441	-	0.0441	- 1,000 bank m^3	- $/1,000 bank m^3	-	0.0295	0.0295	0.0736
5.11 Prepared Soil Loading, F.E.L. & Trucks	25,552.0 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0107	-	0.0107	25,552.0 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0029	-	0.0029	0.0135
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	25,552.0 1,000 bank m^3	1.87 km	212.40 $/1,000 bank m^3xkm	0.0098	-	0.0098	47,726.9 1,000 bank m^3xkm	59.02 $/1,000 bank m^3xkm	0.0027	-	0.0027	0.0126
5.13 Prepared Soil Placement, Trucks	25,552.0 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0037	-	0.0037	25,552.0 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0006	-	0.0006	0.0043
5.14 Prepared Soil Road Construction	155.3 km	-	33,825.11 $/km	0.0051	-	0.0051	155.3 km	6,915.90 $/km	0.0010	-	0.0010	0.0061
5.15 Seed Bed Preparation, Maintenance	2,855.2 hectares	-	768.05 $/ha	0.0021	0.0195	0.0217	- hectares	- $/ha	-	-	-	0.0217
COST CENTRE 6: Supervision, Technical Services												0.1904
6.1 Equipment Maintenance (Staff only)	2,383.0 persons	-	29,970.00 $/person	0.0694	-	0.0694	-	-	-	-	-	0.0694
6.2 Planning (Staff only)	2,170.0 persons	-	29,605.00 $/person	0.0624	-	0.0624	-	-	-	-	-	0.0624
6.3 Mining (Staff only)	2,008.0 persons	-	30,065.00 $/person	0.0587	-	0.0587	-	-	-	-	-	0.0587
TOTAL COSTS						1.5221					0.6055	2.1276

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), B.W.E. SCHEME, MINIMUM LEVEL

TABLE 7.4-4

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0436
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0208	0.0208	-	-	-	0.0128	0.0128	0.0336
1.2 Buildings	6,349.0 persons	-	1,000.00 $/person	0.0065	-	0.0065	234.0 persons	15,000.00 $/person	0.0036	-	0.0036	0.0100
COST CENTRE 2: Removal of Organic Materials & Soils												0.0464
2.1 Clearing	3,897.8 hectares	-	1,266.82 $/ha	0.0050	-	0.0050	3,897.8 hectares	196.85 $/ha	0.0008	-	0.0008	0.0058
2.2 Muskeg Dewatering	798.6 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0015	-	0.0015	798.6 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0002	-	0.0002	0.0017
2.3 Muskeg Loading	14,500.0 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0126	-	0.0126	14,500.0 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0031	-	0.0031	0.0156
2.4 Muskeg Hauling (Including Road Maintenance)	14,500.0 1,000 bank m^3	2.46 km	250.80 $/1,000 bank m^3xkm	0.0091	-	0.0091	35,674.0 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0018	-	0.0018	0.0109
2.5 Muskeg Placement	14,500.0 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0071	-	0.0071	14,500.0 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0005	-	0.0005	0.0076
2.6 Muskeg Road Construction	115.3 km	-	33,825.11 $/km	0.0040	-	0.0040	115.3 km	6,915.90 $/km	0.0008	-	0.0008	0.0048
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												0.9026
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	1,380,197.0 1,000 bank m^3	-	151.48 $/1,000 bank m^3	0.2128	-	0.2128	-	-	-	0.1081	0.1081	0.3209
3.4 Transport (All Conveyors)	1,380,197.0 1,000 bank m^3	16,611.88 m	0.01 $/1,000 bank m^3xkm	0.2470	-	0.2470	25,570.0 m	4,563.59 $/m	0.1188	-	0.1188	0.3658
3.5 Placement (Spreaders)	478,997.0 1,000 bank m^3	-	90.21 $/1,000 bank m^3	0.0440	-	0.0440	-	-	-	0.0168	0.0168	0.0608
3.6 Miscellaneous Equipment	-	-	-	-	0.1263	0.1263	-	-	-	0.0288	0.0288	0.1551
COST CENTRE 4: Tailings Disposal												0.4468
4.1 Area Drainage	163.0 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	163.0 1,000 bank m^3	300.95 $/1,000 bank m^3	0.00005	-	0.00005	0.0003
4.2 Clearing	1,404.9 hectares	-	1,266.82 $/ha	0.0018	-	0.0018	1,404.9 hectares	196.85 $/ha	0.0003	-	0.0003	0.0021
4.3 Construction of Starter Dams & Overburden Dams	65,820.0 1,000 bank m^3	-	986.87 $/1,000 bank m^3	0.0661	-	0.0661	44,003.0 1,000 bank m^3	115.30 $/1,000 bank m^3	0.0052	-	0.0052	0.0713
4.4 Piping of Tailings or Conveying of Dry Tailings	2,041,180.0 1,000 m^3	-	68.74 $/1,000 m^3	0.1428	-	0.1428	-	-	-	0.0323	0.0323	0.1750
4.5 Tailings Sand Placement into Dyke	159,390.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0200	-	0.0200	-	-	-	0.0197	0.0197	0.0398
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	757,870.0 1,000 m^3	-	67.88 $/1,000 m^3	0.0524	-	0.0524	-	-	-	-	-	0.0524
4.7 Recycling of Tailings Water	-	-	-	-	0.0213	0.0213	-	-	-	0.0072	0.0072	0.0285
4.8 Rehandling of Tailings Sludge	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), B.W.E. SCHEME, MINIMUM LEVEL (Continued)

TABLE 7.4-4 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	12.6 km	25,000.00 $/km	0.0003	0.0001	0.0004	0.0004
4.11 Oversize Reject Disposal	51,366.0 1,000 loose m^3	-	1,221.82 $/1,000 loose m^3	0.0639	-	0.0639	51,366.0 1,000 loose m^3	222.67 $/1,000 loose m^3	0.0116	-	0.0116	0.0755
4.12 Oversize Reject Disposal Road Construction	35.8 km	-	33,825.11 $/km	0.0012	-	0.0012	35.8 km	6,915.90 $/km	0.0003	-	0.0003	0.0015
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.0253
5.1 Muskeg Rehandle Loading	2,130.0 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0013	-	0.0013	2,130.0 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0003	-	0.0003	0.0017
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	2,130.0 1,000 bank m^3	3.17 km	224.53 $/1,000 bank m^3xkm	0.0015	-	0.0015	6,755.0 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0003	-	0.0003	0.0018
5.3 Muskeg Rehandle Placement	2,130.0 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0004	-	0.0004	2,130.0 1,000 bank m^3	26.62 $/1,000 bank m^3	0.00006	-	0.00006	0.0004
5.4 Muskeg Rehandle Road Construction	27.7 km	-	33,825.11 $/km	0.0010	-	0.0010	27.7 km	6,915.90 $/km	0.0002	-	0.0002	0.0011
5.5 Overburden Rehandle Loading	3,920.0 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0022	-	0.0022	3,920.0 1,000 bank m^3	131.53 $1,000 bank m^3	0.0005	-	0.0005	0.0027
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	3,920.0 1,000 bank m^3	8.31 km	178.56 $/1,000 bank m^3xkm	0.0059	-	0.0059	32,560.0 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0011	-	0.0011	0.0070
5.7 Overburden Rehandle Placement	3,920.0 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0006	-	0.0006	3,920.0 1,000 bank m^3	22.92 $1,000 bank m^3	0.0001	-	0.0001	0.0007
5.8 Overburden Rehandle Road Construction	3.0 km	-	33,825.11 $/km	0.0001	-	0.0001	3.0 km	6,915.90 $/km	0.00002	-	0.00002	0.0001
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	9,620.0 1,000 bank m^3	-	27.85 $/1,000 bank m^3	0.0003	-	0.0003	9,620.0 1,000 bank m^3	4.36 $/1,000 bank m^3	0.00004	-	0.00004	0.0003
5.11 Prepared Soil Loading, F.E.L. & Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.13 Prepared Soil Placement, Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.14 Prepared Soil Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.15 Seed Bed Preparation, Maintenance	1,630.0 hectares	-	768.05 $/ha	0.0013	0.0081	0.0094	- hectares	- $/ha	-	-	-	0.0094
COST CENTRE 6: Supervision, Technical Services												0.1959
6.1 Equipment Maintenance (Staff only)	2,383.0 persons	-	29,770.00 $/person	0.0727	-	0.0727	-	-	-	-	-	0.0727
6.2 Planning (Staff only)	2,170.0 persons	-	29,605.00 $/person	0.0654	-	0.0654	-	-	-	-	-	0.0654
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.0579	-	0.0579	-	-	-	-	-	0.0579
TOTAL COSTS						1.2847					0.3759	1.6606

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), B.W.E. SCHEME, IMPROVED LEVEL

TABLE 7.4-5

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0436
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0208	0.0208	-	-	-	0.0128	0.0128	0.0336
1.2 Buildings	6,349.0 persons	-	1,000.00 $/person	0.0065	-	0.0065	234.0 persons	15,000.00 $/person	0.0036	-	0.0036	0.0100
COST CENTRE 2: Removal of Organic Materials & Soils												0.0464
2.1 Clearing	3,897.8 hectares	-	1,266.82 $/ha	0.0050	-	0.0050	3,897.8 hectares	196.85 $/ha	0.0008	-	0.0008	0.0058
2.2 Muskeg Dewatering	798.6 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0015	-	0.0015	798.6 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0002	-	0.0002	0.0017
2.3 Muskeg Loading	14,500.0 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0126	-	0.0126	14,500.0 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0031	-	0.0031	0.0156
2.4 Muskeg Hauling (Including Road Maintenance)	14,500.0 1,000 bank m^3	3.05 km	250.80 $/1,000 bank m^3xkm	0.0113	-	0.0113	44,213.0 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0022	-	0.0022	0.0135
2.5 Muskeg Placement	14,500.0 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0071	-	0.0071	14,500.0 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0005	-	0.0005	0.0076
2.6 Muskeg Road Construction	51.9 km	-	33,825.11 $/km	0.0018	-	0.0018	51.9 km	6,915.90 $/km	0.0004	-	0.0004	0.0022
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												0.9026
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	1,380,197.0 1,000 bank m^3	-	151.48 $/1,000 bank m^3	0.2128	-	0.2128	-	-	-	0.1081	0.1081	0.3209
3.4 Transport (All Conveyors)	1,380,197.0 1,000 bank m^3	16,611.88 m	0.01 $/1,000 bank m^3xkm	0.2470	-	0.2470	25,570.0 m	4,563.59 $/m	0.1188	-	0.1188	0.3658
3.5 Placement (Spreaders)	478,997.0 1,000 bank m^3	-	90.21 $/1,000 bank m^3	0.0440	-	0.0440	-	-	-	0.0168	0.0168	0.0608
3.6 Miscellaneous Equipment	-	-	-	-	0.1263	0.1263	-	-	-	0.0288	0.0288	0.1551
COST CENTRE 4: Tailings Disposal											-	1.1110
4.1 Area Drainage	163.0 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	163.0 1,000 bank m^3	300.95 $/1,000 bank m^3	0.00005	-	0.00005	0.0003
4.2 Clearing	1,404.9 hectares	-	1,266.82 $/ha	0.0018	-	0.0018	1,404.9 hectares	196.85 $/ha	0.0003	-	0.0003	0.0021
4.3 Construction of Starter Dams & Overburden Dams	63,320.0 1,000 bank m^3	-	1,025.30 $/1,000 bank m^3	0.0661	-	0.0661	40,640.0 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0048	-	0.0048	0.0708
4.4 Piping of Tailings or Conveying of Dry Tailings	2,041,240.0 1,000 m^3	-	66.63 $/1,000 m^3	0.1384	-	0.1384	-	-	-	0.0323	0.0323	0.1707
4.5 Tailings Sand Placement into Dyke	117,841.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0148	-	0.0148	-	-	-	0.0197	0.0197	0.0346
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	799,424.0 1,000 m^3	-	67.88 $/1,000 m^3	0.0552	-	0.0552	-	-	-	-	-	0.0552
4.7 Recycling of Tailings Water	-	-	-	-	0.0222	0.0222	-	-	-	0.0072	0.0072	0.0294
4.8 Rehandling of Tailings Sludge	-	-	-	-	-		-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), B.W.E. SCHEME, IMPROVED LEVEL (Continued)

TABLE 7.4-5 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	355,068.0 1,000 m^3	-	1,655.58 $/1,000 m^3	0.5982	-	0.5982	-	-	-	0.0721	0.0721	0.6703
4.10 Power Distribution	-	-	-	-	-	-	9.8 km	25,000.00 $/km	0.0002	0.0004	0.0007	0.0007
4.11 Oversize Reject Disposal	51,366.0 1,000 loose m^3	-	1,221.82 $/1,000 loose m^3	0.0639	-	0.0639	51,366.0 1,000 loose m^3	222.67 $/1,000 loose m^3	0.0116	-	0.0116	0.0755
4.12 Oversize Reject Disposal Road Construction	35.8 km	-	33,825.11 $/km	0.0012	-	0.0012	35.8 km	6,915.90 $/km	0.0003	-	0.0003	0.0015
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.1214
5.1 Muskeg Rehandle Loading	8,761.1 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0055	-	0.0055	8,761.1 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0013	-	0.0013	0.0068
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	8,761.1 1,000 bank m^3	0.86 km	224.53 $/1,000 bank m^3xkm	0.0017	-	0.0017	7,551.7 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0003	-	0.0003	0.0021
5.3 Muskeg Rehandle Placement	8,761.1 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0015	-	0.0015	8,761.1 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0002	-	0.0002	0.0017
5.4 Muskeg Rehandle Road Construction	4.3 km	-	33,825.11 $/km	0.0001	-	0.0001	4.3 km	6,915.90 $/km	0.00003	-	0.00003	0.0002
5.5 Overburden Rehandle Loading	17,521.9 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0097	-	0.0097	17,521.9 1,000 bank m^3	131.53 $1,000 bank m^3	0.0023	-	0.0023	0.0120
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	17,521.9 1,000 bank m^3	0.86 km	178.56 $/1,000 bank m^3xkm	0.0028	-	0.0028	15,134.8 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0005	-	0.0005	0.0033
5.7 Overburden Rehandle Placement	17,521.9 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0025	-	0.0025	17,521.9 1,000 bank m^3	22.92 $1,000 bank m^3	0.0004	-	0.0004	0.0029
5.8 Overburden Rehandle Road Construction	10.5 km	-	33,825.11 $/km	0.0004	-	0.0004	10.5 km	6,915.90 $/km	0.0001	-	0.0001	0.0004
5.9 Muskeg Mining, Slurry Transport and Dewatering	1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	26,283.0 1,000 bank m^3	-	211.57 $/1,000 bank m^3	0.0057	-	0.0057	26,283.0 1,000 bank m^3	47.66 $/1,000 bank m^3	0.0013	-	0.0013	0.0069
5.11 Prepared Soil Loading, F.E.L. & Trucks	26,283.0 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0115	-	0.0115	26,283.0 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0031	-	0.0031	0.0146
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	26,283.0 1,000 bank m^3	4.74 km	212.40 $/1,000 bank m^3xkm	0.0269	-	0.0269	124,631.4 1,000 bank m^3xkm	131.26 $/1,000 bank m^3xkm	0.0166	-	0.0166	0.0436
5.13 Prepared Soil Placement, Trucks	26,293.0 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0040	-	0.0040	26,283.0 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0007	-	0.0007	0.0047
5.14 Prepared Soil Road Construction	158.8 km	-	33,825.11 $/km	0.0055	-	0.0055	158.8 km	6,915.90 $/km	0.0011	-	0.0011	0.0066
5.15 Seed Bed Preparation, Maintenance	2,628.3 hectares	-	768.05 $/ha	0.0021	0.0136	0.0156	- hectares	- $/ha	-	-	-	0.0156
COST CENTRE 6: Supervision, Technical Services												0.1959
6.1 Equipment Maintenance (Staff only)	2,383.0 persons	-	29,970.00 $/person	0.0727	-	0.0727	-	-	-	-	-	0.0727
6.2 Planning (Staff only)	2,170.0 persons	-	29,605.00 $/person	0.0654	-	0.0654	-	-	-	-	-	0.0654
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.0579	-	0.0579	-	-	-	-	-	0.0579
TOTAL COSTS						1.9478					0.4731	2.4209

NOTE: Refer to Chapter 6 for Cost Centre Description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), B.W.E. SCHEME, ENHANCED LEVEL

TABLE 7.4-6

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0530
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0244	0.0244	-	-	-	0.0184	0.0184	0.0428
1.2 Buildings	6,464.0 persons	-	1,000.00 $/person	0.0066	-	0.0066	238.0 persons	15,000.00 $/person	0.0036	-	0.0036	0.0102
COST CENTRE 2: Removal of Organic Materials & Soils												0.0405
2.1 Clearing	3,897.8 hectares	-	1,266.82 $/ha	0.0050	-	0.0050	3,897.8 hectares	196.85 $/ha	0.0008	-	0.0008	0.0058
2.2 Muskeg Dewatering	798.6 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0015	-	0.0015	798.6 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0002	-	0.0002	0.0017
2.3 Muskeg Loading	14,500.0 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0126	-	0.0126	14,500.0 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0031	-	0.0031	0.0156
2.4 Muskeg Hauling (Including Road Maintenance)	14,500.0 1,000 bank m^3	1.71 km	250.80 $/1,000 bank m^3xkm	0.0063	-	0.0063	24,805.0 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0012	-	0.0012	0.0076
2.5 Muskeg Placement	14,500.0 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0071	-	0.0071	14,500.0 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0005	-	0.0005	0.0076
2.6 Muskeg Road Construction	51.90 km	-	33,825.11 $/km	0.0018		0.0018	51.90 km	6,915.90 $/km	0.0004	-	0.0004	0.0022
COST CENTRE 3: Overburden, Reject, Oil Sands Handling, and Tailings Disposal												1.1236
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	1,380,197.0 1,000 bank m^3	-	151.48 $/1,000 bank m^3	0.2128	-	0.2128	-	-	-	0.1081	0.1081	0.3209
3.4 Transport (All Conveyors)	2,328,497.0 1,000 bank m^3	24,775.04 m	0.01 $/1,000 bank m^3xkm	0.3376	-	0.3376	38,430.0 m	4,655.8 $/m	0.1821	-	0.1821	0.5197
3.5 Placement (Spreaders)	1,427,297.0 1,000 bank m^3	-	58.37 $/1,000 bank m^3	0.0848	-	0.0848	-	-	-	0.0336	0.0336	0.1184
3.6 Miscellaneous Equipment	-	-	-	-	0.1346	0.1346	-	-	-	0.0299	0.0299	0.1646
COST CENTRE 4: Tailings Disposal - included in Cost Centre 3												
4.1 Area Drainage	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
4.2 Clearing	- hectares	-	- $/ha	-	-	-	- hectares	- $/ha	-	-	-	-
4.3 Construction of Starter Dams & Overburden Dams	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
4.4 Piping of Tailings or Conveying of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.5 Tailings Sand Placement into Dyke	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.7 Recycling of Tailings Water	-	-	-	-	-	-	-	-	-	-	-	-
4.8 Rehandling of Tailings Sludge	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 2 (120,000 BPCD), B.W.E. SCHEME, ENHANCED LEVEL (Continued)

TABLE 7.4-6 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000m^3	-	- $/1,000m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	- km	- $/km	-	-	-	-
4.11 Oversize Reject Disposal	- 1,000 loose m^3	-	- $/1,000 loose m^3	-	-	-	- 1,000 loose m^3	- $/1,000 loose m^3	-	-	-	-
4.12 Oversize Reject Disposal Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
												0.1931
COST CENTRE 5: Establishment of Ultimate Land Use Resources												
5.1 Muskeg Rehandle Loading	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^{3}xkm	-	-	-	- 1,000 bank m^{3}xkm	- $/1,000 bank m^{3}xkm	-	-	-	-
5.3 Muskeg Rehandle Placement	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.4 Muskeg Rehandle Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.5 Overburden Rehandle Loading	3,044.5 1,000 bank m^3	-	377.35 $/1,000 bank m^3	0.0012	-	0.0012	3,044.5 1,000 bank m^3	42.35 $1,000 bank m^3	0.0001	-	0.0001	0.0013
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^{3}xkm	-	-	-	- 1,000 bank m^{3}xkm	- $/1,000 bank m^{3}xkm	-	-	-	-
5.7 Overburden Rehandle Placement	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $1,000 bank m^3	-	-	-	-
5.8 Overburden Rehandle Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.9 Muskeg Mining, Slurry Transport and Dewatering	11,036.7 1,000 bank m^3	-	2,628.92 $/1,000 bank m^3	0.0295	-	0.0295	- 1,000 bank m^3	- $/1,000 bank m^3	-	0.0105	0.0105	0.0400
5.10 Prepared Soil Manufacture	33,110.0 1,000 bank m^3	-	1,454.48 $/1,000 bank m^3	0.0490	-	0.0490	- 1,000 bank m^3	- $/1,000 bank m^3	-	0.0309	0.0309	0.0799
5.11 Prepared Soil Loading, F.E.L. & Trucks	33,110.0 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0145	-	0.0145	33,110.0 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0039	-	0.0039	0.0184
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	33,110.0 1,000 bank m^3	1.1 km	212.40 $/1,000 bank m^{3}xkm	0.0078	-	0.0078	36,307.1 1,000 bank m^{3}xkm	97.10 $/1,000 bank m^{3}xkm	0.0036	-	0.0036	0.0114
5.13 Prepared Soil Placement, Trucks	33,110.0 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0051	-	0.0051	33,110.0 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0008	-	0.0008	0.0059
5.14 Prepared Soil Road Construction	198.6 km	-	33,825.11 $/km	0.0068	-	0.0068	198.6 km	6,915.90 $/km	0.0014	-	0.0014	0.0082
5.15 Seed Bed Preparation, Maintenance	3,311.0 hectares	-	768.05 $/ha	0.0026	0.0254	0.0279	- hectares	- $/ha	-	-	-	0.0279
												0.1995
COST CENTRE 6: Supervision, Technical Services												
6.1 Equipment Maintenance (Staff only)	2,383.0 persons	-	29,970.00 $/person	0.0727	-	0.0727	-	-	-	-	-	0.0727
6.2 Planning (Staff only)	2,170.0 persons	-	29,605.00 $/person	0.0654	-	0.0654	-	-	-	-	-	0.0654
6.3 Mining (Staff only)	2,008.0 persons	-	30,065.00 $/person	0.0614	-	0.0614	-	-	-	-	-	0.0614
TOTAL COSTS						1.1764					0.4333	1.6097

NOTE: Refer to Chapter 6 for Cost Centre description.

8.0 CONCEPTS AND COSTS OF DEVELOPMENT AND RECLAMATION OF 60,000 BPCD OIL SANDS MINE

8.1 OVERVIEW OF DEVELOPMENT PLANS

Four mine plans have been developed for Ore Body No. 4. The dragline and bucket wheel mines at the Minimum and Improved Levels have been designed with similar slewing concepts. Consequently, the arrangements for out-of-pit tailings pond, outside dump, in-pit sludge pond, and in-pit backfill area are similar. No Enhanced Level plans have been developed for the 60,000 BPCD mine size. This subsection should be read in conjunction with the 60,000 BPCD mine plans supplied in Volume II.

Both the dragline and the bucket wheel plans utilize a BWE for overburden removal. The BWE in the dragline scheme is sized to remove only overburden and top reject. The draglines mining the second and third benches excavate only oil sands and centre reject. In this respect, the operating scheme is similar to that used for the 120,000 BPCD dragline schemes where the BWE operates independently of the draglines. In the three BWE mining schemes, three identical bucket wheel excavators are utilized on three benches of approximately equal height.

The dragline and bucket wheel mines both utilize a "Concept 5" tailings disposal scheme. The out-of-pit tailings pond initially stores both sand and sludge. Once the lowest oil sands mining bench is sufficiently advanced to allow the construction of an in-pit pond (about Year 13), the flow of tailings slurry is diverted to the in-pit sludge pond. A combination of tailings and overburden is used to construct the in-pit dyke, overburden being backfilled against the sand dyke. The length of the dyke is equal to the length of the mining bench, and in comparison to dykes in the other ore bodies studied, it is the largest dyke planned. The in-pit pond has sufficient capacity to store all the sludge generated during the life of the mine, as well as a few years of sand. Once the allowable sand has been placed into the sludge pond, the slurry is again routed to the out-of-pit tailings pond. As sludge is pumped from the out-of-pit tailings pond to the in-pit sludge pond, sand from the incoming tailings slurry fills the void created in the tailings pond.

The backfilling of the mine continues in a slewing fashion, from a position directly behind the in-pit dyke, until all the remaining overburden and reject is deposited. The dumping operation is performed in two lifts by a spreader operating from one bench. The reclaimed land surface elevations are close to the elevations prevailing prior to oil sands mining.

The plans for the Improved Level use the same mine layout and operational concepts as the Minimum Level plans. Since the sludge is dewatered at the Improved Level, considerably less sludge storage capacity is needed. As in the Minimum Level, sludge storage is in-pit, but the sludge pond dyke must be constructed of overburden rather than tailings sand. The dyke is progressively constructed from rehandled overburden, starting once the lowest oil sands bench has swung away from the western pit boundary. Construction of the dyke is completed by Year 15. The advance of the lowest bench is too slow to permit a creation of an in-pit tailings pond.

At the Improved Level, the volume of the outside dump is less because the in-pit sludge pond volume is smaller, and its dyke is constructed earlier in the life of the mine. Overburden which would otherwise have to be placed outside of the mine to create a sufficiently large void for the in-pit sludge pond can be backfilled into the pit much earlier. The requirement for a smaller in-pit dyke is intentional, and is the result of the sludge dewatering scheme used at the Improved Level.

The final elevation of the overburden backfill is about ten metres lower than at the Minimum Level, allowing more flexibility for surface contouring. Drainage of the backfill surface is achieved by creating preplanned local elevational differences during the dumping operation. Consequently, the final lake area is about the same at both the Minimum and Improved Levels of Reclamation.

Rather than forming a lake, the void might be utilized as the tailings or sludge pond of an adjoining operation. In this respect the void might be of considerable value to a neighbouring operation. If it is known that the end pit may be utilized for tailings disposal from the next mine, the backfill operation can be modified to maximize the end pit volume. The end pit lake might also function as a make-up water reservoir for a neighboring operation. However, the optimal use of such a "resource" can be achieved only by a regional oil sands development plan.

The out-of-pit tailings pond is located to the east of the mine against a relatively steep sloping hillside. Overall, the pond location is very attractive relative to the mine and the extraction plant. The out-of-pit pond is eventually sanded-in completely, thus minimizing its environmental impact. The final sanded-in pond surface area is larger than if the pond had been constructed on level ground. The pond conforms nicely with the surrounding landscape. The eastern pit boundary is located at an area with a relatively rapid drop in mining cut-off. The west-facing dyke is situated just east of the pit boundary and thus over marginal oil sands. This is an excellent example of maximizing bitumen yields while minimizing mining and tailings disposal costs.

The slewing layout for the application of draglines is rather attractive in Ore Body No. 4. Although some auxiliary mining might be necessary at the apex of the slewing system, this is not seen as a serious disadvantage. The primary reason for a successful combination of draglines with a slewing conveyor system is that Ore Body No. 4 is uniform and that only one dragline per bench is required. When two draglines are employed on a bench in a slewing system, the extra conveyors may create interference at the pivot point of the system.

The 60,000 BPCD mine plans are the only plans developed by the Consultants where an almost identical conveyor layout is possible with the application of either the dragline or the bucket wheel excavator as prime mover. In general, small pie-shaped or small rectangular-shaped mines have nearly identical conveyor layouts for the dragline and bucket wheel plans. In larger mines, other operational limitations result in considerable differences in conveyor layout.

The layout of the Ore Body No. 4 mine has two distinct drawbacks. The most serious is that currently submarginal ore situated at the north-eastern pit boundary may never be recoverable by surface mining. The complete or partial backfilling of this pit is likely to be a requirement even if in situ or underground mining methods are used at some later date to mine this low grade ore. Another disadvantage is that a currently-economical island of ore remains to the west of the in-pit sludge pond. However, this island could be developed in the future as a small pit (say a 20,000 BPCD) feeding a central extraction plant. This may prove to be a practical solution, in that there are numerous small but potentially mineable ore bodies situated between Ore Body No. 4 and the more southerly-located Ore Bodies No. 1 and No. 2.

The major operational differences between the four mine plans can be seen in the four sets of drawings provided in Volume II. Mass balance schedules are provided for overall mining, tailings disposal, and reclamation in subsection 8.2 and 8.3. Schedules for various other items are provided in the computer-printed cost summaries. Cost estimates for selected operating activities for a period of 35 years are provided, and include five years of preproduction and five years for deactivation of the mine. Summary cost comparisons are made in subsection 8.4.

8.2 MINING PLANS EMPLOYING DRAGLINES

Two dragline mine plans have been developed for Ore Body No. 4. The layouts for the Minimum and Improved Level mines in Ore Body No. 4 are very similar. The only major difference occurs with respect to the time at which backfilling of the pit with overburden can be started. At the Minimum Level, backfilling starts in Year 13.

The conveyor distribution point is located immediately north of the pit and south of the outside dump. The distribution point remains fixed in this location for the life of the mine. Three short conveyors with shunting heads connect the distribution point with the slewing face conveyors in the pit: one overburden conveyor and two oil sands conveyors. The face conveyor systems consist of two conveyors, the lengths of which are occasionally adjusted to permit a smooth rotation of the mining faces in a counterclockwise direction around the distribution point.

The overburden BWE is scheduled to remove only overburden, and so operates independently of the dragline production schedule. In the initial years, the BWE leads the oil sands removal by a considerable distance. At the completion of the mine, the oil sands mining faces have caught up with the overburden face. This is due to shallow overburden at the west end of the ore body, and the rather thick overburden at the east end of the mine.

The waste conveyor has been sized assuming full production of overburden plus full production of centre reject from one oil sands mining bench. Whenever possible, areas with high percentages of centre reject are scheduled to coincide with the scheduled shutdown of one or more extraction plant circuits. Centre reject from the bottom bench is backcast directly onto the pit floor as was done in the 120,000 BPCD dragline mine.

The outside dump is formed in two passes of the spreader by slewing a single length of conveyor first in a counterclockwise direction, and then in a clockwise direction. At the Minimum Level the dump conveyor is longer than at the Improved Level, since the quantities to be stored are greater at the Minimum Level. As soon as possible, overburden is returned to the pit as backfill. Oversize reject is hauled by truck and buried with the overburden and in the in-pit tailings pond.

At the Minimum Level, the dump conveyor is moved in-pit as soon as the lowest mining bench has passed the centreline of the in-pit dyke. Initially the spreader sorts material suitable for starter dyke construction from the general overburden being returned to the pit. Stockpiles of construction materials are formed alongside the conveyor. As required, material is taken to the starter dyke construction area by trucks or scraper. Overburden not suitable or not needed for starter dyke construction is placed in the backfill dump being started to the east of the dyke centreline. As the construction of the in-pit dyke west of the dyke centreline progresses, the overburden backfill is simultaneously raised to form the back side of the in-pit dyke. Gradually the backfill conveyor swings away from the dyke and the spreader begins a conventional backfilling operation. The bulk of the in-pit dyke is constructed from tailings sand.

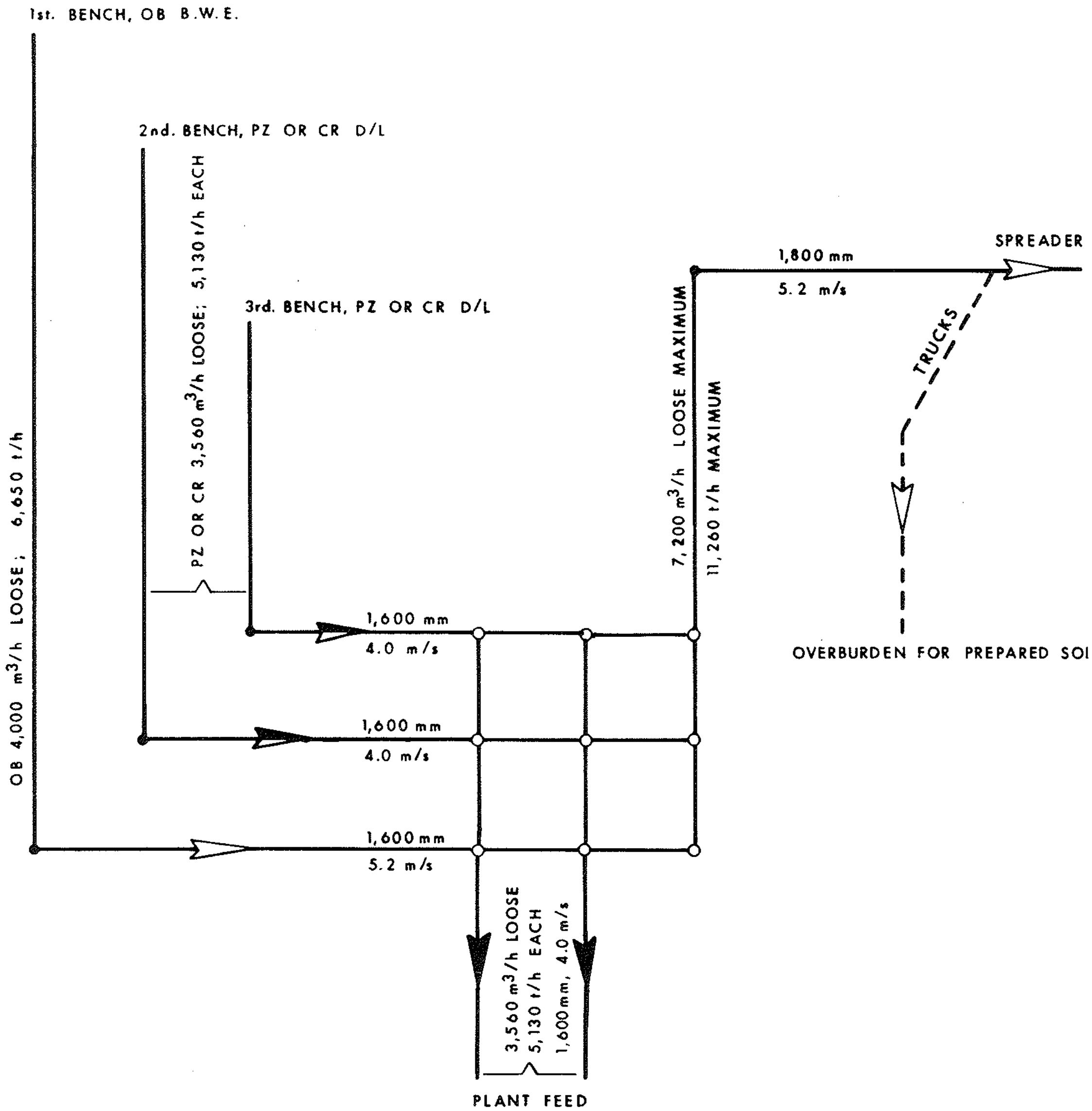
At the Improved Level, the backfill conveyor is laid into the pit as at the Minimum Level, but the conveyor does not extend completely to the southern pit wall. Instead, it is terminated just short of the in-pit dyke. The in-pit dyke is constructed entirely from overburden dumped by the spreader in the vicinity of the dyke. Trucks and scrapers transport the material from the dumping site to the construction site, and the dyke is progressively built with selected overburden. By the time the dyke is completed, the overburden conveyor is extended across the entire length of the pit. The backfill dumping procedure is as in the Minimum Level except that the dump height is lower. Backfilling at the Improved Level begins in Year 5.

The production schedules for both styles of mines are identical, and are detailed in Table 8.2-1. The materials handling system is described in Figure 8.2-1. Details for tailings disposal and reclamation appear in the following subsections.

TABLE 8.2-1

Ore Body No. 4, 60,000 B.P.C.D., - Production Schedule, 1 Bucket Wheel Excavator and 2 Draglines

Year	Top Bench			Middle Bench			Bottom Bench			Mine				
	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Backcast Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Bitumen tonnes $\times 10^6$	Crude barrels $\times 10^6$
-2	6.450	-	6.450	-	-	-	-	-	-	6.450	-	6.450	-	-
-1	8.900	-	8.900	-	-	-	-	-	-	8.900	-	8.900	-	-
1	8.900	-	8.900	1.327	7.953	9.280	0.560	2.960	3.520	10.787	10.913	21.700	2.584	12.715
2	8.900	-	8.900	2.268	9.622	11.890	2.075	9.782	11.857	13.243	19.404	32.647	4.390	21.273
3	8.900	-	8.900	2.268	9.622	11.890	2.613	10.129	12.742	13.781	19.751	33.532	4.360	20.950
4	8.900	-	8.900	2.145	10.209	12.354	2.568	10.164	12.732	13.613	20.373	33.986	4.492	21.584
5	8.900	-	8.900	2.116	10.348	12.464	2.326	10.350	12.676	13.342	20.698	34.040	4.558	21.900
6	8.900	-	8.900	2.116	10.348	12.464	2.326	10.350	12.676	13.342	20.698	34.040	4.558	21.900
7	8.900	-	8.900	1.692	9.311	11.003	2.181	10.006	12.187	12.773	19.317	32.090	4.495	21.900
8	8.900	-	8.900	1.681	9.282	10.963	1.875	9.282	11.157	12.456	18.564	31.020	4.460	21.900
9	8.900	-	8.900	1.490	9.248	10.738	1.875	9.282	11.157	12.265	18.530	30.795	4.459	21.900
10	8.900	-	8.900	1.240	9.203	10.442	1.513	9.242	10.755	11.653	18.445	30.098	4.456	21.900
11	8.900	-	8.900	1.240	9.203	10.442	1.411	9.231	10.642	11.551	18.434	29.985	4.456	21.900
12	8.900	-	8.900	0.961	9.476	10.437	1.411	9.231	10.642	11.272	18.707	29.979	4.469	21.900
13	8.900	-	8.900	0.661	9.770	10.431	0.876	9.779	10.655	10.437	19.549	29.986	4.505	21.900
14	8.900	-	8.900	0.632	9.758	10.390	0.760	9.898	10.658	10.292	19.656	29.948	4.509	21.900
15	8.900	-	8.900	0.371	9.657	10.028	0.629	9.876	10.505	9.900	19.533	29.433	4.499	21.900
16	8.900	-	8.900	0.371	9.657	10.028	0.438	9.845	10.283	9.709	19.502	29.211	4.497	21.900
17	8.900	-	8.900	0.500	9.464	9.965	0.438	9.845	10.283	9.838	19.309	29.147	4.486	21.900
18	8.900	-	8.900	0.561	9.373	9.934	0.676	9.487	10.163	10.137	18.860	28.997	4.460	21.900
19	8.900	-	8.900	0.561	9.373	9.904	0.690	9.466	10.156	10.151	18.839	28.990	4.459	21.900
20	8.900	-	8.900	0.595	9.180	9.776	0.699	9.426	10.125	10.194	18.606	28.800	4.451	21.900
21	8.900	-	8.900	0.598	9.165	9.763	0.742	9.223	9.965	10.240	18.388	28.628	4.444	21.900
22	8.900	-	8.900	0.697	9.163	9.860	0.742	9.223	9.965	10.339	18.386	28.725	4.444	21.900
23	8.900	-	8.900	0.992	9.158	10.150	0.943	9.202	10.145	10.835	18.360	29.195	4.444	21.900
24	8.900	-	8.900	0.992	9.158	10.150	1.173	9.177	10.350	11.065	18.335	29.400	4.444	21.900
25	0.965	-	0.965	0.655	6.045	6.700	1.173	9.177	10.350	2.793	15.222	18.015	3.689	18.178
Total	229.915	-	229.915	28.730	232.746	261.476	32.713	233.633	266.346	291.358	466.379	757.737	109.068	532.700



MATERIALS HANDLING SYSTEM - 60,000 BPCD
1 B.W.E. & 2 DRAGLINES - MINIMUM AND IMPROVED LEVEL

FIGURE 8.2-1

8.2.1 MINIMUM (WET) LEVEL OF RECLAMATION

The major aspects of the development of the 60,000 BPCD mine at the Minimum Level are depicted by five drawings accompanied by tailings disposal and reclamation schedules (See Tables 8.2.1-1 and 8.2.1-2). The mining schedule, which is common to the plans for both levels of reclamation, has been illustrated previously in Table 8.2-1. A drawing-by-drawing discussion follows;

General Mine Layout
(Techman Drawing No. D22910-50-00)

The location and ultimate size of the pit, out-of-pit tailings pond, outside dump, and plant site are shown. The Muskeg River touches the southwest corner of the pit and must be diverted before mining is begun (the cost of such a diversion has not been determined).

Mining and Tailings Disposal - Year 4
(Techman Drawing No. D22918-51-00)

By Year 4, all the working faces of the mine are fully developed. The overburden bench has advanced beyond the first oil sands bench to a maximum distance of 600 m. The two oil sands mining benches will remain almost parallel throughout the life of the mine. The outside waste dump covers approximately half the area designated for this purpose. The tailings pond dyke has reached a crest elevation of 373 m, still 25 m below its ultimate crest height. Surface reclamation is scheduled to begin in Year 9 and consequently only very minimal reclamation (for the purposes of pollution control) has occurred. Muskeg for prepared soil manufacture is being placed in muskeg dumps. Overburden is being placed into predetermined locations in the outside waste dumps.

Mining and Tailings Disposal - Year 13
(Techman Drawing No. D22918-52-00)

The mine has developed sufficiently to allow the construction of the in-pit tailings dyke starter dam. The northern portion has been constructed from selected overburden obtained from the outside waste dump. The overburden for the southern half of this starter dyke has been provided by utilizing the selective dumping capability of the overburden spreader. The out-of-pit tailings pond dyke has reached its ultimate crest height. In the spring of Year 14, the tailings slurry lines are to be directed into the pits, and the sand portion of the tailings dyke is to be constructed. Sludge removal from the out-of-pit tailings pond is also to be started. By Year 13, four years of surface reclamation activities have occurred. Prepared soil has been placed on part of the outside waste dump and dyke slopes.

Mining and Tailings Disposal - Year 22
(Techman Drawing No. D22918-53-00)

The in-pit tailings dyke has been constructed from compacted tailings sand to a crest elevation of 323 m by Year 16. Overburden backfill continues from the downstream side of the in-pit tailings dyke to within about 100 m of the toe of the lowest oil sands mining bench. The out-of-pit tailings pond has been substantially sanded-in, and the sludge pond is approaching its ultimate elevation. The tailings stream has been diverted to the in-pit sludge pond for 5 1/2 years, after which time it was switched back to the out-of-pit pond, beginning in Year 19. Prepared soil has been placed on a portion of the sanded-in beach starting in Year 16, and on the in-pit dyke and backfilled overburden starting in Year 17. Reclamation of the outside waste dump was completed in Year 15.

Material Distribution Plan
(Techman Drawing No. D22916-54-00)

This plan shows the types of materials that must be surfaced with prepared soil. The sludge pond surface is currently considered unreclaimable. Waste dumps and backfill areas are surfaced with acceptable overburden materials by the spreader during the construction of the dumps and therefore require only the application of muskeg, whereas tailings sand surfaces require the application of both overburden and muskeg to

Ore Body No. 4, 60,000 B.P.C.D.-1 B.W.E. & 2 Draglines
TAILINGS SCHEDULE
FOR MINIMUM LEVEL OF RECLAMATION
TABLE 8.2.1-1

Outline of Tailings Disposal Scheme:

- Years 1 - 14 Tailings are Pumped to out-of-Pit Tailings Pond.
- Years 14 - 19 Tailings Diverted to in-Pit Pond to Build Sludge Pond Dyke and Accommodate Some Extra Beach Sand in Order to Reduce Size of out-of-Pit Tailings Pond.
- Years 14-25 Sludge Removal from out-of-Pit Tailings Pond to in-Pit Sludge Pond.
- Years 19-25 Tailings to out-of-Pit Pond to Fill it with Sand.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge Rehandle Volume [$m^3 \times 10^6$]
1	24.766	9.636	3.889	11.240	5.600	5.640	0
2	44.035	17.134	6.916	19.986	3.600	16.386	0
3	44.823	17.440	7.039	20.344	3.400	16.944	0
4	46.234	17.989	7.261	20.984	3.600	17.384	0
5	46.972	18.276	7.377	21.319	3.300	18.019	0
6	46.972	18.276	7.377	21.319	3.100	18.219	0
7	43.838	17.057	6.885	19.897	2.300	17.597	0
8	42.129	16.392	6.616	19.121	1.900	17.221	0
9	42.052	16.362	6.604	19.086	1.500	17.586	0
10	41.859	16.287	6.574	18.998	1.300	17.698	0
11	41.834	16.277	6.570	18.987	1.200	17.787	0
12	42.454	16.518	6.667	19.268	1.100	18.168	0
13	44.365	17.262	6.967	20.135	0.500	19.635	0
14	44.607	17.356	7.005	20.246	10.129	10.117	8.239
15	44.328	17.248	6.962	20.119	6.814	13.305	10.985
16	44.258	17.220	6.951	20.087	5.967	14.120	10.985
17	43.820	17.050	6.882	19.888	0	19.888	10.985
18	42.801	16.653	6.722	19.426	0	19.426	10.985
19	42.753	16.635	6.714	19.404	0	19.404	10.985
20	42.224	16.429	6.631	19.164	0	19.164	10.985
21	41.730	16.237	6.553	18.940	0	18.940	10.985
22	41.725	16.235	6.553	18.938	0	18.938	10.985
23	41.666	16.212	6.544	18.911	0	18.911	10.985
24	41.609	16.190	6.535	18.885	0	18.885	10.985
25	34.545	13.441	5.425	15.679	0	15.679	9.887
	1,058.399	411.812	166.219	480.371	55.310	425.061	127.976

Ore Body No. 4, 60,000 B.P.C.D. - 1 B.W.E. & 2 Draglines
SCHEDULE FOR MINIMUM LEVEL OF RECLAMATION
TABLE No. 8.2.1-2

Soil Composition:
0.20m Muskeg
0.20m Overburden
0.20m Sand (where applicable)

Soil Manufacture:
Layer of muskeg and overburden (where required) are spread onto area to be reclaimed and plowed 0.6 m deep.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1	127	76	25	2.6	51	0
2	127	76	25	2.6	51	0
3	127	76	25	2.6	51	0
4	127	76	25	2.6	51	0
5	127	76	25	2.6	51	0
6	127	76	25	2.6	51	0
7	127	76	25	2.6	51	0
8	127	76	25	2.6	51	0
9	374	224	75	3.8	100	5.0
10	374	224	75	3.7	100	5.0
11	374	224	75	3.6	100	5.0
12	670	402	134	3.1	219	4.9
13	670	402	134	3.0	219	4.8
14	543	326	109	3.2	168	4.7
15	543	326	109	3.2	168	4.6
16	902	541	180	4.2	230	4.5
17	958	575	192	4.0	242	4.1
18	958	575	192	4.0	242	4.1
19	898	539	180	3.7	229	4.5
20	898	539	180	3.3	229	4.5
21	898	539	180	3.3	229	4.5
22	898	539	180	3.3	229	4.5
23	1,049	629	210	2.7	259	4.5
24	1,049	629	210	2.7	259	4.5
25	1,049	629	210	2.7	259	4.5
26	1,049	629	210	2.7	259	4.5
27	1,049	629	210	1.9	259	4.5
28	733	440	147	1.5	293	2.4
29	733	440	147	1.5	293	2.4
30	733	440	147	1.5	293	2.4
	18,418	11,048	3,686		5,286	

form a prepared soil surface. Muskeg dumps require no surfacing. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed. The end-pit is filled with water to form a fresh water lake. All exposed pit walls require the application of prepared soil.

Reclamation Plan
(Techman Drawing No. D22980-55-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurred are shown. Only the sludge pond remains wet and unreclaimable. Plant species are selected according to the reclamation objectives for the Minimum Level as described in Chapter 4.0. Table 8.2.1-2 is a year-by-year summary of prepared soil manufacture and placement.

8.2.2 IMPROVED (DEWATERED) LEVEL OF RECLAMATION

The major aspects of the development of the mines at the Improved Level are depicted by six drawings, accompanied by tailings disposal and reclamation schedules (See Table 8.2.2-1 and 8.2.2-2). The mining schedule, which is common to both the plans at the Minimum and Improved Levels of Reclamation, has been illustrated previously in Table 8.2-1. A drawing-by- drawing discussion follows:

General Mine Layout
(Techman Drawing No. 22910-56-00)

The mine boundaries and advance of the mining faces are identical to those for the plans at the Minimum Level. The major difference exists with respect to the size of tailings pond and outside waste dump, the former being considerably larger and the latter significantly smaller than in the Minimum Level.

Mining and Tailings Disposal - Year 4
(Techman Drawing No. D22918-57-00)

All faces of the mine are fully developed by Year 4. The outside waste dump is nearing completion. Overburden materials suitable for prepared soil manufacture have been selectively separated by the single spreader and trucked into the prepared soil blending pile located between the outside waste dump and the mines. The entire quantity of sludge and soil produced to date is stored in the out-of-pit tailings pond.

Mining and Tailings Disposal - Year 13
(Techman Drawing No. D22918-58-00)

By the end of Year 13, full-face-length backfilling of the mined-out portion of the pit is under way. The construction of the overburden dyke required to form the in-pit sludge pond was started in Year 5 after the installation of a less-than-full-face-length overburden conveyor in the pit. Overburden suitable for the construction of the dyke has been made available by selective dumping by the spreader over a period of 5 years. Beginning in Year 9, sludge has been pumped from the out-of-pit

Ore Body No. 4, 60,000 B.P.C.D.- 1 B.W.E. & 2 Draglines
TAILINGS SCHEDULE
FOR IMPROVED LEVEL OF RECLAMATION
TABLE 8.2.2-1

Outline of Tailings Disposal Scheme:

- Tailings Slurry is Pumped to out-of-Pit Tailings Pond. Commencing With Year 9, Sludge is Pumped from out-of-Pit Tailings Pond to the Treatment Plant where about 50% of Water and Most of Remaining Bitumen are Removed. Treated Sludge is Disposed of to in-Pit Pond. Out-of-Pit Pond Completly Filled With Sand.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge into Treatment [$m^3 \times 10^6$]
1	24.766	9.636	3.889	11.240	6.100	5.140	0
2	44.035	17.134	6.916	19.986	4.400	15.586	0
3	44.823	17.440	7.039	20.344	3.300	17.044	0
4	46.234	17.989	7.261	20.984	3.100	17.884	0
5	46.972	18.276	7.377	21.319	3.400	17.919	0
6	46.972	18.276	7.377	21.319	3.000	18.319	0
7	43.838	17.057	6.885	19.897	3.200	16.697	0
8	42.129	16.392	6.616	19.121	2.700	16.421	0
9	42.052	16.362	6.604	19.086	2.100	16.986	9.835
10	41.859	16.287	6.574	18.998	1.800	17.198	9.835
11	41.834	16.277	6.570	18.987	1.600	17.387	9.835
12	42.454	16.518	6.667	19.268	1.500	17.768	9.835
13	44.365	17.262	6.967	20.135	1.300	18.835	9.835
14	44.607	17.356	7.005	20.246	1.100	19.146	9.835
15	44.328	17.248	6.962	20.119	1.100	19.019	9.835
16	44.258	17.220	6.951	20.087	0.900	19.187	9.835
17	43.820	17.050	6.882	19.888	0.300	19.588	9.835
18	42.801	16.653	6.722	19.426	0	19.426	9.835
19	42.753	16.635	6.714	19.404	0	19.404	9.835
20	42.224	16.429	6.631	19.164	0	19.164	9.835
21	41.730	16.237	6.553	18.940	0	18.940	9.835
22	41.725	16.235	6.553	18.938	0	18.938	9.835
23	41.666	16.212	6.544	18.911	0	18.911	9.835
24	41.609	16.190	6.535	18.885	0	18.885	9.835
25	34.545	13.441	5.425	15.679	0	15.679	8.859
	1,058.399	411.812	166.219	480.371	40.900	439.471	166.219

Ore Body No. 4, 60,000 B.P.C.D. - 1 B.W.E. & 2 Draglines
SCHEDULE FOR IMPROVED LEVEL OF RECLAMATION

TABLE No. 8.2.2-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Alternating layers of muskeg and overburden are scraped at a sloping face of pile by dozers.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1	229	229	1.7	76	0.7	153	0.5
2	229	229	1.7	76	0.7	153	0.5
3	229	229	1.7	76	0.7	153	0.5
4	229	229	1.7	76	0.7	153	0.5
5	258	258	1.7	86	0.7	172	0.5
6	258	258	1.7	86	0.7	172	0.5
7	0	0	0	0	0	0	0
8	0	0	0	0	0	0	0
9	0	0	0	0	0	0	0
10	0	0	0	0	0	0	0
11	561	561	3.5	187	0.6	374	1.6
12	561	561	3.5	187	0.6	374	1.6
13	561	561	3.5	187	0.6	374	1.6
14	660	660	3.7	220	0.5	440	1.3
15	660	660	3.7	220	0.5	440	1.3
16	660	660	3.7	220	0.5	440	1.3
17	660	660	3.7	220	0.5	440	1.3
18	1,471	1,471	5.0	490	0.5	981	0.9
19	1,471	1,471	5.0	490	0.5	981	0.9
20	1,471	1,471	5.0	490	0.5	981	0.9
21	1,471	1,471	3.4	490	0.5	981	2.9
22	1,471	1,471	3.4	490	0.5	981	2.9
23	1,630	1,630	2.7	543	0.5	1,087	3.0
24	1,630	1,630	2.7	543	0.5	1,087	3.0
25	1,630	1,630	2.7	543	0.5	1,087	3.0
26	1,630	1,630	2.7	543	0.5	1,087	3.0
27	1,630	1,630	2.7	543	0.5	1,087	3.0
28	733	733	1.5	244	1.0	489	0.7
29	733	733	1.5	244	1.0	489	0.7
30	733	733	1.5	244	2.5	489	0.7
	23,459	23.459		7,814		15,645	

tailings pond to a treatment plant, and finally disposed of in the sludge pond. Additional prepared soil blending piles have been formed and reclamation of backfill and dyke slopes is under way. The reclamation of the outside waste dump was started in Year 1, and completed in Year 6. Four years later, the reclamation of the out-of-pit dyke and backfill was started.

Mining and Tailings Disposal - Year 22
(Techman Drawing No. D22918-59-00)

The out-of-pit tailings pond has been gradually sanded-in so that by Year 22 the area of wet pond surface is considerably reduced. The overburden backfill has been placed to a mean elevation of 308 m by a single spreader operating in both the high and low dumping modes. Reclamation is current, with the placement of prepared soil progressing with the advances of the backfill and the tailings sand beach.

Material Distribution Plan
(Techman Drawing No. D22916-60-00)

This drawing shows that the main materials to be surfaced with prepared soil are dry tailings sand and overburden. During the course of mining, toxic materials are buried. In this plan most of the muskeg stored in the muskeg dumps is consumed in the manufacture of prepared soil. The blended materials from the prepared soil stockpiles are distributed by trucks. After completion of overburden removal, loading of prepared soil may be done with a front-end loader. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-61-00)

The surface area to be reclaimed as well as the time period during which reclamation occurred are shown. A pie-shaped end-pit lake is formed. Run-off from the backfill and the tailings pond is channeled into this lake. Table 8.2.2-2 is a detailed schedule for the reclamation activities on a year-by-year basis. Plant species are selected according to the reclamation objectives for the Improved Level of Reclamation described in Chapter 4.0.

8.3 MINING PLANS EMPLOYING BUCKET WHEEL EXCAVATORS

Two bucket wheel mine plans have been developed for Ore Body No. 4. The basic layout for these mines is similar to that used in the dragline mine plans (see subsection 8.2, Mining Plans Employing Draglines).

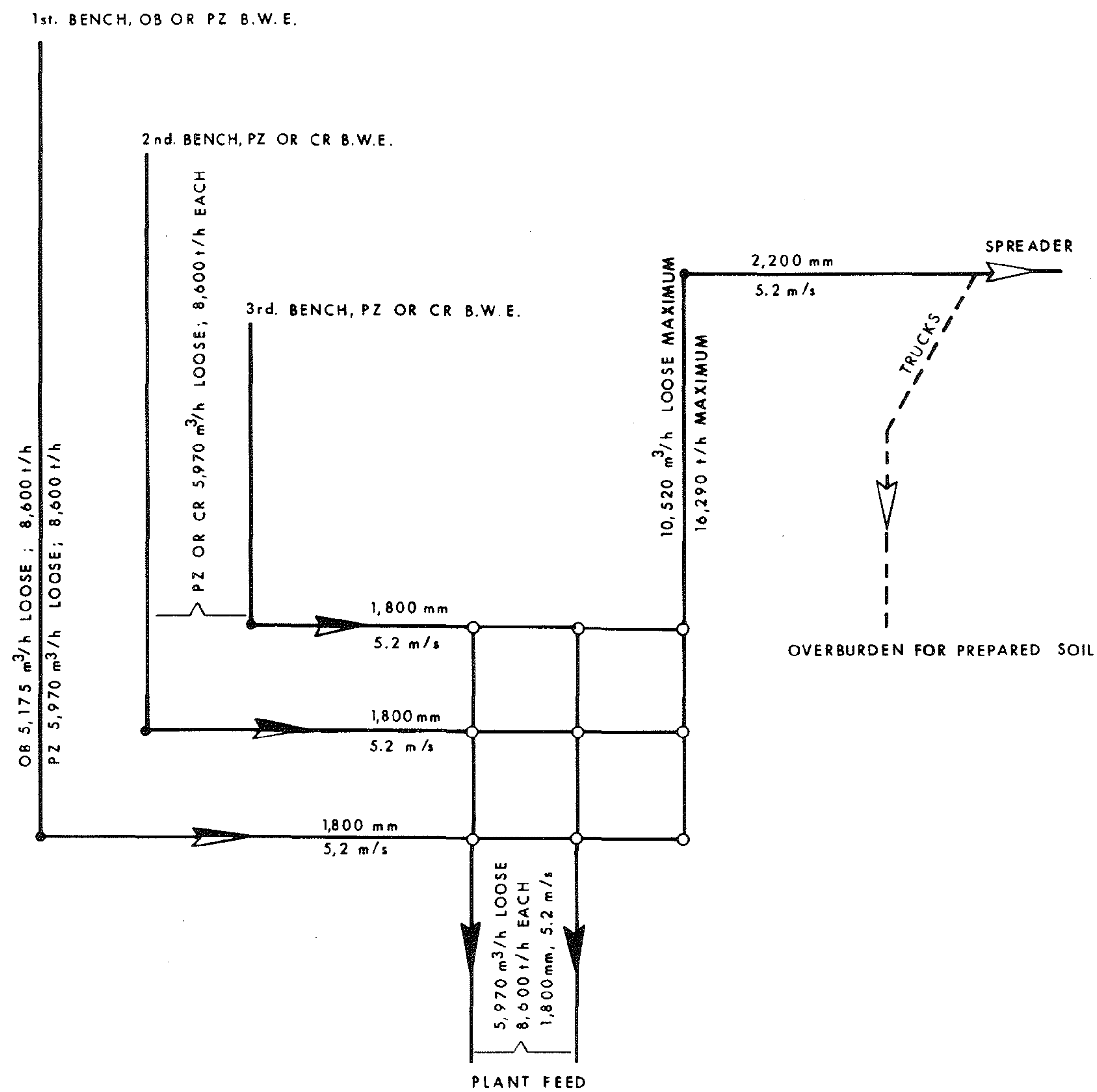
The amounts of materials handled in the bucket wheel mine differ from the amounts handled in the dragline mines. The volume of overburden to be moved is identical, 8.9 million bank cubic meters per year; however, the quantity of plant feed is less because dilution has been reduced.

The production schedules for bucket wheel mines at the Minimum and Improved Levels are identical, and are detailed in Table 8.3-1. The materials handling system is illustrated in Figure 8.3-1. Details for tailings disposal and reclamation appear in the following subsections.

TABLE 8.3-1

Ore Body No. 4, 60,000 B.P.C.D., - Production Schedule, 3 Bucket Wheel Excavators -

Year	Top Bench Waste $bm^3 \times 10^6$	Top Bench Feed $bm^3 \times 10^6$	Top Bench Total $bm^3 \times 10^6$	Middle Bench Waste $bm^3 \times 10^6$	Middle Bench Feed $bm^3 \times 10^6$	Middle Bench Total $bm^3 \times 10^6$	Bottom Bench Waste $bm^3 \times 10^6$	Bottom Bench Feed $bm^3 \times 10^6$	Bottom Bench Total $bm^3 \times 10^6$	Mine Waste $bm^3 \times 10^6$	Mine Feed $bm^3 \times 10^6$	Mine Total $bm^3 \times 10^6$	Mine Bitumen tonnes $\times 10^6$	Mine Crude barrels $\times 10^6$
-2	6.450	-	6.450	-	-	-	-	-	-	6.450	-	6.450	-	-
-1	8.900	-	8.900	-	-	-	-	-	-	8.900	-	8.900	-	-
1	9.254	1.796	11.050	1.314	6.661	7.975	0.491	2.534	3.025	11.059	10.991	22.050	3.020	14.852
2	9.268	1.796	11.050	2.405	9.115	11.520	1.881	8.569	10.450	13.554	19.455	33.009	3.980	19.265
3	9.396	1.880	11.246	2.354	8.922	11.276	2.354	8.922	11.276	14.105	19.724	33.829	4.351	20.904
4	9.400	2.063	11.445	2.166	9.279	11.445	2.332	9.113	11.445	13.898	20.437	34.335	4.502	21.629
5	9.376	2.078	11.456	2.139	9.318	11.457	2.139	9.318	11.457	13.655	20.714	34.369	4.558	21.900
6	9.365	2.039	11.403	2.112	9.291	11.403	2.129	9.274	11.403	13.607	20.602	34.209	4.558	21.900
7	9.122	1.032	10.145	1.882	9.118	11.000	1.981	9.019	11.000	12.985	19.160	32.145	4.558	21.900
8	8.970	0.340	9.311	1.882	9.118	11.000	1.882	9.118	11.000	12.734	18.577	31.311	4.558	21.900
9	8.900	-	8.900	1.625	9.248	10.873	1.917	9.288	11.205	12.441	18.537	30.978	4.558	21.900
10	8.900	-	8.900	1.444	9.223	10.667	1.459	9.226	10.685	11.803	18.449	30.252	4.558	21.900
11	8.900	-	8.900	1.444	9.223	10.667	1.444	9.223	10.667	11.788	18.447	30.235	4.456	21.900
12	8.900	-	8.900	0.960	9.666	10.626	1.393	9.270	10.663	11.254	18.936	30.190	4.478	21.900
13	8.900	-	8.900	0.777	9.834	10.611	0.777	9.834	10.611	10.454	19.668	30.123	4.510	21.900
14	8.900	-	8.900	0.657	9.804	10.461	0.777	9.834	10.611	10.334	19.638	29.972	4.508	21.900
15	8.900	-	8.900	0.448	9.752	10.200	0.541	9.775	10.316	9.889	19.527	29.416	4.498	21.900
16	8.900	-	8.900	0.448	9.752	10.200	0.448	9.752	10.200	9.795	19.504	29.299	4.496	21.900
17	8.900	-	8.900	0.695	9.439	10.134	0.527	9.652	10.179	10.122	19.092	29.213	4.473	21.900
18	8.900	-	8.900	0.707	9.425	10.132	0.707	9.425	10.132	10.313	18.849	29.163	4.460	21.900
19	8.900	-	8.900	0.719	9.370	10.089	0.707	9.425	10.132	10.326	18.795	29.121	4.458	21.900
20	8.900	-	8.900	0.759	9.199	9.958	0.738	9.291	10.029	10.397	18.490	28.887	4.447	21.900
21	8.900	-	8.900	0.759	9.199	9.958	0.759	9.199	9.958	10.418	18.398	28.816	4.444	21.900
22	8.900	-	8.900	1.026	9.184	10.210	0.759	9.199	9.958	10.685	18.383	29.067	4.444	21.900
23	8.900	-	8.900	1.201	9.174	10.375	1.180	9.175	10.355	11.281	18.349	29.630	4.444	21.900
24	3.921	-	3.921	1.201	9.174	10.375	1.201	9.174	10.375	6.322	18.348	24.670	4.444	21.900
25				0.297	2.272	2.569	1.078	8.236	9.314	1.375	10.508	11.883	2.545	12.543
Total	226.922	13.024	239.907	31.421	223.760	255.181	31.601	224.845	256.446	289.944	461.577	751.521	108.306	527.193



MATERIALS HANDLING SYSTEM- 60,000 BPCD
3 B.W.E.- MINIMUM AND IMPROVED LEVEL

FIGURE 8.3-1

8.3.1 MINIMUM (WET) LEVEL OF RECLAMATION

The major aspects of the development of the 60,000 BPCD mine at the Minimum Level are depicted by six drawings, accompanied by tailings disposal and reclamation schedules (see Table 8.3.1-1 and 8.3.1-2). The mining schedule, which is common to the plans at both levels of reclamation described in Chapter 8.0, has been illustrated previously in Table 8.3-1. A drawing-by-drawing discussion follows:

General Mine Layout
(Techman Drawing No. D22910-62-00)

The location and ultimate size of the pit, out-of-pit tailings pond, outside dump, and plant site are shown. Except that outside dump is slightly larger, the layout is identical to the layout of the dragline mine plan at the Minimum Level of Reclamation. The Muskeg River touches the southwest corner of the pit and must be diverted before mining commences (the cost of such a diversion has not been determined).

Mining and Tailings Disposal - Year 4
(Techman Drawing No. D22918-63-00)

By Year 4, all the working faces of the mine are fully developed. The outside waste dump is larger than that in the dragline plan, since in the dragline plan, centre reject from the bottom bench is backcast onto the pit floor, while in the bucket wheel mine all reject for the first four years of operation is placed into the outside dump. Although muskeg is being stockpiled for future use in the manufacture of prepared soil, no separate overburden stockpiles for prepared soil are being formed. Instead, overburden suitable for the manufacture of prepared soil is being placed temporarily into pre-determined positions within the outside waste dump. Beginning in Year 9, this material will be loaded and trucked to the reclamation site.

Mining and Tailings Disposal - Year 13
(Techman Drawing No. D22918-64-00)

The mine has developed sufficiently to allow the construction of the in-pit tailings dyke starter dam. The northern portion has been constructed from selected overburden obtained from the outside waste dump. The southern portion has been supplied with overburden by the overburden conveyor used to backfill the pit. In Year 13, the out-of-pit tailings pond reaches its ultimate crest height. Beginning in Year 14, the tailings slurry lines will be redirected into the pit for 5 years. The upstream portion of the in-pit dyke is to be built from tailings sand while the downstream portion will be constructed with backfill. By Year 13 a large portion of the outside waste dump and out-of-pit tailings pond dyke have been reclaimed with prepared soil.

Mining and Tailings Disposal - Year 22
(Techman Drawing No. D22918-65-00)

The in-pit tailings dyke was completed to a crest elevation of 318 m in Year 16. Sludge transfer from the out-of-pit tailings pond began in Year 14 and continues into Year 25. In Year 19 the tailing slurry lines were redirected into the out-of-pit tailings pond to complete the sanding-in of the pond. Reclamation of the outside dump was completed in Year 15, with reclamation of the in-pit dyke in Year 17 and 18. The reclamation of the backfill was started in Year 19 and continues until Year 27. The out-of-pit tailings pond dyke was resurfaced with prepared soil between Years 9 and 15 and followed by progressive reclamation of the sanded-in beaches. Beach reclamation continues until Year 27.

Material Distribution Plan
(Techman Drawing No. D22916-66-00)

This plan shows the types of materials that must be surfaced with prepared soil. The sludge pond surface is currently considered unreclaimable. Waste dumps and backfill areas are surfaced with acceptable overburden materials by spreader during construction of the dumps, and therefore require only the application of muskeg, whereas tailings sand surfaces require the application of both overburden and muskeg to form a prepared soil surface. Muskeg dumps require no surfacing. All exposed pit walls require the application of prepared soil. The end-pit is filled with water to form a fresh water lake. Refer to Section 10.6, Table 10.6-1, for a mine-by-mine comparison of surfaces to be reclaimed.

Ore Body No. 4, 60,000 B.P.C.D.- 3 B.W.E.
TAILINGS SCHEDULE
FOR MINIMUM LEVEL OF RECLAMATION
TABLE 8.3.1-1

Outline of Tailings Disposal Scheme:

- Years 1 - 14 Tailings are Pumped to out-of-Pit Tailings Pond.
- Years 14 - 19 Tailings Diverted to in-Pit Pond to Build Sludge Pond Dyke and Accommodate Some Extra Beach Sand in Order to Reduce Size of out-of-Pit Tailings Pond.
- Years 14 - 25 Sludge Removal from out-of-Pit Tailings Pond to in-Pit Sludge Pond.
- Years 19 - 25 Tailings to out-of-Pit Pond to Fill it with Sand.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge Rehandle Volume [$m^3 \times 10^6$]
1	24.943	9.705	3.917	11.321	5.600	5.721	0
2	44.151	17.179	6.934	20.039	3.600	16.439	0
3	44.761	17.416	7.030	20.316	3.400	16.916	0
4	46.379	18.046	7.284	21.050	3.600	17.450	0
5	47.009	18.290	7.383	21.336	3.300	18.036	0
6	46.755	18.192	7.343	21.220	3.100	18.120	0
7	43.481	16.918	6.828	19.734	2.300	17.434	0
8	42.158	16.403	6.621	19.134	1.900	17.234	0
9	42.067	16.368	6.606	19.093	1.500	17.593	0
10	41.868	16.290	6.575	19.003	1.300	17.703	0
11	41.864	16.289	6.574	19.000	1.200	17.800	0
12	42.973	16.720	6.749	19.504	1.100	18.404	0
13	44.635	17.367	7.010	20.258	0.500	19.758	0
14	44.567	17.340	6.999	20.227	10.129	10.098	8.135
15	44.316	17.242	6.960	20.113	6.814	13.299	10.847
16	44.263	17.222	6.951	20.089	5.967	14.122	10.847
17	43.326	16.858	6.804	19.664	0	19.664	10.847
18	42.776	16.644	6.718	19.415	0	19.415	10.847
19	42.653	16.596	6.699	19.359	0	19.359	10.847
20	41.962	16.327	6.590	19.045	0	19.045	10.847
21	41.752	16.245	6.557	18.950	0	18.950	10.847
22	41.717	16.232	6.552	18.934	0	18.934	10.847
23	41.641	16.202	6.539	18.899	0	18.899	10.847
24	41.640	16.201	6.539	18.898	0	18.898	10.847
25	23.847	9.279	3.745	10.823	0	10.823	9.759
	1,047.502	407.572	164.506	475.424	55.310	420.114	126.364

Ore Body No. 4, 60,000 B.P.C.D. – 3 Bucket Wheel Excavators
SCHEDULE FOR MINIMUM LEVEL OF RECLAMATION

TABLE No. 8.3.1-2

Soil Composition:
0.20 m Muskeg
0.20 m Overburden
0.20 m Sand (where applicable)

Soil Manufacture:
Layer of muskeg and overburden (where required) are spread onto area to be reclaimed and plowed 0.6 m deep.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1	155	93	31	2.6	62	0
2	155	93	31	2.6	62	0
3	155	93	31	2.6	62	0
4	155	93	31	2.6	62	0
5	155	93	31	2.6	62	0
6	155	93	31	2.6	62	0
7	155	93	31	2.6	62	0
8	155	93	31	2.6	62	0
9	402	241	80	3.8	111	5.0
10	402	241	80	3.7	111	5.0
11	402	241	80	3.6	111	5.0
12	764	458	153	3.1	256	4.9
13	764	458	153	3.0	256	4.8
14	609	365	122	3.2	194	4.7
15	609	365	122	3.2	194	4.6
16	767	460	153	4.2	176	4.5
17	823	494	165	4.0	187	4.1
18	823	494	165	4.0	187	4.1
19	917	550	183	3.7	236	4.5
20	917	550	183	3.3	236	4.5
21	917	550	183	3.3	236	4.5
22	917	550	183	3.3	236	4.5
23	1,068	641	214	2.7	266	4.5
24	1,068	641	214	2.7	266	4.5
25	1,068	641	214	2.7	266	4.5
26	1,087	652	217	2.7	266	4.5
27	1,049	629	210	1.9	266	4.5
28	733	440	147	1.5	293	2.4
29	733	440	147	1.5	293	2.4
30	733	440	147	1.5	293	2.4
	18,812	11,285	3,763		5,432	

Reclamation Plan
(Techman Drawing No. D22980-67-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurred are shown. Only the sludge pond remains wet and unreclaimable. Plant species are selected according to the reclamation objectives for the Minimum Level of Reclamation as described in Chapter 4.0. Table 8.3-2 is a year-by-year summary of prepared soil manufacture and placement.

8.3.2 IMPROVED (DEWATERED) LEVEL OF RECLAMATION

The major aspects of the development of the 60,000 BPCD mine at the Improved Level are depicted by six drawings, accompanied by tailings disposal and reclamation schedules (See Table 8.3.2-1 and 8.3.2-2). The mining schedule, which is common to the plans at both levels of reclamation described in Chapter 8.0, has been illustrated previously in Table 8.3-2. A drawing-by-drawing discussion follows:

General Mine Layout
(Techman Drawing No. D22910-68-00)

The location and ultimate size of the pit, out-of-pit tailings pond, outside dump, and plant site are shown. The out-of-pit tailings pond is larger but the outside waste dump smaller, relative to the BWE plan at the Minimum Level. The in-pit sludge pond size and position allows the backfilling of the mine to begin earlier; therefore a reduced volume of material is stored in the outside dump. The sludge pond dyke is constructed of overburden since construction of the dyke from tailings sand would result in the dilution of the sludge with tailings water.

Mining and Tailings Disposal - Year 4
(Techman Drawing No. D22918-69-00)

By Year 4 all the working faces of the mine are fully developed. Overburden and reject are being conveyed to the outside dump. Overburden suitable for reclamation is being removed from the outside dump and trucked to a prepared soil blending pile for placement into a layered stockpile. The out-of-pit dyke crest height is at 369 metres, still more than 20 metres below its ultimate crest elevation. All tailings sand, sludge, and water is stored in the pond. Sludge treatment will not begin until Year 9, because the mining faces have not advanced far enough to allow the construction of the in-pit sludge pond dyke. Some reclamation of the outside waste dump has occurred and continues until Year 6.

Ore Body No. 4, 60,000 B.P.C.D. 3 B.W.E.
TAILINGS SCHEDULE FOR IMPROVED LEVEL OF RECLAMATION
TABLE 8.3.2-1

Outline of Tailings Disposal Scheme:

- Tailings Slurry is Pumped to out-of-Pit Tailings Pond, Commencing With Year 9 Sludge is Pumped from out-of-Pit Tailings Pond to the Treatment Plant where about 50% of Water and Most of Remaining Bitumen are Removed. Treated Sludge is Disposed of to in-Pit Pond. Out-of-Pit Pond Completly Filled With Sand.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge into Treatment [$m^3 \times 10^6$]
1	24.943	9.705	3.917	11.321	6.100	5.221	0
2	44.151	17.179	6.934	20.039	4.400	15.639	0
3	44.761	17.416	7.030	20.316	3.300	17.016	0
4	46.379	18.046	7.284	21.050	3.100	17.950	0
5	47.009	18.290	7.383	21.336	3.400	17.936	0
6	46.755	18.192	7.343	21.220	3.000	18.220	0
7	43.481	16.918	6.828	19.734	3.200	16.534	0
8	42.158	16.403	6.621	19.134	2.700	16.434	0
9	42.067	16.368	6.606	19.093	2.100	16.993	9.734
10	41.868	16.290	6.575	19.003	1.800	17.203	9.734
11	41.864	16.289	6.754	19.000	1.600	17.400	9.734
12	42.973	16.720	6.749	19.504	1.500	18.004	9.734
13	44.635	17.367	7.010	20.258	1.300	18.958	9.734
14	44.567	17.340	6.999	20.227	1.100	19.127	9.734
15	44.316	17.242	6.960	20.113	1.100	19.013	9.734
16	44.263	17.222	6.951	20.089	0.900	19.189	9.734
17	43.326	16.858	6.804	19.664	0.300	19.364	9.734
18	42.776	16.644	6.718	19.415	0	19.415	9.734
19	42.653	16.596	6.699	19.359	0	19.359	9.734
20	41.962	16.327	6.590	19.045	0	19.045	9.734
21	41.752	16.245	6.557	18.950	0	18.950	9.734
22	41.717	16.232	6.552	18.934	0	18.934	9.734
23	41.641	16.202	6.539	18.899	0	18.899	9.734
24	41.640	16.201	6.539	18.898	0	18.898	9.734
25	23.847	9.279	3.745	10.823	0	10.823	8.762
	1,047.502	407.572	164.506	475.424	40.900	434.524	164.506

Ore Body No. 4, 60,000 B.P.C.D. - 3 Bucket Wheel Excavators
SCHEDULE FOR IMPROVED LEVEL OF RECLAMATION
TABLE No. 8.3.2-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Alternating layers of muskeg and overburden are scraped at a sloping face of pile by dozers.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1	283	283	1.7	94	0.7	189	0.5
2	283	283	1.7	94	0.7	189	0.5
3	283	283	1.7	94	0.7	189	0.5
4	283	283	1.7	94	0.7	189	0.5
5	318	318	1.7	106	0.7	212	0.5
6	318	318	1.7	106	0.7	212	0.5
7	0	0	0	0	0	0	0
8	0	0	0	0	0	0	0
9	0	0	0	0	0	0	0
10	0	0	0	0	0	0	0
11	561	561	3.5	187	0.6	374	1.6
12	561	561	3.5	187	0.6	374	1.6
13	561	561	3.5	187	0.6	374	1.6
14	660	660	3.7	220	0.5	440	1.3
15	660	660	3.7	220	0.5	440	1.3
16	660	660	3.7	220	0.5	440	1.3
17	660	660	3.7	220	0.5	440	1.3
18	1,471	1,471	5.0	490	0.5	981	0.9
19	1,471	1,471	5.0	490	0.5	981	0.9
20	1,471	1,471	5.0	490	0.5	981	0.9
21	1,471	1,471	3.4	490	0.5	981	2.9
22	1,471	1,471	3.4	490	0.5	981	2.9
23	1,630	1,630	2.7	543	0.5	1,087	3.0
24	1,630	1,630	2.7	543	0.5	1,087	3.0
25	1,630	1,630	2.7	543	0.5	1,087	3.0
26	1,630	1,630	2.7	543	0.5	1,087	3.0
27	1,630	1,630	2.7	543	0.5	1,087	3.0
28	733	733	1.5	244	1.0	489	0.7
29	733	733	1.5	244	1.0	489	0.7
30	733	733	1.5	244	2.5	489	0.7
	23,795	23,795		7,926		15,869	

Mining and Tailings Disposal - Year 13
(Techman Drawing No. D22918-70-00)

By Year 5 the mine was sufficiently advanced to allow the backfilling of the pit to begin. Overburden suitable for dyke construction was separated by the spreader, trucked to the dyke, placed and compacted. In Year 9 the sludge treatment operation was started. Prepared soil blending piles were progressively constructed outside of the pit limits from muskeg obtained from the muskeg stripping operation and overburden selected from the backfilling operation. Reclamation of the backfill and tailings pond dyke slopes started in Year 11. In Year 16 the reclamation of the beach commences.

Mining and Tailings Disposal - Year 22
(Techman Drawing No. D22918-71-00)

The out-of-pit tailings pond has been gradually sanded-in, so that by Year 22 the area of the wet pond surface is considerably reduced and clarification of pond water may become slightly more difficult. However, at the Improved Level the return of water from the sludge treatment plant may aid in clarification of pond water. The overburden has been backfilled to a mean elevation of 308 m by a single spreader operating in both the high and low dumping modes. Reclamation is current, with the placement of prepared soil progressing with the advances of the backfill and tailings sand beach.

Material Distribution Plan
(Techman Drawing No. D22916-72-00)

This plan shows the types of materials that must be surfaced with prepared soil. The sludge pond surface is currently considered unreclaimable. During the course of mining, toxic materials are buried. In this plan most of the muskeg stored in the muskeg dumps is consumed in the manufacture of prepared soil. The layered materials from the prepared soil blending pile are distributed by trucks. After completion of overburden removal, loading of prepared soil is accomplished with front-end loaders. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-73-00)

The surfaces to be reclaimed as well as the time period during which reclamation occurred are shown. A pie-shaped end-pit lake is formed. Runoff from the backfill and tailings pond are channelled into this lake. Table 8.3.2-2 is a detailed schedule for the reclamation activities on a year-by-year basis. Plant species are selected according to the reclamation objectives for the Improved Level as described in Chapter 4.0.

8.4 COST SUMMARIES FOR 60,000 BPCD MINE PLANS

Cost summaries are provided for each of the mine plans detailed in this chapter. Rather than following immediately behind the description of the mine plan, the summary tables are grouped at this point in the report for ease of reading and comparison.

Four tables (Tables 8.4-1 to 8.4-4) summarize the quantities, unit costs and $/bbl costs of both capital and operating items. Further details for each cost summary are provided on an annual basis in Volume III. A comparison between the mines costed in this and other chapters of this report follows in Section 10.7.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), DRAGLINE SCHEME, MINIMUM LEVEL

TABLE 8.4-1

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0681
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0332	0.0332	-	-	-	0.0168	0.0168	0.0501
1.2 Buildings	6,171.0 persons	-	1,000.00 $/person	0.0116	-	0.0116	228.0 persons	15,000.00 $/person	0.0064	-	0.0064	0.0180
COST CENTRE 2: Removal of Organic Materials & Soils												0.0590
2.1 Clearing	2,766.1 hectares	-	1,266.82 $/ha	0.0066	-	0.0066	2,766.1 hectares	196.85 $/ha	0.0010	-	0.0010	0.0076
2.2 Muskeg Dewatering	686.5 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0024	-	0.0024	686.5 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0004	-	0.0004	0.0028
2.3 Muskeg Loading	9,915.6 1,000 bank m^3		851.40 $/1,000 bank m^3	0.0158		0.0158	9,915.6 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0039		0.0039	0.0197
2.4 Muskeg Hauling (Including Road Maintenance)	9,915.6 1,000 bank m^3	2.8 km	250.80 $/1,000 bank m^3xkm	0.0130	-	0.0130	27,535.5 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0025	-	0.0025	0.0156
2.5 Muskeg Placement	9,915.6 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0089	-	0.0089	9,915.6 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0006	-	0.0006	0.0096
2.6 Muskeg Road Construction	50.1 km	-	33,825.11 $/km	0.0032	-	0.0032	50.1 km	6,915.90 $/km	0.0006	-	0.0006	0.0038
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.2645
3.1 Overburden B.W.E.	229,915.0 1,000 bank m^3	-	177.34 $/1,000 bank m^3	0.0765	-	0.0765	-	-	-	0.0339	0.0339	0.1104
3.2 Oil Sands Draglines & Hoppers	466,379.0 1,000 bank m^3	-	210.91 $/1,000 bank m^3	0.1846	-	0.1846	-	-	-	0.1580	0.1580	0.3426
3.3 B.W.E. (Overburden & Oil Sands)	- 1,000 bank m^3	-	$/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.4 Transport (All Conveyors)	725,024.0 1,000 bank m^3	16,323.48 m	0.02 $/1,000 bank m^3xkm	0.3977	-	0.3977	17,970.0 m	2,941.29 $/m	0.0992	-	0.0992	0.4969
3.5 Placement (Spreaders)	258,645.0 1,000 bank m^3		117.97 $/1,000 bank m^3	0.0573	-	0.0573	-	-	-	0.0216	0.0216	0.0788
3.6 Miscellaneous Equipment	-		-	-	0.1948	0.1948	-	-	-	0.0409	0.0409	0.2357
COST CENTRE 4: Tailings Disposal											-	0.4509
4.1 Area Drainage	89.6 1,000 bank m^3		1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	89.6 1,000 bank m^3	300.95 $/1,000 bank m^3	0.00005	-	0.00005	0.0003
4.2 Clearing	1,228.1 hectares		1,266.82 $/ha	0.0029	-	0.0029	1,228.1 hectares	196.85 $/ha	0.0005	-	0.0005	0.0034
4.3 Construction of Starter Dams & Overburden Dams	23,513.0 1,000 bank m^3		1,067.75 $/1,000 bank m^3	0.0471	-	0.0471	19,149.0 1,000 bank m^3	207.09 $/1,000 bank m^3	0.0074	-	0.0074	0.0546
4.4 Piping of Tailings or Conveying of Dry Tailings	1,058,439.0 1,000 m^3		64.12 $/1,000 m^3	0.1274	-	0.1274	-	-	-	0.0232	0.0232	0.1506
4.5 Tailings Sand Placement into Dyke	55,310.0 1,000 m^3		123.56 $/1,000 m^3	0.0128	-	0.0128	-	-	-	0.0318	0.0318	0.0446
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	425,061.0 1,000 m^3		67.88 $/1,000 m^3	0.0542	-	0.0542	-	-	-	-	-	0.0542
4.7 Recycling of Tailings Water	-		-	-	0.0405	0.0405	-	-	-	0.0152	0.0152	0.0558
4.8 Rehandling of Tailings Sludge	127,975.0 1,000 m^3		33.04 $/1,000 m^3	0.0079	-	0.0079	-	-	-	0.0050	0.0050	0.0129

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), DRAGLINE SCHEME, MINIMUM LEVEL (Continued)

TABLE 8.4-1 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	14.7 km	25,000.00 $/km	0.0007	0.0003	0.0010	0.0010
4.11 Oversize Reject Disposal	24,206.0 1,000 loose m^3	-	1,384.35 $/1,000 loose m^3	0.0629	-	0.0629	24,206.0 1,000 loose m^3	225.67 $/1,000 loose m^3	0.0103	-	0.0103	0.0732
4.12 Oversize Reject Disposal Road Construction	7.3 km	-	33,825.11 $/km	0.0005	-	0.0005	7.3 km	6,915.90 $/km	0.0001	-	0.0001	0.0006
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.0412
5.1 Muskeg Rehandle Loading	2,423.9 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0028	-	0.0028	2,423.9 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0007	-	0.0007	0.0035
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	2,423.9 1,000 bank m^3	2.26 km	225.33 $/1,000 bank m^3xkm	0.0023	-	0.0023	5,482.7 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0004	-	0.0004	0.0028
5.3 Muskeg Rehandle Placement	2,423.9 1,000 bank m^3		163.24 $/1,000 bank m^3	0.0007	-	0.0007	2,423.9 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0001	-	0.0001	0.0009
5.4 Muskeg Rehandle Road Construction	30.6 km	-	33,825.11 $/km	0.0019	-	0.0019	30.6 km	6,915.90 $/km	0.0004	-	0.0004	0.0023
5.5 Overburden Rehandle Loading	2,526.2 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0026	-	0.0026	2,526.2 1,000 bank m^3	131.53 $1,000 bank m^3	0.0006	-	0.0006	0.0032
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	2,526.2 1,000 bank m^3	4.14 km	178.56 $/1,000 bank m^3xkm	0.0035	-	0.0035	10,454.3 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0007	-	0.0007	0.0042
5.7 Overburden Rehandle Placement	2,526.2 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0007	-	0.0007	2,526.2 1,000 bank m^3	22.92 $1,000 bank m^3	0.0001	-	0.0001	0.0008
5.8 Overburden Rehandle Road Construction	3.5 km	-	33,825.11 $/km	0.0002	-	0.0002	3.5 km	6,915.90 $/km	0.00004	-	0.00004	0.0003
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	11,051.6 1,000 bank m^3	-	27.85 $/1,000 bank m^3	0.0006	-	0.0006	11,051.6 1,000 bank m^3	4.36 $/1,000 bank m^3	0.0001	-	0.0001	0.0007
5.11 Prepared Soil Loading, F.E.L. & Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.13 Prepared Soil Placement, Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.14 Prepared Soil Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.15 Seed Bed Preparation, Maintenance	1,841.8 hectares	-	768.05 $/ha	0.0027	0.0200	0.0227	- hectares	- $/ha	-	-	-	0.0227
COST CENTRE 6: Supervision, Technical Services												0.3531
6.1 Equipment Maintenance (Staff only)	2,283.0 persons	-	29,970.00 $/person	0.1341	-	0.1341	-	-	-	-	-	0.1341
6.2 Planning (Staff only)	1,989.0 persons	-	30,090.00 $/person	0.1124	-	0.1124	-	-	-	-	-	0.1124
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.1067	-	0.1067	-	-	-	-	-	0.1067
TOTAL COSTS						1.7534					0.4835	2.2369

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), DRAGLINE SCHEME, IMPROVED LEVEL

TABLE 8.4-2

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0685
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0334	0.0334	-	-	-	0.0171	0.0171	0.0505
1.2 Buildings	66,171.0 persons	-	1,000.00 $/person	0.0116	-	0.0116	228.0 persons	15,000.00 $/person	0.0064	-	0.0064	0.0180
COST CENTRE 2: Removal of Organic Materials & Soils												0.0582
2.1 Clearing	2,645.7 hectares	-	1,266.82 $/ha	0.0059	-	0.0059	2,565.6 hectares	196.85 $/ha	0.0009	-	0.0009	0.0068
2.2 Muskeg Dewatering	686.5 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0024	-	0.0024	686.5 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0004	-	0.0004	0.0028
2.3 Muskeg Loading	9,915.6 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0158	-	0.0158	9,915.6 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0039	-	0.0039	0.0197
2.4 Muskeg Hauling (Including Road Maintenance)	9,915.6 1,000 bank m^3	2.8 km	250.80 $/1,000 bank m^3xkm	0.0130	-	0.0130	27,535.5 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0025	-	0.0025	0.0156
2.5 Muskeg Placement	9,915.6 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0089	-	0.0089	9,915.6 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0006	-	0.0006	0.0096
2.6 Muskeg Road Construction	50.1 km	-	33,825.11 $/km	0.0032	-	0.0032	50.1 km	6,915.90 $/km	0.0006	-	0.0006	0.0038
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.2682
3.1 Overburden B.W.E.	229,915.0 1,000 bank m^3	-	177.34 $/1,000 bank m^3	0.0765	-	0.0765	-	-	-	0.0339	0.0339	0.1104
3.2 Oil Sands Draglines & Hoppers	466,379.0 1,000 bank m^3	-	210.91 $/1,000 bank m^3	0.1846	-	0.1846	-	-	-	0.1580	0.1580	0.3426
3.3 B.W.E. (Overburden & Oil Sands)	1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	
3.4 Transport (All Conveyors)	725,024.0 1,000 bank m^3	16,564.17 m	0.02 $/1,000 bank m^3xkm	0.4021	-	0.4021	17,850.0 m	2,941.29 $/m	0.0986	-	0.0986	0.5006
3.5 Placement (Spreaders)	258,645.0 1,000 bank m^3	-	117.97 $/1,000 bank m^3	0.0573	-	0.0573	-	-	-	0.0216	0.0216	0.0788
3.6 Miscellaneous Equipment	-	-	-	-	0.1948	0.1948	-	-	-	0.0409	0.0409	0.2357
COST CENTRE 4: Tailings Disposal											-	1.0428
4.1 Area Drainage	96.8 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	96.8 1,000 bank m^3	300.95 $/1,000 bank m^3	0.00005	-	0.00005	0.0003
4.2 Clearing	1,598.1 hectares	-	1,266.82 $/ha	0.0038	-	0.0038	1,598.1 hectares	196.85 $/ha	0.0005	-	0.0005	0.0044
4.3 Construction of Starter Dams & Overburden Dams	22,138.0 1,000 bank m^3	-	1,051.95 $/1,000 bank m^3	0.0437	-	0.0437	18,048.0 1,000 bank m^3	203.96 $/1,000 bank m^3	0.0069	-	0.0069	0.0506
4.4 Piping of Tailings or Conveying of Dry Tailings	1,058,399.0 1,000 m^3	-	67.34 $/1,000 m^3	0.1338	-	0.1338	-	-	-	0.0241	0.0241	0.1579
4.5 Tailings Sand Placement into Dyke	40,900.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0095	-	0.0095	-	-	-	0.0376	0.0318	0.0470
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	439,471.0 1,000 m^3	-	67.88 $/1,000 m^3	0.0560	-	0.0560	-	-	-	-	-	0.0560
4.7 Recycling of Tailings Water		-	-	-	0.0387	0.0387	-	-	-	0.0090	0.0090	0.0477
4.8 Rehandling of Tailings Sludge	1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), DRAGLINE SCHEME, IMPROVED LEVEL (Continued)

TABLE 8.4-2 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	166,218.3 1,000 m^3	-	1,802.56 $/1,000 m^3	0.5625	-	0.5625	-	-	-	0.0449	0.0449	0.6074
4.10 Power Distribution	-	-	-	-	-		12.0 km	25,000.00 $/km	0.0006	0.0003	0.0008	0.0008
4.11 Oversize Reject Disposal	24,206.0 1,000 loose m^3	-	1,317.55 $/1,000 loose m^3	0.0599	-	0.0599	24,236.0 1,000 loose m^3	225.68 $/1,000 loose m^3	0.0103	-	0.0103	0.0701
4.12 Oversize Reject Disposal Road Construction	7.3 km	-	33,825.11 $/km	0.0005	-	0.0005	7.3 km	6,915.90 $/km	0.0001	-	0.0001	0.0006
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.1799
5.1 Muskeg Rehandle Loading	7,841.7 1,000 bank m^3	-	615.18 $/1,000 bank m^3	0.0091	0.00004	0.0091	7,841.7 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0022	-	0.0022	0.0113
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	7,841.7 1,000 bank m^3	0.61 km	224.53 $/1,000 bank m^3xkm	0.0020	-	0.0020	4,821.3 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0004	-	0.0004	0.0024
5.3 Muskeg Rehandle Placement	7,841.7 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0024	-	0.0024	7,841.7 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0004	-	0.0004	0.0028
5.4 Muskeg Rehandle Road Construction	7.6 km	-	33,825.11 $/km	0.0005	-	0.0005	7.6 km	6,915.90 $/km	0.0001	-	0.0001	0.0006
5.5 Overburden Rehandle Loading	15,640.2 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0159	-	0.0159	15,640.2 1,000 bank m^3	131.53 $1,000 bank m^3	0.0039	-	0.0039	0.0198
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	15,640.2 1,000 bank m^3	1.93 km	178.56 $/1,000 bank m^3xkm	0.0101	-	0.0101	30,154.3 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0019	-	0.0019	0.0120
5.7 Overburden Rehandle Placement	15,640.2 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0041	-	0.0041	15,640.2 1,000 bank m^3	22.92 $1,000 bank m^3	0.0007	-	0.0007	0.0048
5.8 Overburden Rehandle Road Construction	9.4 km	-	33,825.11 $/km	0.0006	-	0.0006	9.4 km	6,915.90 $/km	0.0001	-	0.0001	0.0007
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	23,481.9 1,000 bank m^3	-	211.57 $/1,000 bank m^3	0.0093	-	0.0093	23,481.9 1,000 bank m^3	47.66 $/1,000 bank m^3	0.0021	-	0.0021	0.0114
5.11 Prepared Soil Loading, F.E.L. & Trucks	23,481.9 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0190	-	0.0190	23,481.9 1,000 bank m^3	101.00 $/1,000 bank m^3	0.0051	-	0.0051	0.0240
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	23,481.9 1,000 bank m^3	3.2 km	212.40 $/1,000 bank m^3xkm	0.0300	-	0.0300	75,291.0 1,000 bank m^3xkm	131.26 $/1,000 bank m^3xkm	0.0186	-	0.0186	0.0486
5.13 Prepared Soil Placement, Trucks	23,481.9 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0066	-	0.0066	23,481.9 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0011	-	0.0011	0.0077
5.14 Prepared Soil Road Construction	91.9 km	-	33,825.11 $/km	0.0058	-	0.0058	91.9 km	6,915.90 $/km	0.0012	-	0.0012	0.0070
5.15 Seed Bed Preparation, Maintenance	2,347.8 hectares	-	768.05 $/ha	0.0034	0.0233	0.0267	- hectares	- $/ha	-	-	-	0.0267
COST CENTRE 6: Supervision, Technical Services												0.3531
6.1 Equipment Maintenance (Staff only)	2,283.0 persons	-	29,970.00 $/person	0.1341	-	-	-	-	-	-	-	0.1341
6.2 Planning (Staff only)	1,989.0 persons	-	30,090.00 $/person	0.1124	-	-	-	-	-	-	-	0.1124
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.1067	-	-	-	-	-	-	-	0.1067
TOTAL COSTS						2.4135					0.5574	2.9709

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), B.W.E. SCHEME, MINIMUM LEVEL

TABLE 8.4-3

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0656
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0315	0.0315	-	-	-	0.0160	0.0160	0.0474
1.2 Buildings	6,171.0 persons	-	1,000.00 $/person	0.0117	-	0.0117	228.0 persons	15,000.00 $/person	0.0065	-	0.0065	0.0182
COST CENTRE 2: Removal of Organic Materials & Soils												0.0596
2.1 Clearing	2,766.1 hectares	-	1,266.82 $/ha	0.0066	-	0.0066	2,766.1 hectares	196.85 $/ha	0.0010	-	0.0010	0.0077
2.2 Muskeg Dewatering	686.5 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0024	-	0.0024	686.5 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0004	-	0.0004	0.0028
2.3 Muskeg Loading	9,915.6 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0160	-	0.0160	9,915.6 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0039	-	0.0039	0.0199
2.4 Muskeg Hauling (Including Road Maintenance)	9,915.6 1,000 bank m^3	2.8 km	250.80 $/1,000 bank m^3xkm	0.0131	-	0.0131	27,535.5 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0025	-	0.0025	0.0157
2.5 Muskeg Placement	9,915.6 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0090	-	0.0090	9,915.6 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0007	-	0.0007	0.0097
2.6 Muskeg Road Construction	50.1 km	-	33,825.11 $/km	0.0032	-	0.0032	50.1 km	6,915.90 $/km	0.0007	-	0.0007	0.0039
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.2683
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	751,522.0 1,000 bank m^3	-	177.64 $/1,000 bank m^3	0.2532	-	0.2532	-	-	-	0.1296	0.1296	0.3828
3.4 Transport (All Conveyors)	751,552.0 1,000 bank m^3	16,268.14 m	0.02 $/1,000 bank m^3xkm	0.4476	-	0.4476	17,850.0 m	3,610.44 $/m	0.1222	-	0.1222	0.5699
3.5 Placement (Spreaders)	289,945.0 1,000 bank m^3		121.28 $/1,000 bank m^3	0.0667	-	0.0667	-	-	-	0.0265	0.0265	0.0932
3.6 Miscellaneous Equipment	-	-	-	-	0.1834	0.1834	-	-	-	0.0389	0.0389	0.2224
COST CENTRE 4: Tailings Disposal												0.4444
4.1 Area Drainage	89.6 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	89.6 1,000 bank m^3	300.95 $/1,000 bank m^3	0.00005	-	0.00005	0.0003
4.2 Clearing	1,228.1 hectares	-	1,266.82 $/ha	0.0030	-	0.0030	1,288.1 hectares	196.85 $/ha	0.0005	-	0.0005	0.0034
4.3 Construction of Starter Dams & Overburden Dams	19,433.0 1,000 bank m^3	-	1,105.59 $/1,000 bank m^3	0.0408	-	0.0408	13,710.0 1,000 bank m^3	174.50 $/1,000 bank m^3	0.0045	-	0.0045	0.0453
4.4 Piping of Tailings or Conveying of Dry Tailings	1,047,502.0 1,000 m^3	-	64.79 $/1,000 m^3	0.1287	-	0.1287	-	-	-	0.0234	0.0234	0.1521
4.5 Tailings Sand Placement into Dyke	55,310.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0130	-	0.0130	-	-	-	0.0321	0.0321	0.0451
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	420,114.0 1,000 m^3	-	67.88 $/1,000 m^3	0.0541	-	0.0541	-	-	-	-	-	0.0541
4.7 Recycling of Tailings Water	-	-	-	-	0.0410	0.0410	-	-	-	0.0154	0.0154	0.0563
4.8 Rehandling of Tailings Sludge	126,364.1 1,000 m^3	-	33.47 $/1,000 m^3	0.0080	-	0.0080	-	-	-	0.0050	0.0050	0.0130

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), B.W.E. SCHEME, MINIMUM LEVEL (Continued)

TABLE 8.4-3 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	14.7 km	25,000.00 $/km	0.0007	0.0003	0.0010	0.0010
4.11 Oversize Reject Disposal	23,954.0 1,000 loose m^3	-	1,384.55 $/1,000 loose m^3	0.0629	-	0.0629	23,954.0 1,000 loose m^3	225.65 $/1,000 loose m^3	0.0103	-	0.0103	0.0732
4.12 Oversize Reject Disposal Road Construction	7.3 km	-	33,825.11 $/km	0.0005	-	0.0005	7.3 km	6,915.90 $/km	0.0001	-	0.0001	0.0006
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.0429
5.1 Muskeg Rehandle Loading	2,502.4 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0029	-	0.0029	2,502.4 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0007	-	0.0007	0.0036
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	2,502.4 1,000 bank m^3	2.27 km	232.29 $/1,000 bank m^3xkm	0.0025	-	0.0025	5,677.3 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0005	-	0.0005	0.0030
5.3 Muskeg Rehandle Placement	2,502.4 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0008	-	0.0008	2,502.4 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0001	-	0.0001	0.0009
5.4 Muskeg Rehandle Road Construction	30.1 km	-	33,825.11 $/km	0.0019	-	0.0019	30.1 km	6,915.90 $/km	0.0004	-	0.0004	0.0023
5.5 Overburden Rehandle Loading	2,526.2 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0026	-	0.0026	2,526.2 1,000 bank m^3	131.53 $1,000 bank m^3	0.0006	-	0.0006	0.0032
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	2,526.2 1,000 bank m^3	4.14 km	178.56 $/1,000 bank m^3xkm	0.0035	-	0.0035	10,454.3 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0007	-	0.0007	0.0042
5.7 Overburden Rehandle Placement	2,526.2 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0007	-	0.0007	2,526.2 1,000 bank m^3	22.92 $1,000 bank m^3	0.0001	-	0.0001	0.0008
5.8 Overburden Rehandle Road Construction	3.5 km	-	33,825.11 $/km	0.0002	-	0.0002	3.5 km	6,915.90 $/km	0.00005	-	0.00005	0.0003
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	11,287.2 1,000 bank m^3	-	27.85 $/1,000 bank m^3	0.0006	-	0.0006	11,287.2 1,000 bank m^3	4.36 $/1,000 bank m^3	0.0001	-	0.0001	0.0007
5.11 Prepared Soil Loading, F.E.L. & Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.13 Prepared Soil Placement, Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.14 Prepared Soil Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.15 Seed Bed Preparation, Maintenance	1,881.2 hectares	-	768.05 $/ha	0.0027	0.0211	0.0239	- hectares	- $/ha	-	-	-	0.0239
COST CENTRE 6: Supervision, Technical Services												0.3568
6.1 Equipment Maintenance (Staff only)	2,283.0 persons	-	29,970.00 $/person	0.1355	-	0.1355	-	-	-	-	-	0.1355
6.2 Planning (Staff only)	1,989.0 persons	-	30,090.00 $/person	0.1135	-	0.1135	-	-	-	-	-	0.1135
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.1078	-	0.1078	-	-	-	-	-	0.1078
TOTAL COSTS						1.7932					0.4445	2.2377

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), B.W.E. SCHEME, IMPROVED LEVEL

TABLE 8.4-4

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0657
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0316	0.0316	-	-	-	0.0160	0.0160	0.0475
1.2 Buildings	6,171.0 persons	-	1,000.00 $/person	0.0117	-	0.0117	228.0 persons	15,000.00 $/person	0.0065	-	0.0065	0.0182
COST CENTRE 2: Removal of Organic Materials & Soils												0.0589
2.1 Clearing	2,499.2 hectares	-	1,266.82 $/ha	0.0060	-	0.0060	2,499.0 hectares	196.85 $/ha	0.0009	-	0.0009	0.0069
2.2 Muskeg Dewatering	686.5 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0024	-	0.0024	686.5 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0004	-	0.0004	0.0028
2.3 Muskeg Loading	9,915.6 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0160	-	0.0160	9,915.6 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0039	-	0.0039	0.0199
2.4 Muskeg Hauling (Including Road Maintenance)	9,915.6 1,000 bank m^3	2.8 km	250.80 $/1,000 bank m^3xkm	0.0132	-	0.0132	27,535.5 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0025	-	0.0025	0.0157
2.5 Muskeg Placement	9,915.6 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0090	-	0.0090	9,915.6 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0007	-	0.0007	0.0097
2.6 Muskeg Road Construction	50.1 km	-	33,825.11 $/km	0.0032	-	0.0032	50.1 km	6,915.90 $/km	0.0007	-	0.0007	0.0039
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.2807
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	751,522.0 1,000 bank m^3	-	177.64 $/1,000 bank m^3	0.2532	-	0.2532	-	-	-	0.1296	0.1296	0.3828
3.4 Transport (All Conveyors)	751,552.0 1,000 bank m^3	16,506.67 m	0.02 $/1,000 bank m^3xkm	0.4540	-	0.4540	18,730.0 m	3,610.44 $/m	0.1283	-	0.1283	0.5823
3.5 Placement (Spreaders)	289,945.0 1,000 bank m^3	-	121.28 $/1,000 bank m^3	0.0667	-	0.0667	-	-	-	0.0265	0.0265	0.0932
3.6 Miscellaneous Equipment	-	-	-	-	0.1834	0.1834	-	-	-	0.0389	0.0389	0.2224
COST CENTRE 4: Tailings Disposal											-	1.0545
4.1 Area Drainage	96.8 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.0002	-	0.0002	96.8 1,000 bank m^3	300.95 $/1,000 bank m^3	0.0001	-	0.0001	0.0003
4.2 Clearing	1,598.1 hectares	-	1,266.82 $/ha	0.0038	-	0.0038	1,598.1 hectares	196.85 $/ha	0.0006	-	0.0006	0.0044
4.3 Construction of Starter Dams & Overburden Dams	22,138.0 1,000 bank m^3	-	1,051.95 $/1,000 bank m^3	0.0442	-	0.0442	18,048.0 1,000 bank m^3	203.96 $/1,000 bank m^3	0.0070	-	0.0070	0.0512
4.4 Piping of Tailings or Conveying of Dry Tailings	1,047,502.0 1,000 m^3	-	68.18 $/1,000 m^3	0.1355	-	0.1355	-	-	-	0.0243	0.0243	0.1598
4.5 Tailings Sand Placement into Dyke	40,900.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0096	-	0.0096	-	-	-	0.0379	0.0379	0.0475
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	434,824.0 1,000 m^3	-	67.88 $/1,000 m^3	0.0560	-	0.0560	-	-	-	-	-	0.0560
4.7 Recycling of Tailings Water	-	-	-	-	0.0392	0.0392	-	-	-	0.0091	0.0091	0.0482
4.8 Rehandling of Tailings Sludge	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 4 (60,000 BPCD), B.W.E. SCHEME, IMPROVED LEVEL (Continued)

TABLE 8.4-4 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	164,504.0 1,000 m^3	-	1,821.33 $/1,000 m^3	0.5683	-	0.5683	-	-	-	0.0454	0.0454	0.6137
4.10 Power Distribution	-	-	-	-	-	-	12.0 km	25,000.00 $/km	0.0006	0.0002	0.0008	0.0008
4.11 Oversize Reject Disposal	23,954.0 1,000 loose m^3	-	1,361.85 $/1,000 loose m^3	0.0619	-	0.0619	23,954.0 1,000 loose m^3	225.65 $/1,000 loose m^3	0.0103	-	0.0103	0.0721
4.12 Oversize Reject Disposal Road Construction	4.8 km	-	33,825.11 $/km	0.0003	-	0.0003	4.8 km	6,915.90 $/km	0.0001	-	0.0001	0.0004
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.1846
5.1 Muskeg Rehandle Loading	7,953.7 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0093	-	0.0093	7,953.7 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0022	-	0.0022	0.0115
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	7,953.7 1,000 bank m^3	0.67 km	224.53 $/1,000 bank m^3xkm	0.0021	-	0.0021	4,899.6 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0004	-	0.0004	0.0025
5.3 Muskeg Rehandle Placement	7,953.7 1,000 bank m^3	-	191.11 $/1,000 bank m^3	0.0029	-	0.0029	7,953.7 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0004	-	0.0004	0.0033
5.4 Muskeg Rehandle Road Construction	7.6 km	-	33,825.11 $/km	0.0005	-	0.0005	7.6 km	6,915.90 $/km	0.0001	-	0.0001	0.0006
5.5 Overburden Rehandle Loading	15,864.2 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0163	-	0.0163	15,864.2 1,000 bank m^3	131.53 $1,000 bank m^3	0.0040	-	0.0040	0.0203
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	15,864.2 1,000 bank m^3	1.91 km	178.56 $/1,000 bank m^3xkm	0.0102	-	0.0102	30,266.3 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0020	-	0.0020	0.0122
5.7 Overburden Rehandle Placement	15,864.2 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0042	-	0.0042	15,864.2 1,000 bank m^3	22.92 $1,000 bank m^3	0.0007	-	0.0007	0.0049
5.8 Overburden Rehandle Road Construction	9.4 km	-	33,825.11 $/km	0.0006	-	0.0006	9.4 km	6,915.90 $/km	0.0001	-	0.0001	0.0007
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	23,817.9 1,000 bank m^3	-	211.57 $/1,000 bank m^3	0.0096	-	0.0096	23,817.9 1,000 bank m^3	47.66 $/1,000 bank m^3	0.0021	-	0.0021	0.0117
5.11 Prepared Soil Loading, F.E.L. & Trucks	23,817.9 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0194	-	0.0194	23,817.9 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0052	-	0.0052	0.0246
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	23,817.9 1,000 bank m^3	3.19 km	212.40 $/1,000 bank m^3xkm	0.0306	-	0.0306	75,862.2 1,000 bank m^3xkm	131.26 $/1,000 bank m^3xkm	0.0189	-	0.0189	0.0495
5.13 Prepared Soil Placement, Trucks	23,817.9 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0068	-	0.0068	23,817.9 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0011	-	0.0011	0.0079
5.14 Prepared Soil Road Construction	91.9 km	-	33,825.11 $/km	0.0059	-	0.0059	91.9 km	6,915.90 $/km	0.0012	-	0.0012	0.0071
5.15 Seed Bed Preparation, Maintenance	2,381.6 hectares	-	768.05 $/ha	0.0035	0.0243	0.0278	- hectares	- $/ha	-	-	-	0.0278
COST CENTRE 6: Supervision, Technical Services												0.3568
6.1 Equipment Maintenance (Staff only)	2,283.0 persons	-	29,970.00 $/person	0.1355	-	0.1355	-	-	-	-	-	0.1355
6.2 Planning (Staff only)	1,989.0 persons	-	30,090.00 $/person	0.1135	-	0.1135	-	-	-	-	-	0.1135
6.3 Mining (Staff only)	1,891.0 persons	-	30,065.00 $/person	0.1078	-	0.1078	-	-	-	-	-	0.1078
TOTAL COSTS						2.4725					0.5288	3.0012

NOTE: Refer to Chapter 6 for Cost Centre description.

9.0 CONCEPTS AND COSTS OF DEVELOPMENT AND RECLAMATION OF 240,000 BPCD OIL SANDS MINE

9.1 OVERVIEW OF DEVELOPMENT PLANS

This subsection is to be read in conjunction with the drawings supplied in Volume II for the 240,000 BPCD mine plan.

Ore Body No. 1 lends itself best to a combination of slewing and parallel mining. The high level of production inherent in the 240,000 BPCD mine size dictates that a large surface area be involved. In Chapter 2.0, it was shown that very few areas have the potential for this size of mine. Therefore, generally speaking, enough smaller areas must be grouped together to form one large mine. Such a grouping results in a much more irregularly shaped mine than would be the case if the mine were developed from a single ore body.

The large areal extent of a 240,000 BPCD oil sands mine makes development of a mine with only one advancing face very impractical. For example, a bucket wheel mine would require six excavators on as many benches to achieve this level of production. Extremely low working faces would be created, which would lead to very inefficient digging and excessive conveyor shifting. The practical solution is to develop Ore Body No. 1 using two adjacent mines of about half the size, i.e., approximately 120,000 BPCD each. A mine layout combining both slewing and parallel development was selected.

Ore Body No. 1 does not lend itself to dragline mining because of high production requirements combined with a rather complicated shape. The requirement of eight draglines for oil sands removal plus 2 BWE's for overburden removal, combined with all the associated conveyors, makes the operation technically impractical and economically unattractive. The development of Ore Body No. 1 in the parallel fashion, as used in Ore Body No. 2, is also not technically advisable since parallel dragline systems are desirable only if the ore body is rectilinear in shape, or is formed by a combination of distinct rectilinear units. Consequently only bucket wheel mining plans have been developed for Ore Body No. 1.

The mine layout for the north half of the ore body (Part B) involves both slewing and parallel mining techniques. The pit is started in the northeast section of the mine with a slewing layout, then switched to a parallel layout in the northern section, then back to a slewing layout in the western sector, and finally, again to parallel in the southern sector. The centrally located island of low grade oil sands is not mined. The island serves both as conveyor distribution point and a dyke for in-pit tailings disposal ponds. If desired, the mine plan may be altered to allow the mining of a large portion of this island, but such a strategy ignores the ore body selection criteria set early in the study.

The south pit (Part A) is also developed with an alternating slewing and parallel mining scheme. The mine is developed from the west as a slewing operation, but is converted to a parallel system as the first pit opens up. As the neck in the mine is approached, a slewing operation is again used, and continues in use until the mining face has passed the neck. A parallel layout is used in the remainder of the mine.

The described mining sequence is applicable for both the Minimum and the Enhanced Level plans. However, the disposition of overburden and tailings is quite different for the two plans.

At the Minimum Level, a large out-of-pit pond is required. The location of the pond south of the ore body avoids potentially mineable areas. Compared to the pond used for Ore Body No. 2, less dyke volume for an equal storage capacity is needed but the total land surface area of the outside pond is substantially increased. This results in an increased length of tailings lines, and consequently, a general increase in the cost of tailings slurry transport. A large sanded-in pond with adequate drainage, which has been surfaced with prepared soil, is not in itself considered to have a detrimental impact on the environment of the region. The area of the sanded-in pond could be reduced by raising the dyke height; however, a completely encircling dyke would be required, similar to that used for the 120,000 BPCD plans. This option has not been utilized in the 240,000 BPCD plan in order to keep the dyke within the general range of dyke heights utilized in the other plans developed for this study; because a large surface area was available for the tailings pond in this plan, the most economical pond design has been employed.

The north mine (Part B) is developed to include two in-pit tailings ponds. As well, a portion of the mine is used to backfill overburden. The remaining unfilled portion is developed as a lake.

The south mine (Part A) includes a sludge pond which is sized to contain the sludge from the out-of-pit pond, as well as the sludge from the two in-pit ponds located in the north mine. As in the north mine, the void volume not required for backfilled overburden is developed into a lake.

It is possible in most cases to convert a lake formed in an end-pit into a tailings pond for an adjacent mine. Complete dewatering prior to the introduction of tailings is likely unnecessary. The backfilling with dry tailings of the lake formed in the end-pit is more difficult since the lake would have to be completely dewatered. Pit slope failures may result when the lake is lowered. A period for drying out is likely required, as is the reinstallation of peripheral pit dewatering wells. It may also be possible to use the end pit lake as a make-up water reservoir for a neighboring mine.

The disposal sequence for wet tailings of such a large mine is relatively complicated. For eleven years all the tailings are pumped into the out-of-pit tailings pond. The out-of-pit pond is operated in the conventional manner until Year 11.

In the summer months of Year 11, 50 % of the tailings stream is diverted to construct the first in-pit dyke. By Year 12, the entire flow of extraction plant tailings is diverted into the first in-pit pond. This pond is initially operated from Year 11.0 to Year 13.5. In Year 14, the second in-pit dyke is started. Initially the second in-pit pond is operated in a conventional manner from Year 13.5 to Year 17.4. Both of the in-pit dykes require one year for starter dyke construction with overburden material, followed by two years of dyke construction with hydraulically-placed and compacted tailings sand.

In Year 15, a portion of the tailings is diverted from the second pond for the purpose of constructing the sludge pond dyke in the south mine. Two seasons are required to construct the sand portion of the sludge pond dyke. In the meantime, most of the tailings are deposited in the second in-pit pond.

Upon completion of the dyke in the second pond, the entire flow of tailings is again directed into the out-of-pit tailings pond until it is sanded-in in Year 22.4. Then the first in-pit pond is sanded-in between Year 22.4 and 23.4, and the second in-pit pond between Year 23.4 and 26.0. Sludge is pumped into the sludge pond beginning in Year 14.5; this continues until the completion of mining.

The backfilling operation at the Minimum Level requires two spreaders for each pit, or four in total. These machines are initially used to construct the three outside dumps located south, west, and east of the ore body. As room is developed in the pit between the last in-pit dyke and the toe of the lowest mining bench, the spreaders are moved into the pit one at a time. Initially the first relocated spreaders assist in constructing the last in-pit dykes. The second spreader is relocated when additional room is developed in each pit. As the north pit is developed, the outside dump located west of the mine becomes inaccessible. At that time a small dump must be operated east of the mine until backfilling into the pit can be started.

The plan for backfilling with dry tailings at the Enhanced Level proceeds in much the same pattern as the mining. In fact, backfilling follows the mining rather closely. Six spreaders are utilized. The toe of the backfilled tailings sand, overburden, and reject is an average of about 100 meters from the toe of the deep cut portion of the lowest mining bench. The backfilling operation changes alternately from slewing to parallel, or vice versa, as the case may be. The elevation of backfill is above the prevailing elevation of the landscape prior to mining because of the swell of the backfilled materials. The mine cannot be completely backfilled without rehandling large volumes of material. Consequently, the plans show a residual void upon completion of mining. This void is smaller than that created when a wet tailings disposal system is used.

The flexibility of the Enhanced Level, with respect to being able to "landscape" the backfill, is an order of magnitude greater than that of the Minimum or Improved Levels of Reclamation. The size of outside dumps can be reduced by blending into the backfilled areas. At the Minimum Level the dumps are rather massive and, because they are restricted in height to about 75 meters, they cover considerable area. At the Enhanced Level this type of impact is greatly reduced.

The layout of the outside waste dumps of the Ore Body No. 1 mine reflects the interference from Ore Body No. 2. As more mines are developed, this type of interference may increase, having considerable influence on the economics of any particular mine. It should be noted that the Consultants did not take the liberty of developing an integrated, optimized development plan for Ore Bodies No. 1 and 2, although it appears that this would be possible.

The major operational differences between the two mine plans can be seen in the two sets of drawings provided in Volume II. Mass balance schedules are provided for overall mining, tailings disposal, and reclamation in subsection 9.2. Schedules for various other items are provided in the computer-printed cost summaries. Cost estimates for selected operating activities for a period of 35 years are provided, and include five years of preproduction and five years of deactivation of the mine. Summary cost comparisons are made in subsection 9.3.

9.2 MINE PLANS EMPLOYING BUCKET WHEEL EXCAVATORS

Two bucket wheel mine plans have been prepared for Ore Body No. 1. The mine layouts of plans for the Minimum and Enhanced Levels of Ore Body No. 1 are very similar, the only major difference being the backfilling of the mine. At the Minimum Level the pit is used for disposal of tailings sand and sludge, as well as overburden and reject. At the Enhanced Level the backfill is a mixture of dry tailings sand, overburden, and reject for the entire life of the mine.

The size of Ore Body No. 1 has necessitated that the ore body be mined in two parts: in effect, as two mines. Consequently, two separate conveying systems are required to provide plant feed to the extraction plant. As well, two separate waste disposal systems are utilized.

The distribution point for the southern mine is located west of the plant at the most southerly extension of the future sludge pond. From this position, three mining faces are developed, starting in a slewing fashion and continuing later in a parallel fashion. Three trunk conveyors are to be located on pitwall berms to transfer material in a southwesterly direction to the conveyor distribution point.

For the plan at the Minimum Level, two plant feed conveyors are utilized. Once the second distribution point is in operation to the northeast of the first, the in-pit dyke for the sludge pond can be constructed. Rather than continuing with the transport of overburden reject to the outside dump, the two waste conveyors and two spreaders are moved into position in the pit. The first spreader working in-pit separates overburden materials suitable for in-pit starter dyke construction during the course of general backfilling. The first spreader is moved in Year 14, and the second early in Year 15. Prior to the relocation of the spreader, a waste dump is operated immediately south of the mine.

At the Enhanced Level, no in-pit dyke is required. Therefore, once the three spreaders have been relocated in the pit from the outside dump, backfilling may continue through the neck in the mine without the scheduling constraints imposed by the construction of an in-pit dyke.

The large outside waste dumps are not required at the Enhanced Level, because backfill at the Enhanced Level is more extensive than at the Minimum Level.

The layout of the excavating and conveying systems in the northern half of the mine is, in essence, very similar to that in the southern half. At the Minimum Level, three BWE's operate on three mining faces. The distribution point directs waste to two overburden belts which lead to two spreaders placing the materials on an outside dump west of the mine. Plant feed is moved via two conveyors.

By Year 11 the mining faces in the north mine have advanced into the western portion of the mine. At this time, the waste conveyors will be relocated to a small waste dump east of the mine. The second in-pit tailings dyke is started in Year 14 with overburden placed by the first spreader, which has been moved into the pit to begin backfill operations. Nine months later the second spreader is relocated into the pit, and starts depositing backfill on the lower backfill bench formed by the first spreader. All the above manoeuvres can be done by utilizing one centrally located distribution point.

At the Enhanced Level the mine is operated in a similar fashion, in that the faces are three in number, and advance at the same rates as at the Minimum Level. However, the outside waste dump located east of the mine is much smaller, and no scheduling constraints are imposed, since no in-pit dykes are utilized. Again, the backfilling of the mine extends further into the mined-out pit than at the Minimum Level.

At the Enhanced Level, both pits have two dry tailings sand conveyors returning the sand from the extraction plant. At the distribution point the flow of dry sand is distributed to two of the three waste conveyors. The distribution point is designed in such a way that overburden or centre reject is dumped on top of the dry tailings sand being carried by the conveyors. The two materials undergo mixing at the next drive station as they fall from one belt to another. The mixture of dry tailings sand and overburden/centre reject will handle better than dry tailings sand alone as far as dump stability and dusting are concerned.

The materials are distributed by shunting any of the three conveyors coming from the pit, so that they dump on a waste conveyor receiving a partial loading of dry tailings sand. The height of the backfilling operation can be varied regionally by appropriate benching of the three spreaders or locally by varying the high dump thickness of the uppermost spreader to produce the desired relief.

The production schedules for both types of mines are identical, and are detailed in Tables 9.2-1, 9.2-2 and 9.2-3. The materials handling systems are described in Figures 9.2-1 and 9.2-2. Details for the tailings disposal and reclamation appear in the following subsections.

Ore Body No.1, 240,000 B.P.C.D. - Production Schedule -
(Mine A and Mine B)
TABLE 9.2-1

Year	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Bitumen tonnes $\times 10^6$	Crude barrels $\times 10^6$
-2	22.076	-	22.076	-	-
-1	37.323	-	37.323	-	-
1	38.684	31.019	69.703	6.936	33.343
2	53.429	73.479	126.908	16.954	82.457
3	53.123	79.473	132.596	18.539	90.557
4	51.784	81.804	133.588	18.863	91.952
5	47.956	82.225	130.181	18.465	89.247
6	48.701	82.628	131.329	18.385	88.620
7	48.493	84.428	132.921	19.086	92.564
8	47.066	83.711	130.777	19.038	92.559
9	53.844	78.050	131.894	17.712	86.096
10	55.169	76.800	131.969	17.371	84.367
11	55.404	76.571	131.975	17.239	83.514
12	54.114	77.862	131.976	17.132	82.244
13	56.404	75.547	131.951	16.777	80.329
14	52.755	79.195	131.950	17.769	84.694
15	51.683	80.336	132.019	18.134	86.557
16	49.506	83.237	132.743	18.927	90.799
17	53.978	79.211	133.189	18.067	87.461
18	50.248	79.847	130.095	18.232	88.436
19	45.949	85.754	131.703	19.016	91.530
20	45.500	86.450	131.950	18.717	89.424
21	45.430	86.521	131.951	18.609	88.620
22	47.157	84.797	131.954	18.600	89.163
23	49.345	82.611	131.956	18.683	90.077
24	48.255	72.897	121.152	16.623	80.046
25	16.914	56.004	72.918	12.858	61.782
26	3.699	25.690	29.389	6.477	31.665
Total	1,283.989	1,966.147	3,250.136	443.209	2,138.103

TABLE 9.2-2

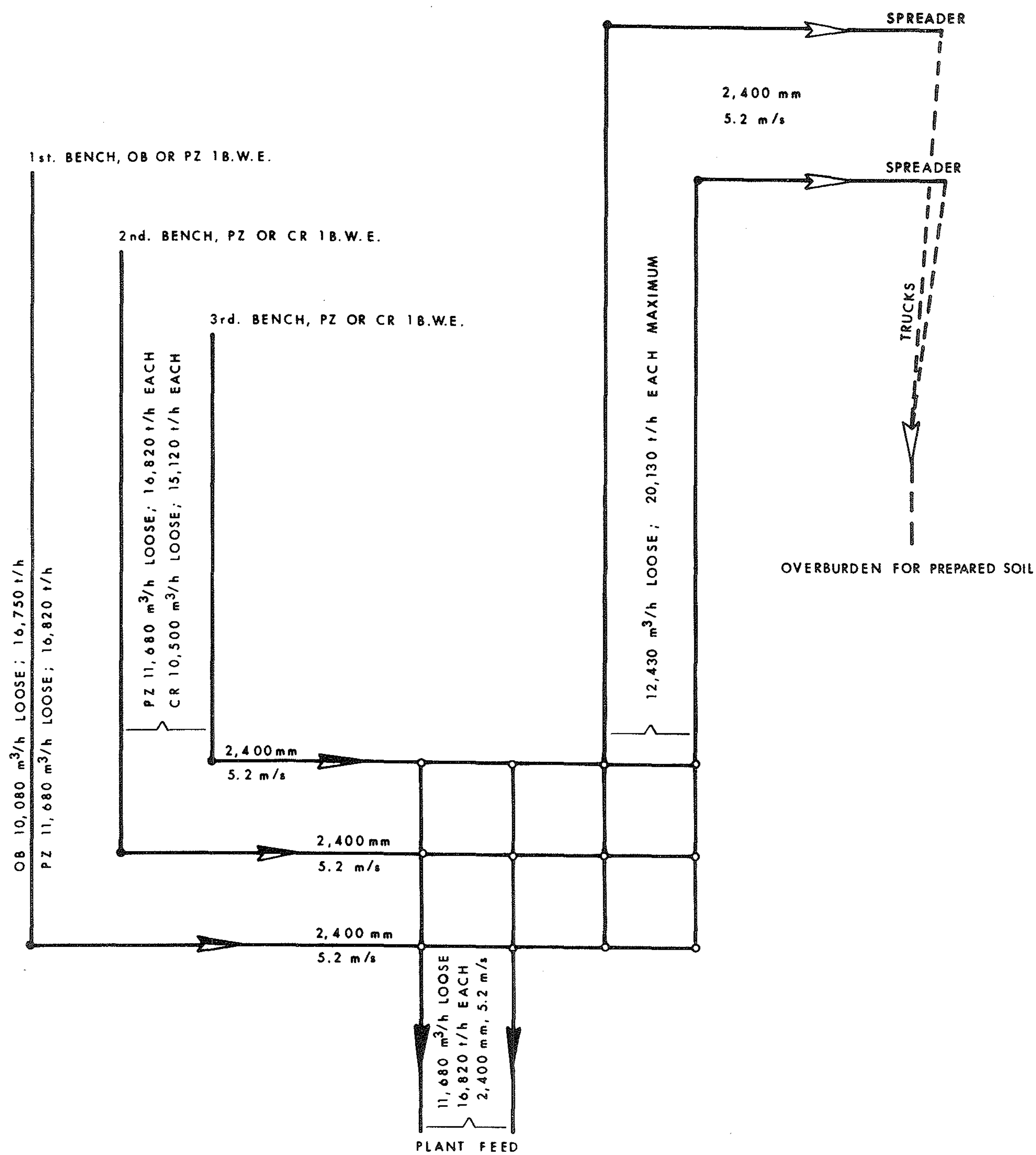
Ore Body No.1 , 240,000 B.P.C.D., Part A - Production Schedule, 6 Bucket Wheel Excavators Scheme

Year	Top Bench			Middle Bench			Bottom Bench			Mine				
	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Bitumen tonnes x 10^6	Crude barrels x 10^6
- 2	–	–	–	–	–	–	–	–	–	–	–			
- 1	15.246	–	15.246	–	–	–	–	–	–	15.246	–	15.246		
1	16.726	4.322	21.048	2.763	8.298	11.061	0,561	1.702	2.263	20.050	14.322	34.372	3.458	16.990
2	18.995	3.071	22.066	5.434	16.688	22.122	4.872	14.766	19.638	29.301	34.525	63.826	8.338	40.962
3	19.247	2.967	22.214	3.414	18.708	22.122	4.606	17.516	22.122	29.267	39.191	66.458	9.400	46.063
4	18.860	3.149	22.009	2.578	19.544	22.122	3.081	19.040	22.121	24.519	41.733	66.252	9.714	47.310
5	17.826	3.980	21.806	1.818	20.305	22.123	2.011	20.111	22.122	21.655	44.396	66.051	9.932	47.848
6	17.236	4.698	21.934	1.082	21.039	22.121	1.404	20.717	22.121	19.722	46.454	66.176	10.331	49.768
7	17.304	5.675	22.979	1.338	20.780	22.118	1.132	20.989	22.121	19.774	47.444	67.218	10.595	51.161
8	15.311	5.524	20.835	1.485	20.631	22.116	1.485	20.637	22.122	18.281	46.792	65.073	10.522	50.954
9	19.595	2.405	22.000	2.253	19.812	22.065	1.857	20.265	22.122	23.705	42.482	66.187	9.528	46.108
10	19.395	2.604	21.999	2.745	19.395	22.140	2.485	19.637	22.122	24.625	41.636	66.261	9.324	45.102
11	18.820	3.178	21.998	2.786	19.360	22.146	2.783	19.339	22.122	24.389	41.877	66.266	9.424	45.619
12	18.494	3.506	22.000	2.950	19.196	22.146	2.783	19.339	22.122	24.227	42.041	66.268	9.509	46.091
13	17.492	4.509	22.001	6.077	16.046	22.123	4.652	17.470	22.122	28.221	38.025	66.246	9.069	44.325
14	17.418	4.583	22.001	4.736	17.385	22.121	5.329	16.793	22.122	27.483	38.761	66.244	9.778	48.044
15	17.543	4.458	22.001	3.253	18.869	22.122	4.654	17.468	22.122	25.450	40.795	66.245	10.305	50.624
16	17.756	4.244	22.000	2.220	19.902	22.122	1.989	20.132	22.121	21.965	44.278	66.243	10.948	53.672
17	17.801	4.199	22.000	3.108	19.014	22.122	2.768	19.354	22.122	23.677	42.567	66.244	9.799	47.597
18	18.140	3.860	22.000	2.570	19.552	22.122	3.108	19.014	22.122	23.818	42.426	66.244	9.528	46.046
19	18.176	3.824	22.000	1.609	20.513	22.122	1.835	20.287	22.122	21.620	44.624	66.244	9.639	46.000
20	18.601	3.400	22.001	1.280	20.843	22.123	1.609	20.513	22.122	21.490	44.756	66.246	9.638	45.959
21	18.817	3.183	22.000	0.937	21.185	22.122	0.950	21.172	22.122	20.704	45.540	66.244	9.825	46.871
22	18.883	3.117	22.000	1.265	20.857	22.122	0.963	21.159	22.122	21.111	45.133	66.244	9.698	46.166
23	20.407	1.593	22.000	1.530	20.593	22.123	1.530	20.593	22.123	23.467	42.779	66.246	9.097	43.138
24	20.407	1.593	22.000	2.009	20.113	22.122	1.575	20.547	22.122	23.991	42.253	66.244	8.943	42.329
25	4.916	0.384	5.300	1.765	14.636	16.401	2.380	19.741	22.121	9.061	34.761	43.822	7.292	34.383
26		–	–	–	–	–	0.598	4.963	5.561	0.598	4.963	5.561	1.037	4.882
Total	459.412	88.026	547.438	63.005	473.264	536.269	63.000	473.264	536.264	585.417	1,034.554	1,619.971	234.671	1,134.012

TABLE 9.2-3

Ore Body No.1 , 240,000 B.P.C.D., Part B - Production Schedule, 6 Bucket Wheel Excavators Scheme

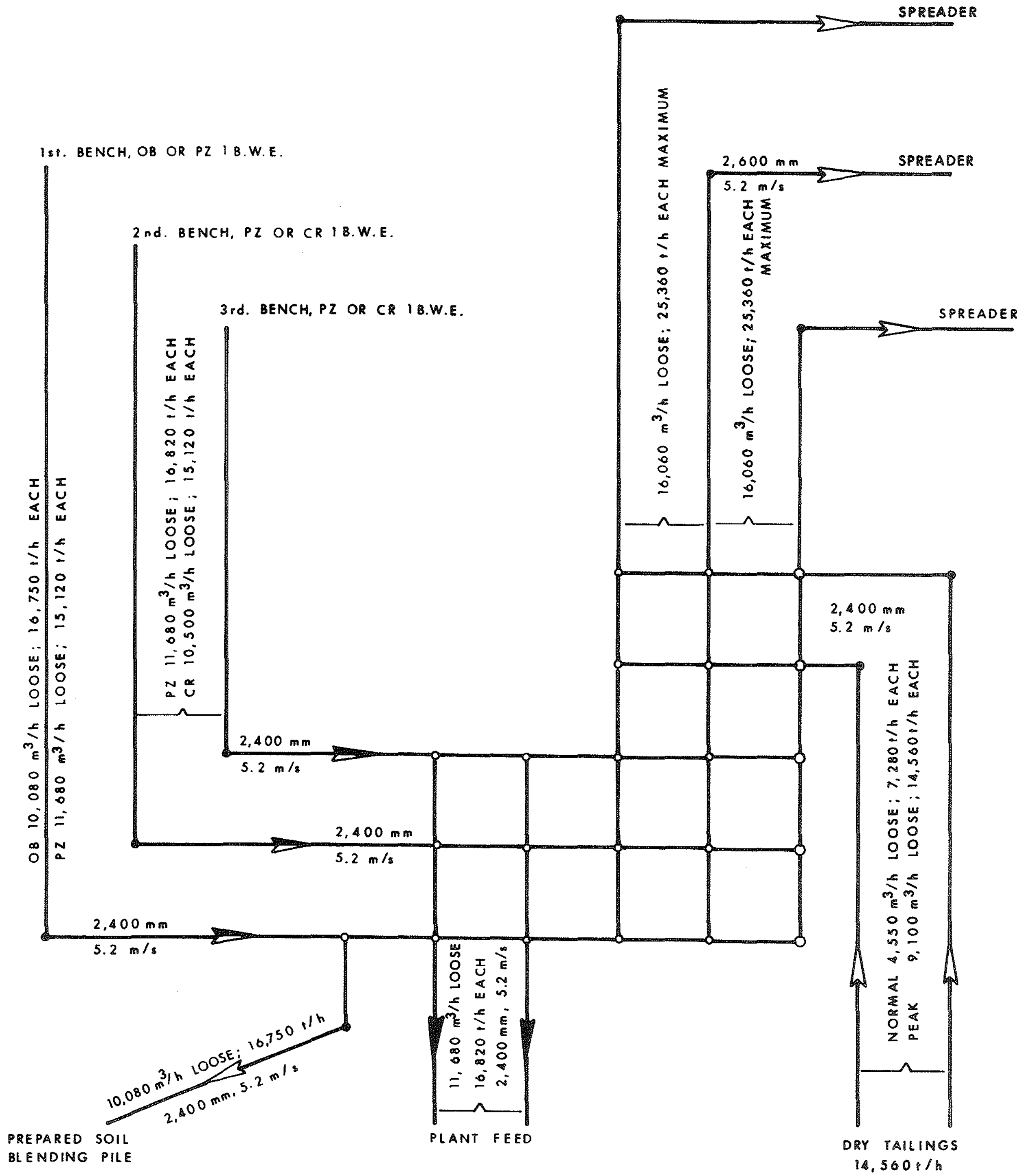
Year	Top Bench			Middle Bench			Bottom Bench			Mine				
	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Waste $bm^3 \times 10^6$	Feed $bm^3 \times 10^6$	Total $bm^3 \times 10^6$	Bitumen tonnes x 10^6	Crude barrels x 10^6
- 2	22.076	-	22.076	-	-	-	-	-	-	22.076	-	22.076	-	-
- 1	22.077	-	22.077	-	-	-	-	-	-	22.077	-	22.077	-	-
1	16.469	5.531	22.000	1.843	9.225	11.068	0.322	1.941	2.263	18.634	16.697	35.331	3.478	16.353
2	18.846	3.154	22.000	2.529	19.182	21.711	2.753	16.618	19.371	24.128	38.954	63.082	8.616	41.495
3	20.005	2.427	22.432	3.064	18.789	21.853	2.787	19.066	21.853	25.856	40.282	66.138	9.139	44.494
4	21.002	2.628	23.630	3.147	18.706	21.853	3.116	18.737	21.853	27.265	40.071	67.336	9.149	44.642
5	18.275	2.149	20.424	4.455	17.398	21.853	3.571	18 282	21.853	26.301	37.829	64.130	8.533	41.399
6	19.902	2.143	21.445	4.797	17.057	21.854	4.880	16.974	21.854	28.979	36.174	65.153	8.054	38.852
7	19.844	2.154	21.998	4.342	17.510	21.852	4.533	17.320	21.853	28.719	36.984	65.703	8.491	41.403
8	19.808	2.191	21.999	4.635	17.218	21.853	4.342	17.510	21.852	28.785	36.919	65.704	8.516	41.605
9	19.809	2.190	21.999	5.250	16.604	21.854	5.080	16.774	21.854	30.139	35.568	65.707	8.184	29.988
10	19.796	2.204	22.000	5.484	16.370	21.854	5.264	16.590	21.854	30.544	35.164	65.708	8.047	39.265
11	19.736	2.267	22.003	5.598	16.255	21.853	5.681	16.172	21.853	31.015	34.694	65.709	7.815	37.895
12	19.691	2.309	22.000	4.954	16.900	21.854	5.242	16.612	21.854	29.887	35.821	65.708	7.623	36.153
13	19.670	2.329	21.999	3.703	18.150	21.853	4.810	17.043	21.853	28.183	37.522	65.705	7.708	36.004
14	19.651	2.349	22.000	2.838	19.015	21.853	2.783	19.070	21.853	25.272	40.434	65.706	7.991	36.650
15	19.700	2.300	22.000	3.388	18.466	21.854	3.145	18.775	21.920	26.233	39.541	65.774	7.829	35.933
16	19.704	2.296	22.000	4.432	18.100	22.532	3.405	18.563	21.968	27.541	38.959	66.500	7.979	37.127
17	19.706	2.293	21.999	5.470	17.736	23.206	5.125	16.615	21.740	30.301	36.644	66.945	8.268	39.864
18	19.702	2.299	22.001	2.540	17.526	20.066	4.188	17.596	21.784	26.430	37.421	63.851	8.704	42.390
19	19.675	2.325	22.000	2.134	19.473	21.607	2.520	19.332	21.852	24.329	41.130	65.459	9.377	45.530
20	19.667	2.332	21.999	2.228	19.625	21.853	2.115	19.737	21.852	24.010	41.694	65.704	9.079	43.465
21	19.765	2.236	22.001	2.672	19.181	21.853	2.289	19.564	21.853	24.726	40.981	65.707	8.784	41.749
22	19.847	2.156	22.003	3.111	18.743	21.854	3.088	18.765	21.853	26.046	39.664	65.710	8.902	42.997
23	19.847	2.156	22.003	2.992	18.862	21.854	3.039	18.814	21.853	25.878	39.832	65.710	9.586	46.939
24	19.847	2.156	22.003	1.839	12.067	13.906	2.578	16.421	18.999	24.264	30.644	54.908	7.680	37.717
25	4.752	0.516	5.268	1.548	10.345	11.893	1.553	10.382	11.935	7.853	21.243	29.096	5.566	27.399
26	-	-	-	1.548	10.345	11.893	1.553	10.382	11.935	3.101	20.727	23.828	5.440	26.783
Total	518.269	59.090	577.359	90.541	438.848	529.389	89.762	433.655	523.417	698.572	931.593	1,630.165	208.538	1,004.091



NOTE: THE SYSTEM IS DUPLICATED IN EACH HALF OF THE 240,000 BPCD MINE

MATERIALS HANDLING SYSTEM - 240,000 BPCD
6 B.W.E. - MINIMUM LEVEL

FIGURE 9.2-1



NOTE: THE SYSTEM IS DUPLICATED IN EACH HALF OF THE 240,000 BPCD MINE

MATERIALS HANDLING SYSTEM - 240,000 BPCD
6 B.W.E. - ENHANCED LEVEL

FIGURE 9.2-2

9.2.1 MINIMUM (WET) LEVEL OF RECLAMATION

The major aspects of the development of the 240,000 BPCD mine at the Minimum Level are depicted by seven drawings, accompanied by tailings disposal and reclamation schedules (See Tables 9.2.1-1 and 9.2.1-2 respectively). The mining schedule, which is common to the plans for both the Minimum and Improved Levels of Reclamation, has been previously illustrated in Table 9.2-1. A drawing-by-drawing discussion follows:

General Mine Layout
(Techman Drawing No. D22910-74-00)

The mine consists of two portions which are operated independently with respect to mining oil sands but in combination with respect to tailings disposal. Mining area "Part A" contains the single sludge pond utilized in this plan. Mining area "Part B" contains tailings sand deposited in two ponds. Both mining areas contain overburden and reject backfill produced within their respective boundaries, as well as a lake formed by combining the final end-pits. The two mining areas are operated simultaneously in conjunction with the out-of-pit tailings pond. Mining area "Part A" utilizes one outside waste dump, while "Part B" requires both a major and a minor outside waste dump. No major river diversions are required by this plan. Kearl Lake is to be drained during the course of mine development, but replaced at the completion of mining by the lake formed by the end-pits.

Mining and Tailings Disposal - Year 1
(Techman Drawing No. D22918-75-00)

By the end of Year 1, all the mining faces are fully developed, and each mine area has 3 BWE's in operation. The waste dumps of each mine are being constructed utilizing two spreaders. After one year of operation, the tailings pond elevation is estimated to be at 339 m. The pond has required the construction of starter dykes in the two flanks of the dyke shown in this drawing. Compared to other plans presented in this report, the distribution points for this mine are located far from the plant site.

Mining and Tailings Disposal - Year 7
(Techman Drawing No. D22918-76-00)

By Year 7, the outside waste dumps are about fifty percent developed. No backfilling of the mine with tailings or overburden has occurred to date. The tailings pond will not reach its maximum crest elevation until another four years have passed. Reclamation of the waste dumps is scheduled to begin in Year 8, with reclamation of tailings dyke slopes and beaches scheduled to start in Year 12. The muskeg stripping operation has resulted in large muskeg dumps being formed north of the mine boundary. Direct deposition of muskeg to the waste dumps is possible, and is practiced in this mine.

Mining and Tailings Disposal - Year 14
(Techman Drawing No. D22918-77-00)

The first in-pit tailings pond was started in Year 11. By Year 14, the second in-pit tailings pond is operational. The in-pit sludge pond dyke has been started. The second in-pit tailings pond remains operational until Year 18. In both mines, overburden is being placed into the pit as backfill. A portion of the overburden is selected for starter dyke construction. Prior to Year 14, the out-of-pit waste dumping operation for the northern mining area was switched to a small dump east of the mine. The advance of the mining faces would otherwise interrupt the overburden conveyors. In the southern mining area, the conveyor distribution point has been relocated, but in the northern mine area, a single conveyor distribution point is maintained throughout the life of the mine.

Mining and Tailings Disposal - Year 22
(Techman Drawing No. D22918-78-00)

The mining faces in both pits are advancing in the same direction, with the overburden backfilling faces being almost parallel to the mining faces. Sludge transfer from the out-of-pit pond was started in Year 15, and continues until Year 23. Sludge transfer from the first in-pit

Ore Body No. 1, 240,000 B.P.C.D. 6 B.W.E.
TAILINGS SCHEDULE
FOR MINIMUM LEVEL OF RECLAMATION
TABLE 9.2.1-1

Outline of Tailings Disposal Scheme:

- Years 1-11 Tailings to out-of-Pit Tailings Pond.
- Years 11-14 Tailings to First in-Pit Tailings Pond.
- Years 14-18 Tailings to Second in-Pit Tailings Pond.
- Years 15-23 Sludge from out-of-Pit Pond to Sludge Pond.
- Years 22-24 Sludge from First in-Pit Tailings Pond to Sludge Pond.
- Years 23-26 Sludge from Second in-Pit Tailings Pond to Sludge Pond.
- Years 18-23 Tailings to out-of-Pit Pond. Fill with Sand.
- Years 23-24 Tailings to First in-Pit Tailings Pond. Fill with Sand.
- Years 24-26 Tailings to Second in-Pit Tailings Pond. Fill with Sand.

YEAR	Volume of Tailings Produced [$m^3 \times 10^6$]	Volume of Recycle Water [$m^3 \times 10^6$]	Volume of Sludge [$m^3 \times 10^6$]	Volume of Sand [$m^3 \times 10^6$]	Sand into Dykes [$m^3 \times 10^6$]	Sand into Beach [$m^3 \times 10^6$]	Sludge Rehandle Volume [$m^3 \times 10^6$]
1	61.991	23.351	10.783	27.857	10.100	17.757	0
2	174.647	65.787	30.379	78.481	7.800	70.681	0
3	178.428	67.211	31.037	80.180	7.200	72.980	0
4	183.837	69.248	31.978	82.611	6.400	76.211	0
5	187.028	70.450	32.533	84.045	4.700	79.345	0
6	183.921	69.280	31.992	82.649	3.700	78.949	0
7	186.785	70.359	32.490	83.936	2.700	81.236	0
8	186.087	70.096	32.369	83.622	2.300	81.322	0
9	181.540	68.383	31.578	81.579	1.600	79.979	0
10	175.078	65.949	30.454	78.675	1.170	77.505	0
11	168.507	63.474	29.311	75.722	18.023	57.699	0
12	170.834	64.350	29.716	76.768	24.132	52.636	0
13	166.470	62.706	28.957	74.807	0	74.807	0
14	175.043	65.936	30.448	78.659	9.294	69.365	0
15	177.284	66.780	30.838	79.666	20.988	58.678	33.829
16	189.933	71.545	33.038	85.350	11.172	74.178	61.658
17	177.285	66.780	30.838	79.667	0	79.667	61.658
18	180.844	68.121	31.457	81.266	0	81.266	61.658
19	196.154	73.888	34.120	88.146	0	88.146	61.658
20	198.506	74.774	34.529	89.203	0	89.203	61.658
21	199.855	75.282	34.764	89.809	0	89.809	61.658
22	196.648	74.074	34.206	88.368	0	88.368	87.686
23	195.621	73.687	34.027	87.907	0	87.907	109.511
24	184.714	69.579	32.120	83.005	0	83.005	83.005
25	126.821	47.771	22.060	56.990	0	56.990	57.684
26	58.175	21.914	10.119	26.142	0	26.142	25.838
	4,462.036	1,680.775	776.151	2,005.110	131.279	1,873.831	767.501

Ore Body No.1, 240,000 B.P.C.D.- 6 Bucket Wheel Excavators
SCHEDULE FOR MINIMUM LEVEL OF RECLAMATION
TABLE No. 9.2.1-2

Soil Composition:
0.20 m Muskeg
0.20 m Overburden
0.20 m Sand (where applicable)

Soil Manufacture:
Layer of muskeg and overburden (where required) are spread onto area to be reclaimed and plowed 0.6 m deep.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by trucks) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by trucks) [km]
1						
2						
3						
4						
5						
6						
7						
8	2,125	1,275	425	4.40	850	0
9	4,858	2,915	972	5.55	1,944	0
10	1,619	972	324	6.01	648	0
11	1,619	972	324	7.73	648	0
12	3,259	1,955	652	7.69	976	1.40
13	2,457	1,474	491	7.34	655	1.20
14	2,457	1,474	491	8.51	655	10.20
15	2,014	1,209	403	5.35	478	6.27
16	1,639	984	328	2.83	443	0.90
17	1,718	1,031	344	1.37	541	0.85
18	2,618	1,571	524	1.87	901	1.60
19	2,640	1,584	528	3.08	728	2.70
20	3,852	2,311	770	4.51	974	5.05
21	3,633	2,180	727	6.78	886	7.75
22	4,028	2,417	806	7.43	965	5.05
23	4,028	2,417	806	6.88	965	5.08
24	6,169	3,701	1,234	5.55	1,553	4.53
25	6,169	3,701	1,234	4.97	1,553	4.24
26	6,461	3,877	1,292	5.51	1,452	4.73
27	6,461	3,877	1,292	5.04	1,452	4.57
28	5,664	3,398	1,133	5.00	1,133	4.39
29	6,111	3,666	1,222	3.95	1,222	4.35
30	6,111	3,666	1,222	3.93	1,222	3.58
	87,710	52,626	17,542		22,844	

tailings pond begins in Year 22 and continues until Year 24. Sludge transfer from the second in-pit pond will last from Year 23 to 26. During these periods, all three ponds are filled with sand to produce dry sanded-in surfaces.

Material Distribution Plan
(Techman Drawing No. D22916-79-00)

This plan shows the types of materials that must be surfaced with prepared soil. Outside waste dumps, in-pit overburden reject backfill, and sanded-in tailings ponds form dry, reclaimable surfaces. However, the sludge pond is currently considered unreclaimable. The end-pit is reclaimed to a fresh water lake. Exposed pit walls require the application of prepared soil. Muskeg dumps do not require surfacing with prepared soil. It should be noted that, during the course of activities, some of the muskeg stored in dumps is used for the manufacture of prepared soil. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan
(Techman Drawing No. D22980-80-00)

The surfaces to be reclaimed as well as the time periods during which reclamation occurred are shown. Only the sludge pond remains wet and unreclaimable. Plant species are selected according to the reclamation objectives for the Minimum Level as described in Chapter 4.0.

9.2.2 ENHANCED (DRY) LEVEL OF RECLAMATION

The major aspects of the development of the 240,000 BPCD mine at the Enhanced Level are depicted by seven drawings, accompanied by tailings disposal and reclamation schedules (See Table 9.2.2-1 and 9.2.2-2). The mining schedule, which is common to the plans for both levels of reclamation, has been illustrated previously in Table 9.2-1. A drawing-by-drawing discussion follows:

General Layout
(Techman Drawing No. D22910-81-00)

The mine boundaries and the advance of the mining faces are identical to those for the plans at the Minimum Level. A difference exists with respect to the disposal of tailings, which in this case are dry and transported by conveyors. A small outside dump is required for each half of the mine.

Mining and Tailings Disposal - Year 1
(Techman Drawing No. D22918-82-00)

By the end of Year 1, the mining faces are fully developed. Three BWE's and three spreaders are in operation in each half of the mine.

Mining and Tailings Disposal - Year 7
(Techman Drawing No. D22918-83-00)

The outside dumps were completed in Year 3. The manufacture of prepared soil commenced in Year 4. The muskeg dewatering plant and blending yard are situated south of the neck in the southern mine. Muskeg is pumped as slurry to the dewatering plant. Overburden is obtained by means of a 2,600 mm conveyor accepting suitable overburden from the southern mine distribution point. A stacker blends the overburden and muskeg into windrows. This material is loaded by a small bucket wheel reclaimer onto a belt conveyor leading to the field stockpile of prepared soil. The outside dump for the southern half of the mine was reclaimed in Year 4 and 5, and the outside dump for the northern half in Year 6 and 7.

Ore Body No. 1, 240,000 B.P.C.D.- 6 B.W.E.
TAILINGS SCHEDULE
FOR ENHANCED LEVEL OF RECLAMATION

TABLE 9.2.2-1

Outline of Tailings Disposal Scheme:

- Dry Tailings Conveyed with Overburden and Center Reject for First 2.5 to 3.25 Years to out-of-Pit Waste Dump, then into Mined-out Pit.

YEAR	Volume of Dry Tailings Produced [$m^3 \times 10^6$]	Dry Tailings Conveying Distance * [m]
1	32.632	40,400
2	77.300	42,850
3	83.606	37,850
4	86.058	41,400
5	86.501	43,850
6	86.925	45,850
7	88.818	50,100
8	88.064	55,150
9	82.109	61,300
10	80.794	62,500
11	80.553	63,350
12	81.911	63,100
13	79.475	60,000
14	83.313	54,100
15	84.513	53,950
16	87.565	49,400
17	83.330	40,250
18	83.999	38,700
19	90.213	39,400
20	90.945	42,550
21	91.020	45,400
22	89.206	50,500
23	86.907	52,350
24	76.688	54,650
25	58.916	56,000
26	27.026	56,750
	2,068.387	

*NOTE:

These are Total Lengths of Four Conveyor Systems from Plant to Distribution Points and Six Conveyor Systems from Distribution Points to Six Spreaders which also Handle Overburden and Centre Reject.

Ore Body No1, 240,000 B.P.C.D.- 6 Bucket Wheel Excavators

SCHEDULE FOR ENHANCED LEVEL OF RECLAMATION

TABLE No. 9.2.2-2

Soil Composition:
0.33m Muskeg
0.66m Overburden

Soil Manufacture:
Stacker deposits layers of Muskeg and overburden into piles. Components are mixed by bucket wheel reclaimer.

YEAR	Area of Reclamation [$m^2 \times 10^3$]	Volume of Prepared Soil [$m^3 \times 10^3$]	Prepared Soil Transport (by conveyors) [km]	Prepared Soil Transport (by trucks) [km]	Volume of Muskeg [$m^3 \times 10^3$]	Muskeg Transport (by pipeline) [km]	Volume of Overburden [$m^3 \times 10^3$]	Overburden Transport (by conveyors) [km]
1	0	0	0	0	0	0	0	0
2	0	0	0	0	0	0	0	0
3	0	0	0	0	0	0	0	0
4	1,310	1,310	1.8	1.5	437	14.6	873	6.7
5	1,310	1,310	1.8	1.5	437	14.6	873	7.4
6	1,480	1,480	6.2	1.8	493	14.6	987	7.9
7	1,480	1,480	6.2	1.8	493	14.6	987	8.1
8	1,800	1,800	4.9	1.0	600	14.6	1,200	6.9
9	1,800	1,800	4.9	2.0	600	14.6	1,200	6.9
10	1,800	1,800	4.9	3.0	600	14.6	1,200	6.8
11	1,800	1,800	4.3	1.5	600	14.6	1,200	4.6
12	1,800	1,800	4.3	2.0	600	14.6	1,200	5.6
13	1,800	1,800	4.3	3.0	600	14.6	1,200	5.3
14	1,800	1,800	4.3	3.7	600	14.6	1,200	3.7
15	1,800	1,800	4.3	4.2	600	14.6	1,200	3.6
16	1,800	1,800	1.5	1.0	600	14.6	1,200	3.7
17	1,800	1,800	1.5	1.3	600	14.6	1,200	4.0
18	1,800	1,800	7.5	1.7	600	14.6	1,200	4.3
19	1,800	1,800	7.5	1.5	600	14.6	1,200	4.5
20	1,800	1,800	7.5	1.0	600	14.6	1,200	4.7
21	1,800	1,800	7.5	0.5	600	14.6	1,200	4.9
22	1,800	1,800	4.5	1.7	600	14.6	1,200	5.2
23	1,800	1,800	4.5	0.6	600	14.6	1,200	5.4
24	1,800	1,800	4.5	1.0	600	14.6	1,200	5.5
25	1,800	1,800	4.5	1.5	600	14.6	1,200	5.5
26	1,800	1,800	4.5	2.0	600	14.6	1,200	1.3
27	1,790	1,790	4.5	3.0	597	14.6	1,193	1.3
28	1,770	1,770	4.2	2.5	590	14.6	1,180	1.3
29	1,770	1,770	4.2	2.0	590	14.6	1,180	1.3
30	1,770	1,770	4.2	1.2	590	14.6	1,180	1.3
	46,880	46,880			15,627		31,253	

Mining and Tailings Disposal - Year 14
(Techman Drawing No. E22918-84-00)

The in-pit backfilling of the mine with dry tailings, overburden, and reject follows closely behind the mining face. The major part of the muskeg removal has been completed. The location of the field stockpile for prepared soil has been changed to a position just west of the mine and north of the southern-most outside dump. Reclamation of the western portion of the southern half of the mine and the eastern portion of the northern mine began in Year 8.

Mining and Tailings Disposal - Year 22
(Techman Drawing No. E22918-85-00)

The conveyor distribution point for the southern half of the mine has been relocated. The prepared soil conveyor now passes to the west of this point enroute to the stockpile site. The current length of the conveyor is 4.5 km. From Year 18 to 21, the conveyor length was 7.5 km and so extended into the northern half of the mine. Compared to the plans at the Minimum Level of Reclamation, the surface area utilized for muskeg dumps is greater. At the Minimum Level a portion of the muskeg is used in prepared soil manufacture, much of it by direct transfer from source to final destinations on the reclamation areas, whereas at the Enhanced Level, the muskeg is obtained from a hydraulic muskeg mine assumed to be located outside of the mine boundary.

Material Distribution Plan
(Techman Drawing No. E22916-86-00)

This drawing shows that the main material to be surfaced with prepared soil is a mixture of dry tailings and overburden. During the course of mining, toxic materials are buried. Compared to the Year 22 position, the major difference in Year 26 is that the backfill is extended eastward. A final C-shaped end-pit lake remains.

Reclamation activities are to continue for about another five years, and for this purpose, suitable overburden is selectively placed during the backfilling operation for retrieval. This is done by re-mining part of the backfill shown to the west of the blending yard. The plant site is to be resurfaced with a layer of prepared soil. There are no unreclaimable surface areas remaining in this plan. Refer to Section 10.6, Table 10.6-1 for a mine-by-mine comparison of surfaces to be reclaimed.

Reclamation Plan

(Techman Drawing No. E22980-87-00)

The surface area to be reclaimed as well as the time period during which reclamation occurred are shown. The end-pit lake submerges the unmined peninsula jutting into the northern half of the mine. Table 9.2.2-2 is a detailed schedule of reclamation activities on a year-by-year basis. Plant species are selected according to the reclamation objectives for the Enhanced Level of Reclamation described in Chapter 4.0.

9.3 COST SUMMARIES FOR 240,000 BPCD MINE PLANS

Cost summaries are provided for each of the mine plans detailed in this chapter. Rather than following immediately behind the description of the mine plan, the summary tables are grouped at this point in the report for ease of reading and comparison.

Two tables (Tables 9.3-1 and 9.3-2) summarize the quantities, unit costs and $/bbl costs of both capital and operating items. Further details for each cost summary are provided on an annual basis in Volume III. A comparison between the mines costed in this and other chapters of this report follows in Section 10.7.

COST ANALYSIS ($/bbl.), ORE BODY NO. 1 (240,000 BPCD), B.W.E. SCHEME, MINIMUM LEVEL

TABLE 9.3-1

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0455
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0228	0.0228	-	-	-	0.0149	0.0149	0.0377
1.2 Buildings	10,885.0 persons	-	1,000.00 $/person	0.0051	-	0.0051	392.0 persons	15,000.00 $/person	0.0028	-	0.0028	0.0078
COST CENTRE 2: Removal of Organic Materials & Soils												0.0517
2.1 Clearing	8,772.0 hectares	-	1,266.82 $/ha	0.0052	-	0.0052	8,772.0 hectares	196.85 $/ha	0.0008	-	0.0008	0.0060
2.2 Muskeg Dewatering	1,486.7 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0013	-	0.0013	1,486.7 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0002	-	0.0002	0.0015
2.3 Muskeg Loading	29,306.0 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0117	-	0.0117	29,306.0 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0028	-	0.0028	0.0145
2.4 Muskeg Hauling (Including Road Maintenance)	29,306.0 1,000 bank m^3	4.93 km	250.80 $/1,000 bank m^3xkm	0.0170	-	0.0170	144,608.1 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0033	-	0.0033	0.0203
2.5 Muskeg Placement	29,306.0 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0066	-	0.0066	29,306.0 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0005	-	0.0005	0.0070
2.6 Muskeg Road Construction	126.5 km	-	33,825.11 $/km	0.0020	-	0.0020	126.5 km	6,915.90 $/km	0.0004	-	0.0004	0.0024
COST CENTRE 3: Overburden, Reject, Oil Sands Handling												1.1145
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	3,250,136.0 1,000 bank m^3	-	137.21 $/1,000 bank m^3	0.2086	-	0.2086	-	-	-	0.1070	0.1070	0.3156
3.4 Transport (All Conveyors)	3,250,136.0 1,000 bank m^3	57,333.0 m	0.004 $/1,000 bank m^3xkm	0.3887	-	0.3887	7,3680.0 m	4,215.78 $/m	0.1453	-	0.1453	0.5340
3.5 Placement (Spreaders)	1,283,989.0 1,000 bank m^3	-	91.33 $/1,000 bank m^3	0.0548	-	0.0548	-	-	-	0.0289	0.0289	0.0837
3.6 Miscellaneous Equipment	-	-	-	-	0.1461	0.1461	-	-	-	0.0351	0.0351	0.1812
COST CENTRE 4: Tailings Disposal											-	0.4034
4.1 Area Drainage	38.2 1,000 bank m^3	-	1,239.20 $/1,000 bank m^3	0.00002	-	0.00002	38.2 1,000 bank m^3	300.95 $/1,000 bank m^3	0.000005	-	0.000005	0.00003
4.2 Clearing	4,825.0 hectares	-	1,266.82 $/ha	0.0029	-	0.0029	4,825.0 hectares	196.85 $/ha	0.0004	-	0.0004	0.0033
4.3 Construction of Starter Dams & Overburden Dams	22,592.0 1,000 bank m^3	-	1,659.23 $/1,000 bank m^3	0.0175	-	0.0175	8,222.0 1,000 bank m^3	262.17 $/1,000 bank m^3	0.0010	-	0.0010	0.0185
4.4 Piping of Tailings or Conveying of Dry Tailings	4,462,036.0 1,000 m^3	-	70.99 $/1,000 m^3	0.1482	-	0.1482	-	-	-	0.0235	0.0235	0.1716
4.5 Tailings Sand Placement into Dyke	131,279.0 1,000 m^3	-	123.56 $/1,000 m^3	0.0076	-	0.0076	-	-	-	0.0110	0.0110	0.0185
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	0.0595	-	0.0595	-	-	-	-	-	0.0595
4.7 Recycling of Tailings Water	-	-	-	-	0.0325	0.0325	-	-	-	0.0067	0.0067	0.0392
4.8 Rehandling of Tailings Sludge	767,501.0 1,000 m^3	-	45.68 $/1,000 m^3	0.0164	-	0.0164	-	-	-	0.0074	0.0074	0.0238

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 1 (240,000 BPCD), B.W.E. SCHEME, MINIMUM LEVEL (Continued)

TABLE 9.3-1 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price ?	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	21.7 km	25,000.00 $/km	0.0003	0.0003	0.0005	0.0005
4.11 Oversize Reject Disposal	112,070.0 1,000 loose m^3	-	1,095.78 $/1,000 loose m^3	0.0574	-	0.0574	112,070.0 1,000 loose m^3	204.67 $/1,000 loose m^3	0.0107	-	0.0107	0.0682
4.12 Oversize Reject Disposal Road Construction	9.4 km	-	33,825.11 $/km	0.0001	-	0.0001	9.4 km	6,915.90 $/km	0.00003	-	0.00003	0.0002
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.0556
5.1 Muskeg Rehandle Loading	13,830.8 1,000 bank m^3	-	615.19 $/1,000 bank m^3	0.0040	-	0.0040	13,830.8 1,000 bank m^3	149.00 $/1,000 bank m^3	0.0010	-	0.0010	0.0049
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	13,830.8 1,000 bank m^3	4.99 km	224.53 $/1,000 bank m^3xkm	0.0072	-	0.0072	69,029.4 1,000 bank m^3xkm	42.43 $/1,000 bank m^3xkm	0.0014	-	0.0014	0.0086
5.3 Muskeg Rehandle Placement	13,830.8 1,000 bank m^3	-	163.24 $/1,000 bank m^3	0.0011	-	0.0011	13,830.8 1,000 bank m^3	26.62 $/1,000 bank m^3	0.0002	-	0.0002	0.0012
5.4 Muskeg Rehandle Road Construction	332.9 km	-	33,825.11 $/km	0.0053	-	0.0053	332.9 km	6,915.90 $/km	0.0011	-	0.0011	0.0063
5.5 Overburden Rehandle Loading	12,562.1 1,000 bank m^3	-	541.57 $/1,000 bank m^3	0.0032	-	0.0032	12,562.1 1,000 bank m^3	131.53 $1,000 bank m^3	0.0008	-	0.0008	0.0040
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	12,562.1 1,000 bank m^3	4.47 km	178.56 $/1,000 bank m^3xkm	0.0047	-	0.0047	56,111.8 1,000 bank m^3xkm	34.00 $/1,000 bank m^3xkm	0.0009	-	0.0009	0.0056
5.7 Overburden Rehandle Placement	12,562.1 1,000 bank m^3	-	140.93 $/1,000 bank m^3	0.0008	-	0.0008	12,562.1 1,000 bank m^3	22.92 $1,000 bank m^3	0.0001	-	0.0001	0.0010
5.8 Overburden Rehandle Road Construction	12.6 km	-	33,825.11 $/km	0.0002	-	0.0002	12.6 km	6,915.90 $/km	0.00004	-	0.00004	0.0002
5.9 Muskeg Mining, Slurry Transport and Dewatering	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.10 Prepared Soil Manufacture	52,625.9 1,000 bank m^3	-	27.85 $/1,000 bank m^3	0.0007	-	0.0007	52,625.9 1,000 bank m^3	4.36 $/1,000 bank m^3	0.0001	-	0.0001	0.0008
5.11 Prepared Soil Loading, F.E.L. & Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.13 Prepared Soil Placement, Trucks	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.14 Prepared Soil Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.15 Seed Bed Preparation, Maintenance	8,771.0 hectares	-	768.05 $/ha	0.0032	0.0198	0.0230	- hectares	- $/ha	-	-	-	0.0230
COST CENTRE 6: Supervision, Technical Services												0.1499
6.1 Equipment Maintenance (Staff only)	4,115.0 persons	-	29,195.00 $/person	0.0562	-	0.0562	-	-	-	-	-	0.0562
6.2 Planning (Staff only)	3,388.0 persons	-	28,025.00 $/person	0.0444	-	0.0444	-	-	-	-	-	0.0444
6.3 Mining (Staff only)	3,622.0 persons	-	29,740.00 $/person	0.0493	-	0.0493	-	-	-	-	-	0.0493
TOTAL COSTS						1.4119					0.4088	1.8207

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 1 (240,000 BPCD), B.W.E. SCHEME, ENHANCED LEVEL

TABLE 9.3-2

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
COST CENTRE 1: Civil Construction-Type Activities												0.0580
1.1 Mine-Power Distribution & Control	-	-	-	-	0.0280	0.0280	-	-	-	0.0219	0.0219	0.0500
1.2 Buildings	11,116.0 persons	-	1,000.00 $/person	0.0052	-	0.0052	400.0 persons	15,000.00 $/person	0.0028	-	0.0028	0.0080
COST CENTRE 2: Removal of Organic Materials & Soils												0.0419
2.1 Clearing	7,536.1 hectares	-	1,266.82 $/ha	0.0045	-	0.0045	7,536.0 hectares	196.85 $/ha	0.0007	-	0.0007	0.0052
2.2 Muskeg Dewatering	1,486.7 1,000 bank m^3	-	1,850.00 $/1,000 bank m^3	0.0013	-	0.0013	1,486.7 1,000 bank m^3	300.00 $/1,000 bank m^3	0.0002	-	0.0002	0.0015
2.3 Muskeg Loading	29,306.0 1,000 bank m^3	-	851.40 $/1,000 bank m^3	0.0117	-	0.0117	29,306.0 1,000 bank m^3	207.00 $/1,000 bank m^3	0.0028	-	0.0028	0.0145
2.4 Muskeg Hauling (Including Road Maintenance)	29,306.0 1,000 bank m^3	3.01 km	250.80 $/1,000 bank m^3xkm	0.0104	-	0.0104	88,324.1 1,000 bank m^3xkm	48.70 $/1,000 bank m^3xkm	0.0020	-	0.0020	0.0124
2.5 Muskeg Placement	29,306.0 1,000 bank m^3	-	479.40 $/1,000 bank m^3	0.0066	-	0.0066	29,306.0 1,000 bank m^3	34.60 $/1,000 bank m^3	0.0005	-	0.0005	0.0070
2.6 Muskeg Road Construction	67.3 km	-	33,825.11 $/km	0.0011	-	0.0011	67.3 km	6,915.90 $/km	0.0002	-	0.0002	0.0013
COST CENTRE 3: Overburden, Reject, Oil Sands Handling, and Tailings Disposal												1.4402
3.1 Overburden B.W.E.	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.2 Oil Sands Draglines & Hoppers	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	-	-	-	-	-	-
3.3 B.W.E. (Overburden & Oil Sands)	3,250,136.0 1,000 bank m^3	-	137.21 $/1,000 bank m^3	0.2086	-	0.2086	-	-	-	0.1070	0.1070	0.3156
3.4 Transport (All Conveyors)	5,318,523.0 1,000 bank m^3	92,126.8 m	0.002 $/1,000 bank m^3xkm	0.5443	-	0.5443	115,250.0 m	4,547.13 $/m	0.2451	-	0.2451	0.7894
3.5 Placement (Spreaders)	3,352,376.0 1,000 bank m^3	-	51.89 $/1,000 bank m^3	0.0814	-	0.0814	-	-	-	0.0464	0.0464	0.1277
3.6 Miscellaneous Equipment	-	-	-	-	0.1679	0.1679	-	-	-	0.0395	0.0395	0.2074
COST CENTRE 4: Tailings Disposal - Included in Cost Centre 3											-	
4.1 Area Drainage	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
4.2 Clearing	- hectares	-	- $/ha	-	-	-	- hectares	- $/ha	-	-	-	-
4.3 Construction of Starter Dams & Overburden Dams	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
4.4 Piping of Tailings or Conveying of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.5 Tailings Sand Placement into Dyke	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.7 Recycling of Tailings Water	-	-	-	-	-	-	-	-	-	-	-	-
4.8 Rehandling of Tailings Sludge	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-

NOTE: Refer to Chapter 6 for Cost Centre description.

COST ANALYSIS ($/bbl.), ORE BODY NO. 1 (240,000 BPCD), B.W.E. SCHEME, ENHANCED LEVEL (Continued)

TABLE 9.3-2 (Continued)

COST ITEM	Quantity 1	Quantity 2	Unit Price 1	Operating Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Operating Cost ($/bbl.)	Quantity 3	Unit Price 2	Capital Cost ($/bbl.)	Additional Cost ($/bbl.)	Total Capital Cost ($/bbl.)	Subtotal ($/bbl.)
4.9 Sludge Treatment	- 1,000 m^3	-	- $/1,000 m^3	-	-	-	-	-	-	-	-	-
4.10 Power Distribution	-	-	-	-	-	-	- km	- $/km	-	-	-	-
4.11 Oversize Reject Disposal	-	-	-	-	-	-	-	-	-	-	-	-
4.12 Oversize Reject Disposal Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
COST CENTRE 5: Establishment of Ultimate Land Use Resources												0.1194
5.1 Muskeg Rehandle Loading	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.2 Muskeg Rehandle Hauling (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	- -	-	-	-
5.3 Muskeg Rehandle Placement	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $/1,000 bank m^3	-	-	-	-
5.4 Muskeg Rehandle Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.5 Overburden Rehandle Loading	6,833.0 1,000 bank m^3	-	373.56 $/1,000 bank m^3	0.0012	-	0.0012	6,833.0 1,000 bank m^3	43.86 $1,000 bank m^3	0.0001	-	0.0001	0.0013
5.6 Overburden Rehandle Hauling (incl. Road Maintenance)	- 1,000 bank m^3	- km	- $/1,000 bank m^3xkm	-	-	-	- 1,000 bank m^3xkm	- $/1,000 bank m^3xkm	-	-	-	-
5.7 Overburden Rehandle Placement	- 1,000 bank m^3	-	- $/1,000 bank m^3	-	-	-	- 1,000 bank m^3	- $1,000 bank m^3	-	-	-	-
5.8 Overburden Rehandle Road Construction	- km	-	- $/km	-	-	-	- km	- $/km	-	-	-	-
5.9 Muskeg Mining, Slurry Transport and Dewatering	115,627.0 1,000 bank m^3	-	2,329.44 $/1,000 bank m^3	0.0170	-	0.0170	- 1,000 bank m^3	- $/1,000 bank m^3	-	0.0083	0.0083	0.0253
5.10 Prepared Soil Manufacture	46,880.0 1,000 bank m^3	-	1,090.34 $/1,000 bank m^3	0.0239	-	0.0239	- 1,000 bank m^3	- $/1,000 bank m^3	-	0.0202	0.0202	0.0441
5.11 Prepared Soil Loading, F.E.L. & Trucks	46,880.0 1,000 bank m^3	-	430.12 $/1,000 bank m^3	0.0094	-	0.0094	46,880.0 1,000 bank m^3	115.03 $/1,000 bank m^3	0.0025	-	0.0025	0.0120
5.12 Prepared Soil Transport, Trucks (incl. Road Maintenance)	46,880.0 1,000 bank m^3	1.84 km	212.40 $/1,000 bank m^3xkm	0.0086	-	0.0086	83,577.0 1,000 bank m^3xkm	57.98 $/1,000 bank m^3xkm	0.0023	-	0.0023	0.0108
5.13 Prepared Soil Placement, Trucks	46,880.0 1,000 bank m^3	-	150.57 $/1,000 bank m^3	0.0033	-	0.0033	46,880.0 1,000 bank m^3	24.52 $/1,000 bank m^3	0.0005	-	0.0005	0.0038
5.14 Prepared Soil Road Construction	281.3 km	-	33,825.11 $/km	0.0044	-	0.0044	281.3 km	6,915.90 $/km	0.0009	-	0.0009	0.0054
5.15 Seed Bed Preparation, Maintenance	4,688.0 hectares	-	768.05 $/ha	0.0017	0.0150	0.0167	- hectares	- $/ha	-	-	-	0.0167
COST CENTRE 6: Supervision, Technical Services												0.1531
6.1 Equipment Maintenance (Staff only)	4,115.0 persons	-	29,195.00 $/person	0.0562	-	0.0562	-	-	-	-	-	0.0562
6.2 Planning (Staff only)	3,388.0 persons	-	28,025.00 $/person	0.0444	-	0.0444	-	-	-	-	-	0.0444
6.3 Mining (Staff only)	3,779.0 persons	-	29,730.00 $/person	0.0525	-	0.0525	-	-	-	-	-	0.0525
TOTAL COSTS						1.3085					0.5041	1.8126

NOTE: Refer to Chapter 6 for Cost Centre description.

10.0 MAJOR FACTORS IN THE DEVELOPMENT AND RECLAMATION OF OIL SANDS MINES

10.1 ORE BODY CONFIGURATION

Next to tailings disposal techniques employed, the shape of the ore body to be mined is likely the most significant factor determining the areal extent to which reclamation activities can be carried out. Three major general shapes are of concern: the uniform and areally large ore body, the uniform and areally small ore body, and the longitudinally-extended ore body.

The uniform ore body may range in size from very small to around 150,000 BPCD. At sizes larger than 150,000 BPCD it is very doubtful if the mine can be operated as a single mine. Subdivision into two operating units is likely unavoidable since technical operating limits in equipment application become the determining factor. The ore bodies are too shallow to permit more than 3 benches (4 in deeper mines), and consequently there may be severe congestion on benches due to the numbers of excavators and conveyors needed if more than 150,000 BPCD of production are demanded from a single pit. For example, Ore body No. 1 exceeded the maximum level of production attainable from one mining face, and was therefore subdivided into two mines. Nevertheless, such mines can be operated jointly, with many advantages for overburden and tailings disposal. The sizes of neither the draglines nor BWE's suggested in this study are likely to be exceeded in the near future, namely 80 m^3 (dragline bucket) and 100,000 m^3/day (BWE production) respectively. As operating experience is gained, draglines and BWE's with higher capacities may be shown to be practical.

In the small size range (mines of around 60,000 BPCD), it appears that draglines and BWE's can be operated in the same mine layout with equal ease. As the ore body size increases beyond 60,000 BPCD, the capacity of reasonably-dimensioned draglines becomes marginal, and the duplication of excavators, i.e. 2 draglines per bench must be considered. A uniformly shaped ore body, preferably one which is rectangular in shape, can still be planned to be rather efficient, even with duplication of draglines on benches.

Duplication of excavators on benches (especially when a series of contiguous ore bodies is to be mined) must be carefully weighed against other possible development alternatives, however. Considering the efficiency of mining alone, better solutions exist.

Very small ore bodies will likely remain equally attractive for either draglines or BWE. The overburden will be excavated by either a bucket wheel excavator, shovel, or front end loaders. The choice for transport is determined by the efficiency with which a conveyor system can be operated, as compared to truck haulage. Very small ore bodies will lend themselves best to shovel, front end loader, and truck operating techniques. A cluster of small ore bodies would be particularly attractive for this mining method.

A single small ore body, while not presenting a drawback to mining, is somewhat problematic with respect to tailings disposal. In some small mines it will be possible to conduct some in-pit tailings disposal but in others this may be totally impractical. This implies that the mined-out pit remains empty, while all tailings are disposed out-of-pit. A regional plan integrating the development of several small pits is a very attractive solution for the disposal of wet tailings from these mines. The mined-out pit from one operation might be used permanently for the tailings disposal or temporarily for the make-up water reservoir of another mine.

Longitudinally-extended ore bodies are ideal for wet tailings disposal methods. The efficient sequencing of ponds is the most prominent concern. Longitudinally-extended ore bodies that have an excessively high overall ratio of length to width may exhibit the tailings disposal scheduling problems inherent in a uniform ore body.

The relative advantages of shape are minimized when dry tailings rather than wet tailings are produced. The major concerns are slope stability of the backfill and the stability of the pit floor. Backfilled slopes must remain stable for considerable periods of time. Consequently, the backfilling of the large uniform pits may still be more troublesome than the backfilling of medium-sized pits. At the other extreme, it may become more costly to backfill small individual pits than to backfill the medium-sized pits. Considerable expense may be incurred with small pits for backfilling, especially as the distance between such pits increases.

The ore body configuration as defined by geologic criteria is seldom suitable for mining. Adjustments to the boundaries must be made by striking a balance between the higher grade ore that must be left and the lower grade ore that must be mined. When making such boundary adjustments, consideration must be given to the tailings disposal requirements as well as mining requirements. Design tools such as simulation programs must be available to examine a sufficient number of options before a final decision is reached.

A serious question regarding the future time-value of oil sands resources remains. Shapes that are ideal for tailings disposal, and which thus offer greater reclamation potential, may cause a greater volume of marginal oil sands to be rendered unmineable. Ore at the pit boundary is sterilized once the pit is backfilled with wet tailings products and, to some extent, even when backfilled with dry tailings. The construction of conventional tailings ponds in such a way that an open corridor is left between the tailings ponds and the pit wall is likely impractical for all but the most ideally sized and regularly shaped ore bodies.

It is costly to completely surround wet tailings with containment structures within a mined-out pit. Alternatively, narrow, strip-type, sanded-in ponds could be used advantageously when the tailings sand must be kept from contacting the pit wall. In all cases, it also must be shown that the overall mass balance within the mine permits the use of the selected type of containment structures or facilities. With deeper ore bodies, slope stability is most certainly a factor in determining whether such a scheme should be attempted or not. When waste overburden and reject are placed on the pit floor geotechnical problems related to dyke stability may be further compounded.

It appears that much more can be gained in the line of oil sands resource conservation by following a regional development plan, in which marginal ore may be conserved through the optimization of ore bodies and the utilization of smaller outside dumps and out-of-pit tailings ponds. As well, much more effective reclamation is possible. The latter tailings disposal scheme allows for permanent reclamation, whereas the former results in considerable temporary reclamation while the mines await further development some time in the future.

10.2 CHARACTERISTICS OF MATERIALS HANDLING EQUIPMENT

The major excavating equipment selected for a particular operation will not have a significant influence on the success of reclamation. Draglines, BWE's, BWR, spreaders, and conveyors will be the major materials handling equipment in the medium and large mines. Small mines may make more use of combinations of shovels, loaders, and trucks. For reclamation the major concern is that a means be provided whereby sufficient quantities of overburden and muskeg suitable for prepared soil manufacture are isolated from the other material being handled by the mining system.

The mines as developed in this report require relatively high initial capital investments. Mines smaller than about 40,000 BPCD may be able to take advantage of low initial capital investments required for a fleet of mobile equipment. The small size of ore bodies less than 40,000 BPCD may preclude the use of conveyor transport to some extent for certain phases of the operation. This does not necessarily imply poor economics for small ore bodies, but rather that an innovative materials handling technology other than that currently used by oil sands mine operators and discussed in this study may have to be applied. It is strongly suspected that even for such small mines the major concern will again be to isolate suitable reclamation material at strategic locations for later use in reclamation activities.

The advantages and disadvantages of draglines or bucket wheels, single and multiple benches, and parallel or slewing conveyor layouts have been demonstrated in the twelve mine plans developed in this study. A clear-cut decision with respect to the suitability of a prime excavator cannot be made based on a comparison of the selective capabilities of the dragline and BWE. In Chapter 5.1, it was demonstrated that finer selectivity can result in decreased net oil yield. On the other hand, less selectivity results in more low grade oil sands being processed as plant feed resulting in a higher net oil yield. Unfortunately, this is at the expense of greater capital and higher operating costs for the extraction plant, and most certainly involves the generation of larger quantities of tailings and sludge. However, the choice of excavator is clearly indicated when the total capital and operating costs of the primary mining system are considered.

At the 60,000 BPCD size, the total capital and operating costs of the primary mining system for both a dragline and a bucket wheel mine are nearly identical. The costs of two draglines and one BWE balance the costs of three BWE's, provided a similar conveyor layout is employed. As the mine size is increased to 120,000 BPCD, a typical primary mining system in a dragline plan requires four draglines and one BWE, while a bucket wheel mine plan requires three BWE's and considerably less installed length of conveyors than the dragline plan requires. With its distribution of equipment, the dragline system is at a great disadvantage when its capital and operating costs are compared to those of the bucket wheel system. Economy of scale in oil sands is realized with the bucket wheel excavator but not with the dragline.

The application of the dragline should be restricted to those ore bodies where the geology will allow the efficient application of this type of excavator. The overall cost effectiveness of a dragline drops dramatically when pit geometry and operating conditions vary from the optimum for which the dragline has been designed. The possibility of hybrid systems, BWE's on the upper and middle bench, and dragline on the lowest bench appear attractive. The flexibility of the bucket wheel section could be used to created an ideal working pit geometry for the dragline.

Direct transfer of suitable materials from the mining face to the reclamation site is possible only for a very small percentage of the time. Intermediate stockpiling of suitable overburden materials is the rule for all oil sands mines using wet tailings disposal methods. At the Minimum and Improved Levels of Reclamation, techniques were described whereby suitable overburden could be isolated. Periodically the spreaders separate suitable material for reloading and removal by front-end loaders and trucks into storage sites outside of the pit. Some time later this material is loaded onto off-highway trucks and transported directly to the reclamation site (at the Minimum Level) or to a blend pile (at the Improved Level). Only at the Enhanced Level is there an opportunity to tap directly into the materials handling system of the mine to obtain overburden, but a direct transfer to the reclamation site is also not possible. Overburden is transported by mining conveyors only to an intermediate blending yard. In the yard, material is handled by specialized equipment, eventually arriving at strategic field depots by means of conveyors installed only for prepared soil transport.

At the Minimum Level of Reclamation, the overburden and muskeg are placed into separate storage dumps and then reloaded and transported by front end loaders and trucks to the reclamation site. At the Improved Level, the muskeg and overburden are rehandled even further by forming blend piles. At the Enhanced Level, more rehandling occurs as both a blending yard and a stockpile area are utilized. At all levels, field placement is by means of a truck or scraper fleet. The common element at all three levels is that the entire prepared soil manufacturing scheme is independent of the mining scheme.

At all levels muskeg must be obtained as either a by-product of the muskeg stripping operation or from another source such as a hydraulic muskeg mine. As a by-product of muskeg removal operations, suitable muskeg is temporarily stored in muskeg storage depots until required. Occasionally, when a reclamation area is situated near the muskeg removal sites, some direct transport may be possible. At the Minimum and Improved Levels, storage of muskeg can only rarely be avoided, since there is usually a 10 to 12 year delay between the time the muskeg is first removed and the time it is used for major reclamation. At the Enhanced Level, the storage period will be the shortest, generally only one or two years in a blended state in the field.

Should there exist a series of neighbouring mines at various stages of development, interesting possibilities for the exchange of muskeg may occur. Schemes which obtain muskeg from hydraulic mines would gain most from a regionally integrated oil sands mining plan. It is difficult to operate a hydraulic muskeg mine within the boundary of an operating mine because of complications with respect to mine dewatering. The possibility of removing muskeg hydraulically from a mine in the initial development phase and piping the product to an operating mine elsewhere is of considerable interest.

The storage of muskeg within a suitably designed dump does not affect its usefulness for reclamation. However, the drying out of muskeg results in physical and chemical degradation of the product, diminishing its reclamation value. Drying out is likely to occur only at the Minimum Level of Reclamation, and then only to a minor extent when material is left on the reclamation site to thaw prior to blending in the field.

10.3 TAILINGS AND TAILINGS HANDLING TECHNIQUES

The characteristics of the extraction plant tailings determine the type of tailings disposal to be used and, consequently, the extent to which satisfactory reclamation is achieved. The design of optimal disposal plans for wet tailings is much more difficult than for dry tailings. However, this does not imply that the disposal of dry tailings is achieved without difficulty.

The characteristics of the tailings slurry determine the volumes of water, sludge, and sand that must be contained within ponds. The manner in which tailings products are disposed can be manipulated in each mine to achieve various end results. Generation of the maximum dry land surface area is the prime objective of such manipulation. The greatest amount of dry land surface area is formed from tailings slurry by creating sanded-in pond surfaces. A much smaller portion is also generated by the dyke slopes.

Currently it is possible to think only in terms of reclaiming dry land. The reclaiming of a sludge pond surface requires further research and field testing before an estimate of its feasibility can be rendered. It is not known whether the reclaiming of a thickened sludge pond will be more easily achieved than that of an unthickened sludge pond. Sludge ponds pose a long-term maintenance problem.

The maximum area of dry reclaimable land surfaces is obtained when an extraction plant producing dry tailings is employed. Currently operating and proposed extraction plants are based on the Clark's Hot Water process and produce a tailings slurry. The extraction plants assumed at the Enhanced Level use a high temperature, unhydrous process producing a dry mixture of sand and fines as tailings. The addition of water to dry tailings to produce a slurry for transport purposes would result in overall higher tailings disposal costs. Although pipeline transport is generally cheaper than conveying, there are many additional costs for wet tailings disposal. In addition, all the problems associated with wet tailings disposal would be needlessly created.

The disposal of extraction plant tailings appears to fall into three general categories: systems that rely entirely on natural settling for consolidation (conventional tailings), systems where consolidation of selected fractions is achieved by mechanical means (partially dewatered tailings), and systems which require no consolidation (dry tailings). The characteristics of these three categories of tailings have been considered in the development of materials handling schemes under the headings of Minimum, Improved and Enhanced Levels of Reclamation.

Subsection 5.3, Tailings Disposal Techniques, explained the possibilities for tailings disposal by illustrating the advantages and disadvantages in terms of Ore Body No. 2. These concepts, with the appropriate modifications, are adaptable to any mining scheme which may be considered.

Certain generalizations can be made with regard to tailings disposal:

a. The characteristics of the selected tailings disposal method determine the areal extent to which reclamation can occur. No other factor is as significant.

b. The mass balance for tailings must be known within close tolerances when a sequential disposal scheme is utilized. An error, for example, in sizing a pond or estimating dyke sand available may irreparably alter the entire mining and reclamation plan. Any major changes in mine layout and material distribution will require a complete reworking of the mine plan to determine if the planned tailings disposal plan is still feasible.

c. The extent to which reclaimable dry land surfaces are developed depends on the success that a given tailings disposal plan has in concentrating sludge into a containment structure with minimum surface area. This will involve rehandling of tailings sludge as well as some water.

d. There is very limited flexibility with respect to "landshaping" of tailings disposal facilities in the Minimum and Improved operation modes of tailings disposal. The pond area, the pond elevation, and the pond location are all severely restricted by the mining and tailings mass balance. In comparison, the dry scheme used in the Enhanced Level has fewer limitations. The fact that an outside tailings pond is not utilized allows for much more imagination with respect to land shaping. Considerable variety may be worked into the post-mining landscape by taking advantage of the overall swell of the mixture of overburden, reject, and dry tailings.

e. The siting of out-of-pit tailings dykes in the Minimum and Improved Levels of Reclamation is often difficult since these must be positioned to avoid sterilization of oil sands. This is much less of a concern at the Enhanced Level since the outside tailings dump can be located nearer to the pits and be blended into the remainder of the in-pit backfill.

f. The relative merits of wet tailings and sludge pond surfaces being below or above the former land surface should be further investigated. The re-establishment and behavior of the long term post-mining groundwater regime may be the dominant influence. The overall disposal scheme selected because of efficient materials handling characteristics may have to be altered should contrary conclusions be established regarding the post-mining groundwater regime.

g. A tailings disposal plan must be completed with respect to mass balance, disposal schedule, volume, and siting prior to the start of mining. The expansion of a mine beyond its initially planned boundary may be impossible with respect to tailings disposal unless special provisions are made in the mine plan for such a course of action. Additional tailings disposal sites may be required, and these may not be available at that time. Use of additional out-of-pit areas as an afterthought, so to speak, may create serious disposal problems for neighbouring mines.

h. Wet tailings disposal costs rise rapidly as the quantity of dyked sands compared to spigotted sand increases. The amount of dyking can be reduced by utilizing the pit wall as a dyke. In any given mine plan, it must be determined whether this practice is acceptable with regards to oil conservation. The sterilization of pit-boundary ore must be weighed against the cost of out-of-pit disposal either on or off-lease, or the deposition of tailings using dyking that prevents or at least minimizes the contact of tailings with low grade in-place oil sands remaining in the pit wall. The construction of long overburden dykes increases the tailings disposal costs significantly.

i. Conflicting evidence exists as to whether the net inflow of water from rainfall would be balanced by the evaporation from the pond, and to a minor degree, the transpiration and seepage from the reclaimed dyke. Some sources claim that a balance exists, while others claim that there would be a net increase in pond water levels.

10.4 OBJECTIVES IN RECLAMATION

The purpose of this study is not to establish firm guidelines for the reclamation of oil sands mines. Rather, the objective is to show the range of operating possibilities that exist, and the associated costs. For instance, this report does not state that a certain mixture and depth of overburden and muskeg is required for successful reclamation, but rather that a number of possibilities with increasing likelihood for successful long term reclamation are possible. Local conditions would require modification to any set of suggestions which could be developed.

Furthermore, the study has been designed to demonstrate advantages to reclamation over the range of operating conditions described: Minimum, Improved, Enhanced. In many respects the terms Minimum, Improved, and Enhanced are measures of "reclaimability". The prepared soil manufacturing method described at each level is only one element in the whole concept of reclaimability. An Enhanced Level prepared soil manufacturing scheme would, with some modification, also work with a Minimum Level tailings disposal scheme. Alternatively, it may be desirable to choose differing ratios or depths of materials than those suggested. In any case, the cost data provided are helpful in estimating the costs against the added advantages of a particular reclamation scheme.

Until examples of long term success in reclamation under various conditions have been demonstrated, and until experimental evidence on the depth and amendment requirements and physical and chemical characteristics of overburden materials is accumulated, there will be reason to question the reclamation proposals contained herein. This would suggest either of two courses of action: reclaim to a level higher than may appear to be adequate, or provide for the possibility of upgrading the reclamation in the future.

The former alternative must, of necessity, rely on reclamation experience elsewhere, as demonstrated in other areas of Alberta and worldwide. Currently, such experience indicates that depths of prepared soil in the neighbourhood of one metre are required.

The latter alternative requires the stockpiling of selected materials in locations where they are readily accessible if and when needed. The hazard inherent in this alternative is that there may be insufficient material available to adequately improve the reclamation, either because a large amount was wasted in the first unsuccessful attempt, or because insufficient material was stockpiled. Certain combinations of site-specific conditions may completely rule out any opportunity for corrective action in the future.

The use of both muskeg and suitable overburden materials in the manufacture of prepared soil is, to a great extent, site specific. The availability of muskeg to meet the conditions outlined for the three cited levels of reclamation must be determined before a commitment to a certain type of prepared soil manufacturing method is made. Likewise, the availability of suitable overburden, both in total and with respect to time, must be determined for each mine plan. An optimum combination of material mixtures and schedules of reclamation with due regard to costs can then be developed for each mine plan. Such planning may indicate that earlier completion of reclamation may only be possible at the expense of lower quality of overburden being utilized in the manufacture of prepared soil.

The direct transfer of muskeg and overburden from the stripping operation at the Minimum and Improved Levels of Reclamation is possible only under special conditions. Otherwise, stockpiling is required until the time that the products are required for reclamation purposes. Site specific influences related to the location, quantity, and quality of the muskeg and overburden sources, combined with the shape of the ore body, are the most crucial factors affecting direct transfer potential.

The precise depth and component ratio of prepared soil to be utilized must reflect the availability of the component materials as well as the depth requirement demanded by the inherent characteristics of underlying materials. A guideline should be developed which sets the depth of prepared soil according to end land use, characteristics of component materials, and characteristics of underlying material. The 1:2 ratio of muskeg to overburden used to develop the costs is likely the highest muskeg level tolerable in a prepared soil. Trafficability and scouring resistance will improve with an increase in prepared soil depth and may, in fact, be the major consideration for the development of a commercial forest land use. A guideline based on the characteristics of the components of prepared soil, and the characteristics of underlying materials to be reclaimed will simplify the preparation of reclamation plans as well as the approval of plans.

The selection of plant species for revegetation depends on the planned final land uses. Consideration must be given to the avoidance of planting practices with conflicting objectives. For instance, areas designated for wildlife browsing should be planted with appropriate browse species. Areas to be planted with trees should only be seeded with companion grasses after carefully weighing the harmful effects of elevated rodent populations which are associated with low survival rates for tree plantings, and the effects of competition for moisture. When forestry is the planned land use, attempts should be made to check erosion by constructing sufficiently shallow slopes, adequate drainage ditches, functional settling ponds, durable service roads, etc., rather than endanger tree planting success by the presence of thick mats of grass.

Grass plantings on tailings dykes at GCOS and Syncrude are present primarily for the purposes of erosion control. In time, trees may naturally colonize the slope, but this approach to reclamaton will require a much longer period of commitment to foster satisfactory reclamation. It is not apparent whether the economic advantage lies with a very long term commitment to maintenance, at relatively low initial cost, or with a high initial outlay scheme that achieves success and self-sufficiency earlier.

In this study it is presupposed that only dry areas can be reclaimed. Thus, much thought is given to maximizing the dry land area created at each level of reclamation. The sludge from conventionally operated tailings ponds can be concentrated in sludge ponds. A further reduction of wet surface area could be obtained by partial dewatering of sludge.

Operational experience at the GCOS tailings pond indicates that during plant shut-down, when no tailings are being added to the pond, the surface is free of bitumen within two weeks. At these times, ducks appear not to be affected by other constituents of the pond water. More research on the long term effects on waterfowl using tailings ponds must be completed however, before the risks can be completely discounted. At the Enhanced Level, water quality is no longer a concern since only fresh water lakes will remain in the post-mining environment.

Unless reclaimable dry land surfaces are developed on sludge ponds, there appears to be no reason to expect that such ponds will support vegetation before very many years to come. However, possibilities for thickening or poldering sludge pond surfaces should not be ignored.

Final voids that are not used by neighbouring mines for tailings disposal should be considered for development as fresh water lakes after completion of mining. This would require filling with water from the Athabasca River. The pumping station formerly used to supply fresh water to the plant would be used to fill the end-pit.

The disposal of saline water generated by the mine dewatering operation is not addressed in depth in this study. The possibility of collecting the saline water in one saline lake formed from a mined-out pit appears attractive. Total confinement may be possible. If quantities are large, a gradual, controlled release into the Athabasca River from one or more control stations may be possible. There are many examples in western Canada of both large and small lakes with extremely high salt concentrations. A saline lake formed in a mined-out pit would not be any less attractive. A regional assessment is required to determine if this is a practical solution.

10.5 ENERGY CONSUMPTION IN RECLAMATION ACTIVITIES

An objective of the study is to measure the direct consumption of energy required to achieve various reclamation targets. Ore body No. 2 is examined in this respect, since it includes mine plans at all three levels of reclamation. The consumption of energy is estimated on the basis of operating costs. Since the unit operating costs include a known expense for oil, gas, and electricity, these amounts are split from the rest of the operating costs. The costs of fertilizer are included since fertilizer is for all practical purposes energy, although in the nonconventional form. The costs are generated by means of a computer program, using only the energy portions of the operating costs. Table 10.5-1 gives a comparison between the three 120,000 BPCD dragline mines.

TABLE 10.5-1

Mine Plan		Direct Energy Cost ($/bbl of crude)		
		Cost Centre 4.0	Cost Centre 5.0	Cost Centre 4.0 + 5.0
Minimum:	Dragline	0.072	0.006	0.078
Improved:	Dragline	0.119	0.021	0.140
Enhanced:	Dragline	0.135	0.015	0.150

The total energy consumption for each barrel of crude oil produced cannot be estimated from the above costs alone. Operating costs of the extraction plant and upgrading plant, utilities, offices, shops and warehouses were not developed. The "net energy cost" of reclamation is determined by comparing the total costs of mine plans against each other. Each cost centre contributes to the net cost but Cost Centre 4.0, Tailings Disposal, and Cost Centre 5.0, Establishment of Ultimate Land Use Resources, are the most important. Those two cost centres account for the largest portion of the added energy cost due to reclamation requirements alone.

The figures show a rapid increase of energy costs between plans at the Minimum and Improved Levels, the combined increase for Cost Centre 4.0 and 5.0 being twofold. However, the increase from the Minimum to the Enhanced Level is only slightly greater. In other mine plans the costs due to reclamation will be of the same order-of-magnitude.

The cost of fertilization is a portion of the total cost of Cost Centre 5.0. In preparing the costs, repeated annual fertilizations were assumed. This implies that a hectare reclaimed in Year 5 would be repeatedly fertilized for 25 years by Year 30. This amount of fertilization is likely unnecessary. Possibly only 5 or 10 years of refertilization would be sufficient. The following tabulation compares the costs of repeated fertilization for 25 years against the costs for fertilizing only 5 years for the dragline scheme of Ore Body 2. Fertilization rate is assumed to 340 kg per hectare per year at $65 per hectare per year.

Mine Plan	Fertilizer Costs per 1000 Barrels (Total)	
	5 Years	25 Years
Minimum Level	0.9 cents ($950,000)	2.3 cents ($2,400,000)
Improved Level	0.8 cents ($850,000)	1.6 cents ($1,700,000)
Enhanced Level	0.8 cents ($800,000)	2.3 cents ($2,400,000)

Although comparatively small on a 25 year basis, fertilization does rank among some of the items estimated in Cost Centre 5. On a 5 year period of fertilization, the cost is reduced to less than 1 cent per 1,000 barrels or $1,000,000 in total. This simple comparison can be repeated for all the mine plans by accumulating the cost of fertilization for the areas to be reclaimed shown in reclamation tables in Chapter 7.0, 8.0, and 9.0.

A determination of energy costs of manufacturing the machinery and materials (other than fertilizers) consumed by the operation, or the energy consumed by the labour force, etc. would lead to much larger costs. Such cost analysis is not impossible, but implies a complete assessment of all the energy indirectly purchased, which is beyond the scope of this study.

There is an indirect operating cost associated with each direct cost. A small portion of this is in the form of conventional energy consumed, for example in the running of offices, staff transport, etc. A further explanation with respect to direct and indirect costs is contained in Section 10.7.

10.6 SUMMARY OF PHYSICAL CHARACTERISTICS OF MINES

The physical characteristics of the mine plans developed in this study are numerically expressed in Table 10.6-1. Part A of the table is specifically related to mining, and in many respects the overall economics of the mines. Part B summarizes overall physical characteristics of the mine such as pit area, tailings pond areas and dump areas. Parts C and D are comparisons of reclaimability of each mine in terms of size and Level of Reclamation.

Mining Quantities (Part A)

In part A, in order to comparatively evaluate the Ore Bodies, the following quantities were expressed on a unit basis per barrel of synthetic crude oil: total mining volume, overburden, centre reject, total waste (OB+CR), and plant feed. Also shown are R-Factor and plant feed grade.

The volume of waste is composed of overburden, top reject, centre reject and mining loss. Plant feed constitutes ore grade oil sand and some dilution material from reject/oil sand interfaces or from centre reject when these are too thin to be separated by draglines or bucket wheels. Tonnes of bitumen in the plant feed are shown as well as barrels of synthetic crude produced by the extraction and upgrading processes.

Table 10.6-1 shows mining R-Factors which are lower than the R-Factors shown in Tables 2.7-1 to 2.7-4. The latter tables are based on geology only (pit walls are assumed vertical), while Table 10.6-1 is based on actual mining quantities. The total quantity of material moved will increase in volume because extra overburden must be removed at the pit walls, and because of dilution at the ore-reject interfaces (see R-Factor definition).

It can be seen that the mining R-Factors for Ore Bodies 4 and 2 (dragline scheme) are very close. The difference between the two ore bodies

SUMMARY OF PHYSICAL CHARACTERISTICS OF MINES

TABLE 10.6-1

	ORE BODY NO. 4, 60,000 BPCD				ORE BODY NO. 2, 120,000 BPCD						ORE BODY NO. 1 240,000 BPCD	
	DRAGLINE SCHEME		B.W.E. SCHEME		DRAGLINE SCHEME			B.W.E. SCHEME			B.W.E. SCHEME	
	Minimum	Improved	Minimum	Improved	Minimum	Improved	Enhanced	Minimum	Improved	Enhanced	Minimum	Enhanced
A. Mining Quantities												
Waste (bm^3x10^6)	291.358	291.358	289.905	289.905	511.381	511.381	511.381	478.997	478.997	478.997	1,283.989	1,283.989
Plant feed (bm^3x10^6)	466.379	466.379	461.629	461.629	983.521	983.521	983.521	901.200	901.200	901.200	1,966.147	1,966.147
Bitumen (tonnesx10^6)	109.068	109.068	108.306	108.306	214.643	214.643	214.643	204.193	204.193	204.193	443.209	443.209
Barrels Crude (bblx10^6)	532.700	532.700	527.193	527.193	1,029.329	1,029.329	1,029.329	982.643	982.643	982.643	2,138.103	2,138.103
Mining R-Factor	.686	.686	.687	.687	.685	.685	.685	.706	.706	.706	.650	.650
Average Plant Feed Grade (% by weight)	11.68	11.68	11.72	11.72	10.90	10.90	10.90	11.32	11.32	11.32	11.26	11.26
Mining Volume per Barrel of Crude (bm^3/bbl)	1.422	1.422	1.426	1.426	1.426	1.452	1.452	1.405	1.405	1.405	1.520	1.520
Overburden per Barrel of Crude (bm^3/bbl)	.432	.432	.425	.425	.324	.324	.324	.291	.291	.291	.457	.457
Centre Reject per Barrel of Crude (bm^3/bbl)	.115	.115	.125	.125	.166	.166	.166	.196	.196	.196	.144	.144
Total Waste per Barrel of Crude (bm^3/bbl)	.547	.547	.550	.550	.490	.490	.490	.487	.487	.487	.601	.601
Plant Feed per Barrel of Crude (bm^3/bbl)	.876	.876	.876	.876	.955	.955	.955	.917	.917	.917	.920	.920
B. Surface Area Affected												
Pit Area (ha/10^9bbl)	2378	2378	2403	2403	2450	2450	2450	2278	2278	2278	2006	2006
Out-of-Pit Pond (ha/10^9bbl)	1937	2632	1958	2659	1259	1259	0	1194	1194	0	2117	0
Outside Dumps (ha/10^9bbl)	533	268	721	336	762	435	168	199	0	1093	790	260
Total (Area C&D) (ha/10^9bbl)	4849	5279	5082	5398	4471	4144	2618	3671	3471	3371	4913	2267
Total (Area C&D) (ha)	2583	2812	2679	2846	4602	4266	2695	3607	3411	3312	10505	4847
C. Reclamation Area												
Out-of-Pit Pond (ha/10^9bbl)	1937	2632	1958	2659	1259	1259	0	552	1194	0	2117	0
Outside Dumps (ha/10^9bbl)	533	268	721	336	762	435	168	199	0	1093	790	260
Inside Ponds (ha/10^9bbl)	237	0	239	0	963	862	0	576	759	0	538	0
Inside Dumps (ha/10^9bbl)	409	978	347	988	150	408	2271	0	0	1752	372	1665
End-Pit Lake (ha/10^9bbl)	925	989	1002	1020	566	762	179	1059	1083	525	580	342
D. Unreclaimed Areas												
Out-of-pit Pond (wet) (ha/10^9bbl)	0	0	0	0	0	0	0	642	0	0	0	0
In-pit Pond (wet) (ha/10^9bbl)	807	411	816	395	711	419	0	641	435	0	516	0

becomes evident when the average plant feed grades are compared. Ore Body 4 has a high ore grade with thick overburden, while Ore Body 2 has a low ore grade with much less overburden.

It should be noted that the apparent physical differences between the bucket wheel and dragline schemes of Ore Body 2 are mainly due to the use of updated geological information in the dragline scheme. The pit area in the bucket wheel version of Ore Body 2 is 170 hectares smaller in size when compared to that of the dragline mine. The mine size difference caused by the type of excavator employed, however, is expected to be a maximum of 50 hectares.

In contrast, the difference of 25 hectares between the dragline and bucket wheel plans in Ore Body 4 is caused by a variation in mine plan caused by the difference in operating characteristics assumed for the dragline and bucket wheel excavators. Consequently, when comparing the bucket wheel and dragline schemes of Ore Body 4, it is apparent that the selectivity criteria have only minor effects on the overall distribution of mining quantities. With 1.0% more plant feed and 0.5% more waste, the dragline scheme produces only 1.0% more synthetic crude.

It has become apparent that the selectivity criteria used in this study are not the decisive factors in making the selection of prime excavators. Depending on the size of the mine, the capital and operating costs of prime excavators are far better indicators upon which to base the choice of the excavator.

Comparing the three ore bodies on a per barrel basis reveals that Ore Body 4 has the best ore, i.e. highest grade plant feed and least centre reject, but relatively thick overburden. Ore Body 2 has the poorest ore, i.e. lowest grade plant feed and most centre reject, but least overburden. Thin overburden accounts for the lowest total waste per barrel in Ore Body 2. Ore Body 1 has an average plant feed grade halfway in between that of the other two ore bodies, but the very thick overburden in this mine results in the highest total waste (OB and CR) per barrel of crude. Ore Body 1 has the highest mining volume per barrel of crude and consequently the poorest R-Factor.

Overall, the summary indicates that the ore bodies studied are very comparable and will allow a good comparison to be made with respect to relative reclaimability and costs.

Surface Area Affected (Part B)

Parts B,C and D of Table 10.6-1 are related primarily to reclamation. In order to make a comparison between different mine plans, the reclaimed and unreclaimed surface areas are expressed in hectares per barrel of synthetic crude oil.

Pit Areas: Ore Bodies 2 and 4 have approximately the same area per barrel since the average pay zone thickness is almost identical. Ore Body 1 shows the smallest number of hectares per billion barrels of synthetic crude as it has the thickest pay zone. The range of pit areas is from 2000 to 2500 hectares per billion barrels.

Out-of-Pit Ponds: At the Minimum Level, Ore Body 2 has an areally small but high tailings pond sited on a relatively flat ground surface. The pond is dyked along its entire perimeter. Ore Bodies 4 and 1 have tailings ponds with lower crest elevations and are built more or less on a land slope. The size of the out-of-pit pond is also influenced by the tailings disposal schedule, which was least flexible in Ore Body 4.

At the Improved Level of Ore Body 2, the outside tailings ponds require identical surface areas, since the out-of-pit tailings pond design from the Minimum Level plans was utilized. In contrast, at the Improved Level, the out-of-pit tailings pond of Ore Body 4 stores all tailings sand produced, thus requiring a larger surface area.

At the Enhanced Level, tailings ponds are not utilized since dry tailings sand is part of the dumping operation (see Outside Dumps).

Outside Dumps: At the Minimum Level, the dragline plans for Ore Body 2 have large volumes of overburden and reject stored out-of-pit compared to the bucket wheel scheme where overburden and reject are dumped onto the pit floor. In both Ore Bodies 2 and 4, the bottom bench dragline backcasts centre reject onto the pit floor at all times. In Ore Body 4 the out-of-pit waste storage volume in the dragline scheme is smaller compared to that in the B.W.E. scheme due to the backcasting procedure used by the bottom bench dragline.

In the Minimum Level plan for Ore Body 1, thick overburden results in large outside waste dumps. The size of these dumps is also influenced by the overall mining and tailings disposal schedule, which allows backfill to occur only when no further tailings and sludge storage capacity is needed.

At the Improved Level, the above mentioned statements regarding the Minimum Level plan characteristics also apply for Ore Body 2. In the Improved Level plans for Ore Body 4, again more waste must be stored in the BWE scheme because no backcasting is possible in the bucket wheel plan. However, earlier diversion of waste into the pit is possible at the Improved Level in Ore Body 4, since the volume of the treated sludge pond is smaller, allowing for earlier construction of the required in-pit dyke which, in turn, allows for backfilling to begin earlier in the life of the pit.

At the Enhanced Level in Ore Body 2, the bucket wheel scheme has a considerably larger waste dump than the dragline scheme, due more to a different design philosophy than a constraint of mine layout. In the parallel mining method, as used in the dragline scheme of Ore Body 2, the dumping operation with conveyors and spreaders can follow the bottom bench more closely than is possible in slewing operations. In general, parallel mining should result in an earlier diversion of dumping into the pit, and provide for a smaller outside dump.

Total Areas Affected (ha/10^9bbl): The smallest disturbed area per barrel of synthetic crude results from the employment of dry tailings disposal schemes outlined in the Enhanced Level of Reclamation. For a given ore body (pit), only relatively small additional disturbance is caused by out-of-pit dumping of preproduction overburden and one to four production years of dry tailings sand, reject and overburden. A range of 2,300 to 3,400 hectares per billion barrels of synthetic crude is seen. The unit disturbance appears to decrease with mine size.

In general, the Improved Level of Reclamation should result in a somewhat smaller area of total disturbance than the Minimum Level, as shown by the trends in Ore Body 2 for both the dragline and bucket wheel plans. The reason the area is larger at the Improved Level than at the Minimum Level in Ore Body 4 is that there is a larger outside tailings pond because of the inflexibility of the tailings disposal scheme for a small ore body, as discussed elsewhere in Chapter 10.

The comparison of the five Minimum Level mine plans reveals that the Ore Body 2 bucket wheel scheme has the smallest total disturbed area per barrel of crude. This is due to the gross reduction of the outside waste dumps, since almost all of the waste is dumped onto the pit floor before in-pit tailings ponds are constructed on top of it. However, this is at the expense of raising the mean crest elevation of in-pit ponds. The second lowest areal disturbance is in the Ore Body 2 dragline scheme; this is due to its higher and areally smaller outside tailings pond. The remaining three Minimum Level plans of Ore Bodies 4 and 1 have approximately the same surface disturbance, 5000 hectares per billion barrels of synthetic crude produced.

In addition to the areal extent of surface disturbance, the relative quality of disturbance must be considered. The Enhanced Level yields dry and relatively easily reclaimable areas. In the Improved Level plans, the surface disturbance includes a pond of thickened sludge which has, at considerable cost, a higher reclamation potential than the untreated sludge in the ponds of the Minimum Level. In four of the five Minimum Level plans, all sludge is rehandled into one in-pit pond and all tailings ponds are sanded-in in an attempt to reduce the wet tailings surface area. Only one plan (the Ore Body 2 bucket wheel scheme) leaves a series of wet tailings ponds with no sludge rehandle.

<u>Total Areas Affected (ha)</u>: The same total area of surficial disturbance as above is expressed in absolute numbers (in hectares).

Reclamation Area (Part C)

Out-of-Pit Ponds: The discussion of Outside Tailings from Part B applies here also, the only exception being the reclamation of the out-of-pit tailings pond in the Minimum Level bucket wheel plan for Ore Body 2. In this plan, the out-of-pit pond is a wet pond and only beaches and dykes can be reclaimed. All the out-of-pit tailings ponds from other schemes are completely sanded-in, covered with prepared soil, and revegetated.

Outside Dumps: All outside dumps are reclaimed. For comparison of areal extent, see discussion in Part B above. Only muskeg is added at the Minimum Level and a prepared soil is manufactured by onsite blending with overburden. At the Improved and Enhanced Level, prepared soil (blended muskeg and overburden) is hauled onto the reclamation site using trucks.

In-pit Ponds: The extent of these is dependent mainly on the tailings disposal scheme used. A large number here will generally indicate a desirable tailings disposal scheme. Large reclaimable (sanded-in) tailings ponds reduce the size of the outside tailings pond. Ore Body 4 demonstrates the inflexibility of a small pit with respect to tailings disposal design. The Ore Body 2 dragline plan at the Minimum and Improved Levels and Improved Level bucket wheel plan have the best "wet" tailings disposal schemes as far as the extent of surface disturbances is concerned. These plans feature small, reclaimed out-of-pit tailings ponds combined with large, reclaimed in-pit ponds. To compare these three "wet" schemes further, see the discussion of unreclaimed areas (in Part D).

Inside Dumps: The largest areal utilization values appear for Enhanced Level schemes. Again, the larger the number the more desirable the scheme. The larger number indicates a reduction of the volume of waste being stored in outside waste dumps. The Ore Body 2 Minimum and Improved Level bucket wheel plans have no inside waste dumps at the surface. In these plans the backfill is covered by tailings ponds or an end-pit lake and, in such cases, zero or a low number also indicates a minimization of out-of-pit waste storage. At the Improved level of Ore Body 4, large volumes of waste are backfilled into the mined-out pit. This is possible because all tailings sand is stored in a sanded-in out-of-pit tailings pond. As a rule, it may be more desirable to store more tailings sand in-pit at the expense of some extra waste having to go to the outside waste dump. This is not practical in Ore Body 4 at the Improved Level because of constraints inherent in the tailings disposal scheme, as discussed previously. The objective for all the plans is to backfill as much of the waste into the pit as the mining and tailings disposal schedules would permit.

<u>End-pit Lakes</u>: A portion of the pit which is remaining after in-pit tailings disposal and waste backfill are completed is filled with water to create a lake. Its size is a consequence of the tailings disposal and backfill scheme employed. In the Enhanced Levels only small lakes are formed, mainly because it is possible to put most of the dry tailings sand, reject, and overburden back into the pit. Ore Body 2 Minimum and Improved Level bucket wheel plans have large end-pit lakes. A considerable amount of backfill has been submerged by the lake. The mine design purposely included large end-pit lakes as a reclamation feature. The dragline versions of the above were designed so as to limit the extent of the end-pit lake. The lakes seen in the plans for all levels of Ore Bodies 4 and 1 are strictly a result of the tailings disposal and backfill schedules. In general, the unit area of end-pit lakes decreases with mine size.

<u>Unreclaimed Areas</u> (Part D)

<u>Out-of-Pit Ponds</u>: The only scheme using an out-of-pit tailings pond where sludge and water remain after completion of mining is the bucket wheel plan for the Minimum Level in Ore Body 2. In many respects, it is considered less desirable to have residual tailings water and sludge stored at a high elevation in out-of-pit tailings ponds as compared to being stored at a lower elevation in an in-pit pond. This tailings disposal concept was incorporated in order to make a comparison of alternatives.

In-Pit Ponds: The most desirable schemes are those of the Enhanced Level, where there are no unreclaimed surfaces (no tailings water or sludge). A comparison of the five Minimum Level schemes shows that Ore Body 4 has the largest unreclaimable in-pit surface. The reason for this is that approximately five years of tailings sand was beached into the sludge pond, which at shallow beach slopes caused the ratio of sludge surface to sludge volume to increase. While the Minimum Level bucket wheel plan for Ore Body 2 does not have the largest unreclaimed area in-pit, this plan has the largest area of unreclaimable surface area when combined with unreclaimable out-of-pit tailings pond area, approximately 1,300 hectares per billion barrels of synthetic crude produced. In general, the amount of unreclaimed surface area decreases on a unit basis as the mine size increases. The unreclaimed area in the Improved Level is approximately the same in all four schemes studied. The surface area of treated sludge, which is the unreclaimable surface specified, is roughly half of the unreclaimed area of the Minimum Level (except the Ore Body 2 bucket wheel plan) as the volume of sludge is reduced by 50% when undergoing the treatment. It appears that the unreclaimed area for Improved Level plans remains constant near 400 to 600 hectares per billion barrels of synthetic crude.

10.7 COST COMPARISONS

The cost analysis conducted for this study analysed 44 distinct cost items under six distinct major cost groupings. These are:

Cost Centre 1: Civil Construction-Type Activities
Cost Centre 2: Removal of Organic Materials and Soils
Cost Centre 3: Overburden Reject, Oil Sands Handling
Cost Centre 4: Tailings Disposal
Cost Centre 5: Establishment of Ultimate Land Use Resources
Cost Centre 6: Supervision, Technical Services

In an oil sands mining and extraction operation, many other significant costs are incurred. The cost items selected for comparison by this study are those which are liable to undergo change when reclamation objectives are changed.

The costs summarized in Table 10.7-1 are direct operating costs. As well as direct costs, a mine will incur indirect costs associated with the direct costs. Indirect costs will include: overhead costs in the form of general administrative services required in the course of conducting work (administrative and clerical staff, warehousing, office facilities, transportation, etc.); financing costs incurred as a result of borrowing capital to place the mine in operation and working capital required to carry out the daily financial transactions of the operation; royalty and taxation costs (payable to the federal, provincial and municipal governments, etc.); and costs for various miscellaneous services which must be purchased and supplied to the mining and extraction and upgrading operations.

The terms of reference of this project limited the scope of the costing activities to mining and reclamation activities. Consequently, the cost analysis ignored the cost of the extraction plant, of the upgrading plant, of the utility plant and utilities, and of other miscellaneous onsite and offsite facilities (such as bridges, roads, infrastructure, etc.). Furthermore, costs whose evaluation would not affect the conclu-

SUMMARY OF COST ANALYSIS ($/bbl.)

TABLE 10.7-1

COST ITEM	ORE BODY NO. 2, 120,000 BPCD						ORE BODY NO. 4 60,000 BPCD				ORE BODY NO. 1 240,000 BPCD	
	DRAGLINE SCHEME (1,029,329,000 bbls.)			B.W.E. SCHEME (982,643,000 bbls.)			DRAGLINE SCHEME (532,700,000 bbls.)		B.W.E. SCHEME (527,193,000 bbls.)		B.W.E. SCHEME (2,138,103,000 bbls.)	
	Minimum	Improved	Enhanced	Minimum	Improved	Enhanced	Minimum	Improved	Minimum	Improved	Minimum	Enhanced
COST CENTRE 1: Civil Construction-Type Activities	0.0588	0.0588	0.0699	0.0436	0.0436	0.0530	0.0681	0.0685	0.0656	0.0657	0.0455	0.0580
1.1 Mine-Power Distribution & Control	0.0492	0.0492	0.0601	0.0336	0.0336	0.0428	0.0501	0.0505	0.0474	0.0475	0.0377	0.0500
1.2 Buildings	0.0096	0.0096	0.0097	0.0100	0.0100	0.0102	0.0180	0.0180	0.0182	0.0182	0.0078	0.0080
COST CENTRE 2: Removal of Organic Materials & Soils	0.0475	0.0455	0.0455	0.0464	0.0464	0.0405	0.0590	0.0582	0.0596	0.0589	0.0517	0.0419
2.1 Clearing	0.0053	0.0053	0.0053	0.0058	0.0058	0.0058	0.0076	0.0068	0.0077	0.0069	0.0060	0.0052
2.2 Muskeg Dewatering	0.0019	0.0019	0.0019	0.0017	0.0017	0.0017	0.0028	0.0028	0.0028	0.0028	0.0015	0.0015
2.3 Muskeg Loading	0.0176	0.0176	0.0176	0.0156	0.0156	0.0156	0.0197	0.0197	0.0199	0.0199	0.0145	0.0145
2.4 Muskeg Hauling (incl. Road Maintenance)	0.0115	0.0094	0.0094	0.0109	0.0135	0.0076	0.0156	0.0156	0.0157	0.0157	0.0203	0.0124
2.5 Muskeg Placement	0.0086	0.0086	0.0086	0.0076	0.0076	0.0076	0.0096	0.0096	0.0097	0.0097	0.0070	0.0070
2.6 Muskeg Road Construction	0.0025	0.0026	0.0026	0.0048	0.0022	0.0022	0.0038	0.0038	0.0039	0.0039	0.0024	0.0013
COST CENTRE 3: Overburden, Reject, Oil Sands Handling	1.3396	1.3434	1.3434	0.9026	0.9026	0.9026	1.2645	1.2682	1.2683	1.2807	1.1145	1.1145
3.1 Overburden B.W.E.	0.0772	0.0772	0.0772	----	----	----	0.1104	0.1104	----	----	----	----
3.2 Oil Sands, Draglines & Hoppers	0.3943	0.3943	0.3943	----	----	----	0.3426	0.3426	----	----	----	----
3.3 B.W.E. (Overburden and Oil Sands)	----	----	----	0.3209	0.3209	0.3209	----	----	0.3828	0.3828	0.3156	0.3156
3.4 Transport (All Conveyors)	0.6171	0.6209	0.6209	0.3658	0.3658	0.3658	0.4969	0.5006	0.5699	0.5823	0.5340	0.5340
3.5 Placement (Spreaders)	0.0577	0.0577	0.0577	0.0608	0.0608	0.0608	0.0788	0.0788	0.0932	0.0932	0.0837	0.0837
3.6 Miscellaneous Equipment	0.1934	0.1934	0.1934	0.1551	0.1551	0.1551	0.2357	0.2357	0.2224	0.2224	0.1812	0.1812
COST CENTRE 4: Tailings Disposal	0.4465	1.2945	0.3072	0.4468	1.1110	0.2210	0.4509	1.0428	0.4444	1.0545	0.4034	0.3257
4.1 Area Drainage	0.0002	0.0002	----	0.0003	0.0003	----	0.0003	0.0003	0.0003	0.0003	0.00003	----
4.2 Clearing	0.0020	0.0020	----	0.0021	0.0021	----	0.0034	0.0044	0.0034	0.0044	0.0033	----
4.3 Construction of Starter Dams and Overburden Dams	0.0413	0.2068	----	0.0713	0.0708	----	0.0546	0.0506	0.0453	0.0512	0.0185	----
4.4 Piping of Wet Tailings or Conveying of Dry Tailings	0.1824	0.1970	0.2438	0.1750	0.1707	0.1539	0.1506	0.1579	0.1521	0.1598	0.1716	0.2554
4.5 Tailing Sand Placement into Dyke	0.0370	0.0338	---	0.0398	0.0346	----	0.0446	0.0470	0.0451	0.0475	0.0185	----
4.6 Tailings Overboarding & Sanding or Placement of Dry Tailings	0.0576	0.0608	0.0634	0.0524	0.0552	0.0671	0.0542	0.0560	0.0541	0.0560	0.0595	0.0702
4.7 Recyling of Tailings Water	0.0350	0.0333	----	0.0285	0.0294	----	0.0558	0.0477	0.0563	0.0482	0.0392	----
4.8 Rehandling of Tailings Sludge	0.0215	----	----	----	----	----	0.0129	----	0.0130	----	0.0238	----
4.9 Sludge Treatment	----	0.6912	----	----	0.6703	----	----	0.6074	----	0.6137	----	----
4.10 Power Distribution	0.0009	0.0007	----	0.0004	0.0007	----	0.0010	0.0008	0.0010	0.0008	0.0005	----
4.11 Oversize Reject Disposal	0.0685	0.0685	----	0.0755	0.0755	----	0.0732	0.0701	0.0732	0.0721	0.0682	----
4.12 Oversize Reject Disposal Road Construction	0.0001	0.0001	----	0.0015	0.0015	----	0.0006	0.0006	0.0006	0.0004	0.0002	----
COST CENTRE 5: Establishment of Ultimate Land Use Resources	0.0501	0.1273	0.1712	0.0253	0.1214	0.1931	0.0412	0.1799	0.0429	0.1846	0.0556	0.1194
5.1 Muskeg Rehandle Loading	0.0035	0.0083	----	0.0017	0.0068	----	0.0035	0.0113	0.0036	0.0115	0.0049	----
5.2 Muskeg Rehandle Hauling (Including Road Maintenance)	0.0031	0.0065	----	0.0018	0.0021	----	0.0028	0.0024	0.0030	0.0025	0.0086	----
5.3 Muskeg Rehandle Placement	0.0035	0.0021	----	0.0004	0.0017	----	0.0009	0.0028	0.0009	0.0033	0.0012	----
5.4 Muskeg Rehandle Road Construction	0.0070	0.0007	----	0.0011	0.0002	----	0.0023	0.0006	0.0023	0.0006	0.0063	----
5.5 Overburden Rehandle Loading	0.0042	0.0147	0.0021	0.0027	0.0120	0.0013	0.0032	0.0198	0.0032	0.0203	0.0040	0.0013
5.6 Overburden Rehandle Hauling (Including Road Maintenance)	0.0037	0.0063	----	0.0070	0.0033	----	0.0042	0.0120	0.0042	0.0122	0.0056	----
5.7 Overburden Rehandle Placement	0.0010	0.0036	----	0.0007	0.0029	----	0.0008	0.0048	0.0008	0.0049	0.0010	----
5.8 Overburden Rehandle Road Construction	0.0005	0.0004	----	0.0001	0.0004	----	0.0003	0.0007	0.0003	0.0007	0.0002	----
5.9 Muskeg Mining, Slurry Transport & Dewatering	----	----	0.0373	----	----	0.0400	----	----	----	----	----	0.0253
5.10 Prepared Soil Manufacture	0.0007	0.0085	0.0736	0.0003	0.0069	0.0799	0.0007	0.0114	0.0007	0.0117	0.0008	0.0441
5.11 Prepared Soil Loading, F.E.L. & Trucks	----	0.0178	0.0135	----	0.0146	0.0184	----	0.0240	----	0.0246	----	0.0120
5.12 Prepared Soil Transport, Trucks (Incl. Road Maintenance)	----	0.0277	0.0126	----	0.0436	0.0114	----	0.0486	----	0.0495	----	0.0108
5.13 Prepared Soil Placement, Trucks	----	0.0057	0.0043	----	0.0047	0.0059	----	0.0077	----	0.0079	----	0.0038
5.14 Prepared Soil Road Construction	----	0.0079	0.0061	----	0.0066	0.0082	----	0.0070	----	0.0071	----	0.0054
5.15 Seed Bed Preparation, Maintenance	0.0228	0.0173	0.0217	0.0094	0.0156	0.0279	0.0227	0.0267	0.0239	0.0278	0.0230	0.0167
COST CENTRE 6: Supervision, Technical Services	0.1870	0.1870	0.1904	0.1959	0.1959	0.1995	0.3531	0.3531	0.3568	0.3568	0.1499	0.1531
6.1 Equipment Maintenance (Staff only)	0.0694	0.0694	0.0694	0.0727	0.0727	0.0727	0.1341	0.1341	0.1355	0.1355	0.0562	0.0562
6.2 Planning (Staff only)	0.0624	0.0624	0.0624	0.0654	0.0654	0.0654	0.1124	0.1124	0.1135	0.1135	0.0444	0.0444
6.3 Mining (Staff only)	0.0552	0.0552	0.0587	0.0579	0.0579	0.0614	0.1067	0.1067	0.1078	0.1078	0.0493	0.0525
TOTAL COSTS ($/bbl)	2.1294	3.0565	2.1276	1.6606	2.4209	1.6097	2.2369	2.9709	2.2377	3.0012	1.8207	1.8126

NOTE: Refer to Chapter 6 for Cost Centre Description.

sions of the study were ignored even if these costs were mining or reclamation costs. Some of the costs not included are mine dewatering (wells and disposal), personnel transportation (staff and labour), emergency stockpile of plant feed (stackers, conveyors, reclaimers), construction and maintenance (service roads, utilities, sanitary and industrial waste disposal, etc.), blasting, pollution control (services and facilities), specialized services (engineering, surveying, training), equipment and operators for sundry duties, field maintenance shops, field utilities and various similar items not specifically mentioned in Chapter 6.0 in the descriptions of each of the 44 cost sub-centres.

A quick overview of the cost items contained in each cost centre may seem to indicate that disproportionate effort was expended in developing certain cost sub-centre items; however, to establish the overall cost for a cost centre, the various activities constituting that cost centre must be separated in a manner which will allow the item to be correctly costed. All the cost sub-centre items in given cost centres cannot be costed on the same basis, for example volume, time, or distance. As well, some cost sub-centre items have lump sum investments at regular or irregular intervals in the life of the operation. Tables detailing the components of each cost sub-centre item over the life of the operation are provided for each mine in Chapters 7, 8 and 9. Yearly details are available in the computerized cost summaries provided in Volume III.

A rough estimate of the cost delineated to the total costs of the oilsands mining operation would be as follows. In total, the omitted miscellaneous items applicable to the cited cost centres are expected to amount to approximately \$0.70, \$0.60 and \$0.50 of combined direct capital and operating costs per barrel of crude bitumen for the 60,000, 120,000, and 240,000 BPCD mines, respectively. Overhead that can be associated with the mining, tailings disposal and reclamation is estimated to be approximately 30% of the direct costs shown. In total, the costs provided in Table 10.7-1 are expected to be from one-fifth to slightly more than one-quarter of the total project capital and operating costs, assuming that the cost of the extraction and upgrading plants at the Enhanced Level remains comparable in cost to the extraction and upgrading plants at the Minimum and Improved Levels. On this basis overall costs would range from approximately $9.00 to $11.00 per barrel of bitumen. This estimate of order-of-magnitude excludes costs of financing, royalties and taxes.

In many areas, the costs provided in Table 10.7-1 show trends. These trends will be discussed in detail later in this section but some overall comments are appropriate at this point.

The overall costs ($/bbl of synthetic crude) for the twelve developed mine plans are as follows:

	Level of Reclamation		
Mine	Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	2.24	2.97	--
60,000 BPCD (Bucket Wheel)	2.24	3.00	--
120,000 BPCD (Dragline)	2.13	3.06	2.13
120,000 BPCD (Bucket Wheel)	1.66	2.42	1.61
240,000 BPCD (Bucket Wheel)	1.82	--	1.81

The cost per barrel of the Minimum and Enhanced Level at a production of 120,000 BPCD are comparable. However, a spread of approximately $.75 per barrel exists between the Minimum and Improved Levels. The 60,000 BPCD mines are the costliest to operate. The lowest costs, overall, appear to be possible for mines in the 120,000 to 240,000 BPCD range.

Reclamation costs increase with the Level of Reclamation under consideration, irrespective of mine size. The cost at the Enhanced Level is 5 to 8 times that at the Minimum Level when compared with a plan where the tailings ponds are not sanded-in and no sludge rehandle occurs, namely the 120,000 BPCD bucket wheel scheme. When sludge is rehandled at the Minimum Level, the cost spread between this level and the Enhanced Level is only 3 to 5 fold. The difference between the Minimum and Improved Level is in the 5 to 7, and 3 to 4 fold ranges, respectively.

The overall costs ($/bbl) for reclamation activities as defined in Cost Centre 5 are as follows:

	Level of Reclamation		
Mine	Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	0.04	0.18	--
60,000 BPCD (Bucket Wheel)	0.04	0.18	--
120,000 BPCD (Dragline)	0.05	0.13	0.17
120,000 BPCD (Bucket Wheel)	0.03	0.12	0.19
240,000 BPCD (Bucket Wheel)	0.06	--	0.12

The lowest costs of reclamation occur at the Minimum Level. Costs at the Improved and Enhanced Levels are comparable. The costs for the 60,000 BPCD operation at the Improved level are higher than those in the 120,000 BPCD mine, since a greater surface per barrel must be reclaimed due to disadvantageous constraints inherent in the tailings disposal scheme. In the Enhanced Level schemes, the effects of economy of scale in the prepared soil manufacturing system are evident.

The extra cost of reclamation at the Enhanced Level is largely balanced by a reduction in the cost of tailings disposal. Consequently the total costs at the Minimum and Enhanced Levels are very similar (all cost centres). An uncertainty exists, though, with respect to the cost of the extraction plant required to produce the dry tailings assumed at the Enhanced Level. The very attractive total cost may be offset by a higher extraction plant capital and operating cost.

The overall costs ($/bbl) of mining activities as defined in Cost Centre 4 are as follows:

Mine	Level of Reclamation Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	1.26	1.27	--
60,000 BPCD (Bucket Wheel)	1.27	1.28	--
120,000 BPCD (Dragline)	1.34	1.34	1.34
120,000 BPCD (Bucket Wheel)	0.90	0.90	0.90
240,000 BPCD (Bucket Wheel)	1.11	--	11.11

For the 60,000 BPCD size, the costs of overburden, reject and oil sands handling are identical for dragline and bucket wheel mining at the Minimum and Improved Levels. The Enhanced Level was not costed but it is strongly suspected that the cost trends would be similar to those of the larger mines. The lowest cost occurred in the 120,000 BPCD bucket wheel mine. The largest mine has slightly higher overburden, reject and oil sands mining costs.

The higher total cost of the 120,000 BPCD dragline plan compared to the bucket wheel plan is due to the combined effect of higher capital and operating cost for the dragline mining system as applied in Ore Body 2. In total, there are five prime excavators in the dragline scheme, compared to only three in the bucket wheel scheme. As well, the difference in parallel mining with five excavators as compared to slewing mining with three, shows up as extra conveyor capital and operating costs in the case of the dragline mine. The conveying cost is almost double in the dragline scheme.

A comparison of mining schemes at the 60,000 BPCD mine size indicates a rather different trend. In Ore Body 4 the bucket wheel mining costs are slightly higher than the costs of the dragline scheme. Both mines have the same number of excavators, i.e. three, but the influence of economy of scale with respect to equipment results in less favourable costs for the bucket wheels. Compared to Ore Body 2 the bucket wheel excavators in Ore Body 4 are not ideally sized, driving up the unit capital and operating costs of the excavator. The dragline mining costs are lower due to the more optimal match of this excavator to the ore body.

The excavating costs of Ore Body 1 are almost the same as the costs for Ore Body 2. Economy of scale is not realized in the 240,000 BPCD mine and it appears that this trend prevails for mines as small as 150,000 BPCD. Moreover, additional costs are incurred for conveyors and spreaders in Ore Body 1 compared to Ore Body 2.

Overburden, reject and oil sands transport requirements must be reckoned with. A transport system is significantly different between the dragline and bucket wheel mines (120,000 BPCD size) examined in this study. The conveying systems are complex and difficult to estimate without detailed mine planning. Undue emphasis should not be placed on the prime excavators' capabilities alone. A mine design with respect to the handling of overburden, reject and oil sands can be made only by considering combined conveying and excavating costs. The cost analysis of the twelve mining schemes indicates that transport of materials is more expensive than the excavating costs.

The overall costs ($/bbl) of tailings disposal activities as defined in Cost Centre 4 are as follows:

	Level of Reclamation		
Mine	Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	0.45	1.04	--
60,000 BPCD (Bucket Wheel)	0.44	1.05	--
120,000 BPCD (Dragline)	0.45	1.29	0.30
120,000 BPCD (Bucket Wheel)	0.45	1.11	0.22
240,000 BPCD (Bucket Wheel)	0.40	--	0.33

At the Minimum Level, the lowest cost is achieved in the largest mine but this is to some extent due to site specific conditions favourable to tailings disposal. Tailings disposal at the Improved Level is up to 3 times more costly due to primary sludge thickening costs. These costs may be offset if the thickening facility were operated in conjunction with a bitumen recovery facility. The most economic tailings disposal cost is achieved at the Enhanced Level, being one-half to three-quarters the cost at the Minimum Level.

Tailings disposal is very similar in Ore Bodies 1 and 2 at the Minimum Level. Some cost advantages seen in the Minimum Level plan for Ore Body 1 are mainly due to favourable outside tailings pond dyke construction. At the Enhanced Level of Ore Body 1, the dry placement cost, analogous to the tailings disposal costs, is higher than equivalent cost in Ore Body 2 due to extra spreaders (6 versus 2) and conveyors utilized.

The costs ($/bbl) of removal of organic materials and soils as defined in Cost Centre 2 are as follows:

	Level of Reclamation		
Mine	Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	0.06	0.06	--
60,000 BPCD (Bucket Wheel)	0.06	0.06	--
120,000 BPCD (Dragline)	0.05	0.05	0.05
120,000 BPCD (Bucket Wheel)	0.05	0.05	0.04
240,000 BPCD (Bucket Wheel)	0.05	--	0.04

The costs for this activity fall into a very narrow absolute range, i.e. 0.04 to 0.06. Removal of organic materials and soils is site specific allowing comparisons to be made only within a given mine size. Upon examination of the cost in some detail, a slight cost advantage appears to exist at the Enhanced Level.

The civil construction-type activites included in Cost Centre 1 are similar to the muskeg removal costs just discussed. Early in the study the list of items included in Cost Centre 1 was rather extensive. However, most of the items were eliminated after determining that these items remained rather proportional to the mine size or that they were not affected by a change in level of reclamation. Consequently, only two items were costed: mine-power distribution and control for the operation of the pit, and capital and operating costs of a building for staff. The following costs were determined. For all practical purposes, cost item 1.1 could be added into Cost Centre 3 and cost item 1.2 into Cost Centre 6.

	Level of Reclamation		
Mine	Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	0.07	0.07	--
60,000 BPCD (Bucket Wheel)	0.07	0.07	--
120,000 BPCD (Dragline)	0.06	0.06	0.07
120,000 BPCD (Bucket Wheel)	0.04	0.04	0.05
240,000 BPCD (Bucket Wheel)	0.05	--	0.05

The overall costs of supervision and technical service staff have been grouped into Cost Centre 6 rather than distributing the costs over various cost centres. The costs are as follows:

	Level of Reclamation		
Mine	Minimum	Improved	Enhanced
60,000 BPCD (Dragline)	0.35	0.35	--
60,000 BPCD (Bucket Wheel)	0.36	0.36	--
120,000 BPCD (Dragline)	0.19	0.19	0.19
120,000 BPCD (Bucket Wheel)	0.20	0.20	0.20
240,000 BPCD (Bucket Wheel)	0.15	--	0.15

Detailed comparison indicates that a slight advantage is realized in the dragline schemes with respect to equipment maintenance staff requirements and mining. The planning staff remains identical for both bucket wheel and dragline schemes. Overall, a significant economy of scale is released, the costs almost doubling at the 60,000 BPCD mine as compared to the 240,000 BPCD mine. At a given mine size, an increase of costs occurs at the Enhanced Level, the Minimum and Improved remaining identical.

The relative cost of each cost centre has been assessed in Table 10.7-2. On a combined capital and operating costs basis, it can be seen that overburden, reject and oil sands handling (Cost Centre 3) accounts for an average of 40 - 60% of the combined mining, tailings disposal, and reclamation costs. Tailings disposal (Cost Centre 4) is the second largest cost ranging from an average low of 15% at the Enhanced Level to an average high of 40% at the Improved Level. Reclamation-related costs (Cost Centre 5) range from an average of 2% at the Minimum Level, to 5% at the Improved Level, and to 9% at the Enhanced Level. Supervision and Technical services (Cost Centre 6) account for an overall average of 10%, civil construction-type activities (Cost Centre 1) account for 3% and removal of organic materials and soils (Cost Centre 2) account for 2%. In terms of total project costs, the percentages cited represent one-fifth to one-quarter of the costs. For example, a 10% shown on the table would indicate an equivalent to 2.5% of the total oil sands project costs (excluding taxes, royalties and financing).

In order to assess the influence of the cost of capital on the various plans, a present value analysis of costs was prepared. The summary of present value analysis of costs is shown on Table 10.7-3. In general, the trends evident before discounting remain with one major exception. The overall cost is least at the Enhanced Level, and highest at the Improved, the Minimum Level being between these two costs but closer to the cost of the Enhanced Level. In Ore Body 1 the trend is reversed, resulting in the Minimum Level being 2 cents per barrel cheaper than the Enhanced Level. This reversal in trend is primarily due to the cost trends observed for tailings disposal (Cost Centre 4) and for reclamation (Cost Centre 5). At the 120,000 BPCD mine size, the cost of tailings disposal has a considerable spread, the Enhanced being almost half that at the Minimum Level.

At the Enhanced Level major amounts of reclamation are done early in the life of the mine. The major portion of the cost is in muskeg mining, dewatering and blending facilities and, consequently, on a discounted basis the costs at the Enhanced Level become higher.

The reclamation costs at the various levels and mine sizes are compared to cost of reclamation at the Minimum Level in the following chart:

CONTRIBUTION TO TOTAL COST BY COST CENTRE

TABLE 10.7-2

COST CENTRE	ORE BODY NO. 2, 120,000 BPCD						ORE BODY NO. 4 60,000 BPCD				ORE BODY NO. 1 240,000 BPCD					
	DRAGLINE SCHEME			B.W.E. SCHEME			DRAGLINE SCHEME		B.W.E. SCHEME		B.W.E. SCHEME		AVERAGE BY LEVEL			OVERALL AVERAGE
	Minimum	Improved	Enhanced	Minimum	Improved	Enhanced	Minimum	Improved	Minimum	Improved	Minimum	Enhanced	Minimum	Improved	Enhanced	
COST CENTRE 1: Civil Construction-Type Activities	3%	2%	3%	2%	2%	3%	3%	2%	3%	2%	3%	3%	3%	2%	3%	3%
COST CENTRE 2: Removal of Organic Materials & Soils	2%	2%	2%	3%	2%	3%	3%	2%	3%	2%	3%	2%	3%	2%	2%	2%
COST CENTRE 3: Overburden, Reject, Oil Sands Handling	63%	44%	63%	54%	37%	56%	56%	43%	56%	43%	61%	62%	58%	42%	62%	53%
COST CENTRE 4: Tailings Disposal	21%	42%	15%	27%	46%	14%	20%	35%	20%	35%	22%	18%	22%	40%	15%	26%
COST CENTRE 5: Establishment of Ultimate Land Use Resources	2%	4%	8%	2%	5%	12%	2%	6%	2%	6%	3%	7%	2%	5%	9%	6%
COST CENTRE 6: Supervision, Technical Services	9%	6%	9%	12%	8%	12%	16%	12%	16%	12%	8%	8%	12%	9%	9%	10%
TOTAL	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%

NOTE: Refer to Chapter 6 for Cost Centre description.

PRESENT VALUE ANALYSIS OF TOTAL COSTS ($/bbl.)

TABLE 10.7-3

			ORE BODY NO. 2, 120,000 BPCD						ORE BODY NO. 4, 60,000 BPCD				ORE BODY NO. 1, 240,000 BPCD	
	Present Value %	COST	DRAGLINE SCHEME 1,029,329,000 bbl			B.W.E. SCHEME 982,643,000 bbl			DRAGLINE SCHEME 532,700,000 bbl		B.W.E. SCHEME 527,700,000 bbl		B.W.E. SCHEME 2,138,103,000 bbl	
			Minimum	Improved	Enhanced	Minimum	Improved	Enhanced	Minimum	Improved	Minimum	Improved	Minimum	Enhanced
COST CENTRE 1	0%	OPERATING	0.0342	0.0342	0.0377	0.0272	0.0272	0.0310	0.0448	0.0450	0.0432	0.0433	0.0279	0.0332
		CAPITAL	0.0246	0.0246	0.0322	0.0164	0.0164	0.0221	0.0232	0.0235	0.0224	0.0224	0.0177	0.0247
		TOTAL COST	0.0588	0.0588	0.0699	0.0436	0.0436	0.0530	0.0681	0.0685	0.0656	0.0657	0.0455	0.0580
	10%	OPERATING	0.0088	0.0088	0.0097	0.0069	0.0069	0.0079	0.0113	0.0114	0.0109	0.0109	0.0068	0.0081
		CAPITAL	0.0190	0.0190	0.0250	0.0126	0.0126	0.0170	0.0177	0.0179	0.0170	0.0170	0.0131	0.0182
		TOTAL COST	0.0279	0.0279	0.0348	0.0195	0.0195	0.0249	0.0290	0.0293	0.0280	0.0280	0.0199	0.0263
COST CENTRE 2	0%	OPERATING	0.0401	0.0385	0.0385	0.0392	0.0392	0.0343	0.0500	0.0492	0.0505	0.0498	0.0437	0.0354
		CAPITAL	0.0073	0.0070	0.0070	0.0072	0.0071	0.0062	0.0091	0.0090	0.0092	0.0091	0.0080	0.0064
		TOTAL COST	0.0475	0.0455	0.0455	0.0464	0.0464	0.0405	0.0590	0.0582	0.0596	0.0589	0.0517	0.0418
	10%	OPERATING	0.0188	0.0185	0.0185	0.0180	0.0187	0.0161	0.0242	0.0236	0.0244	0.0239	0.0170	0.0144
		CAPITAL	0.0034	0.0033	0.0033	0.0033	0.0034	0.0029	0.0044	0.0043	0.0044	0.0043	0.0031	0.0026
		TOTAL COST	0.0222	0.0218	0.0218	0.0213	0.0222	0.0190	0.0285	0.0279	0.0288	0.0282	0.0201	0.0169
COST CENTRE 3	0%	OPERATING	0.9379	0.9276	0.9276	0.6300	0.6300	0.6300	0.9109	0.9153	0.9510	0.9574	0.7982	0.7982
		CAPITAL	0.4017	0.4158	0.4158	0.2726	0.2726	0.2725	0.3536	0.3529	0.3173	0.3233	0.3163	0.3163
		TOTAL COST	1.3396	1.3434	1.3434	0.9026	0.9026	0.9026	1.2645	1.2682	1.2683	1.2807	1.1145	1.1145
	10%	OPERATING	0.2211	0.2179	0.2179	0.1436	0.1436	0.1436	0.2186	0.2196	0.2327	0.2338	0.1855	0.1855
		CAPITAL	0.2274	0.2285	0.2285	0.1519	0.1519	0.1519	0.2284	0.2278	0.2049	0.2057	0.1904	0.1904
		TOTAL COST	0.4485	0.4464	0.4464	0.2955	0.2955	0.2955	0.4470	0.4474	0.4376	0.4395	0.3759	0.3759
COST CENTRE 4	0%	OPERATING	0.3615	1.1437	0.2036	0.3697	0.9621	0.1398	0.3565	0.9086	0.3521	0.9190	0.3422	0.2040
		CAPITAL	0.0850	0.1508	0.1035	0.0771	0.1489	0.0812	0.0944	0.1343	0.0923	0.1355	0.0613	0.1217
		TOTAL COST	0.4465	1.2945	0.3072	0.4468	1.1110	0.2210	0.4509	1.0428	0.4444	1.0545	0.4034	0.3257
	10%	OPERATING	0.1054	0.2541	0.0392	0.1054	0.1554	0.0304	0.0844	0.1645	0.0842	0.1668	0.0803	0.0403
		CAPITAL	0.0440	0.0679	0.0496	0.0402	0.0509	0.0419	0.0420	0.0579	0.0420	0.0585	0.0300	0.0630
		TOTAL COST	0.1494	0.3221	0.0888	0.1456	0.2063	0.0723	0.1265	0.2223	0.1262	0.2253	0.1104	0.1033
COST CENTRE 5	0%	OPERATING	0.0453	0.1085	0.1242	0.0226	0.0933	0.1419	0.0381	0.1422	0.0396	0.1462	0.0501	0.0845
		CAPITAL	0.0047	0.0188	0.0470	0.0027	0.0280	0.0512	0.0032	0.0376	0.0032	0.0384	0.0055	0.0349
		TOTAL COST	0.0501	0.1273	0.1712	0.0253	0.1214	0.1931	0.0412	0.1799	0.0429	0.1846	0.0556	0.1194
	10%	OPERATING	0.0041	0.0088	0.0212	0.0025	0.0097	0.0265	0.0037	0.0146	0.0039	0.0155	0.0038	0.0119
		CAPITAL	0.0005	0.0016	0.0236	0.0003	0.0031	0.0275	0.0003	0.0040	0.0004	0.0043	0.0004	0.0137
		TOTAL COST	0.0045	0.0104	0.0448	0.0028	0.0128	0.0540	0.0040	0.0187	0.0043	0.0197	0.0041	0.0257
COST CENTRE 6	0%	OPERATING	0.1870	0.1870	0.1904	0.1959	0.1959	0.1995	0.3531	0.3531	0.3568	0.3568	0.1499	0.1531
		CAPITAL	--	--	--	--	--	--	--	--	--	--	--	--
		TOTAL COST	0.1870	0.1870	0.1904	0.1959	0.1959	0.1995	0.3531	0.3531	0.3568	0.3568	0.1499	0.1531
	10%	OPERATING	0.0462	0.0462	0.0470	0.0484	0.0484	0.0493	0.0874	0.0874	0.0883	0.0883	0.0364	0.0372
		CAPITAL	--	--	--	--	--	--	--	--	--	--	--	--
		TOTAL COST	0.0462	0.0462	0.0470	0.0484	0.0484	0.0493	0.0874	0.0874	0.0883	0.0883	0.0364	0.0372
TOTAL COSTS	0%	OPERATING	1.6061	2.4396	1.5221	1.2847	1.9478	1.1764	1.7534	2.4135	1.7932	2.4725	1.4119	1.3085
		CAPITAL	0.5233	0.6169	0.6055	0.3759	0.4731	0.4333	0.4835	0.5574	0.4445	0.5288	0.4088	0.5041
		TOTAL COSTS	2.1294	3.0565	2.1276	1.6606	2.4209	1.6097	2.2369	2.9709	2.2377	3.0012	1.8207	1.8126
	10%	OPERATING	0.4045	0.5543	0.3535	0.3248	0.3829	0.2738	0.4296	0.5211	0.4445	0.5393	0.3298	0.2974
		CAPITAL	0.2943	0.3204	0.3301	0.2083	0.2217	0.2412	0.2928	0.3120	0.2687	0.2898	0.2370	0.2880
		TOTAL COSTS	0.6988	0.8747	0.6836	0.5331	0.6047	0.5150	0.7224	0.8331	0.7133	0.8291	0.5668	0.5853

NOTE: Refer to Chapter 6 for Cost Centre description.

RECLAMATION COSTS AS MULTIPLE OF MINIMUM LEVEL

Before discounting	Minimum	Improved	Enhanced
60,000 BPCD	1	4.5	-
120,000 BPCD	1	3 to 7	5 to 8
240,000 BPCD	1	-	2
After discounting			
60,000 BPCD	1	5	-
120,000 BPCD	1	4 to 7	9 to 19
240,000 BPCD	1	-	6

It can be concluded that the cost of reclamation at the Enhanced Level compared to that at the Minimum Level is an order or two of magnitude greater. At the 120,000 BPCD size, the increase is up to 8-fold before discounting and up to 19-fold after discounting. For the 240,000 BPCD, the increase was 2-fold and 6-fold, before and after discounting, respectively. The spread between the Minimum and Improved Levels is much less, being at most 7 times greater, both before and after discounting. Nonetheless, the absolute cost compared to the costs incurred in the course of operating an oilsands project is extremely small.

COST CENTRE TRENDS

COST CENTRE 1: Civil Construction-Type Activities

1.1 Mine Power Distribution and Control

The cost of Mine Power Distribution at the Enhanced Level increased over the Minimum and Improved Levels by approximately 1 cent per barrel. Extra spreaders and conveyors utilized at the Enhanced Level lead to greater power requirements.

1.2 Buildings

Trends of building costs show economy of scale. The range, from 0.0078 to 0.0182 $/bbl, an increase of 2.3 times, reflects increased efficiency with size of operation in oil sands mining. The costs in this item are tied to the manpower requirements costed in Cost Centre 6.0.

COST CENTRE 2: Removal of Organic Materials and Soil

Removal of organic materials and soils reflects design constraints on the mine plan with regards to levels of reclamation, and to site specific conditions at ore bodies such as volumes of muskeg, and dimensions of mines.

Little change in cost is seen between the muskeg removal activities at the Minimum and Improved Levels. However, the costs at the Enhanced Level are about 1 cent per barrel less. The main difference between the levels of reclamation is muskeg hauling. At the Minimum and Improved Levels, muskeg hauling allows for more handling to meet reclamation requirements. At the Enhanced Level, in comparison, only the shortest and most convenient muskeg disposal route is chosen, i.e. muskeg is moved out of the way of the mining operation only. A unique situation exists in the 120,000 BPCD Minimum Level bucket wheel plan, where some direct transfer of muskeg over long haul distances is included so as to avoid the necessity of rehandling the muskeg.

COST CENTRE 3: Overburden, Reject, Oil Sands Handling

3.1 Overburden BWE

Economy of scale in the operation of bucket wheel excavators removing overburden is evident here. With respect to dragline mining schemes only, overburden BWE in Ore Body 2 rates at 2,900 bank m^3/hr., while Ore Body 4 at 1,780 bank m^3/hr. Ore body 4 has an overburden BWE capacity of 61% of that of Ore Body 2 but at the same time the mine produces only 52% as much synthetic crude. This is due to considerably thicker overburden in Ore Body 4 than in Ore Body 2.

3.2 Oil Sands, Draglines and Hoppers

Operating cost per dragline and hopper in Ore Body 4 is 84% of the operating cost in Ore Body 2. Production of crude in Ore Body 4 is 3.5% larger per dragline than in Ore Body 2. The capital cost in Ore Body 4 is 98% of that in Ore Body 2.

3.3 BWE (Overburden and Oil Sands)

Economy of scale exists with respect to the BWE's employed. A comparison of BWE's in the various bucket wheel mining schemes follows:

Ore Body 1	Largest BWE's; least expensive per barrel of crude \$0.1356/bbl, or \$0.2076/bank m^3 of material mined.
Ore Body 2	Medium BWE's; \$0.3209/bbl, or \$0.2285/bank m^3 of material mined.
Ore Body 4	Smallest BWE's; most expensive per barrel of crude \$0.3828/bbl, or \$0.2685/bank m^3 of material mined.

3.1, 3.2 and 3.3 Overall Prime Excavator Cost Trends

While economy of scale prevails for the BWE's, no significant economy of scale is evident with the draglines in the 100 - 140 m^3 (80 to 110 cubic yard) range. A bigger mine requires more units. If Ore Body 2 were mined with one dragline per bench, some economy of scale would result between Ore Body 2 and 4; however, a 160 m^3 (200 yard) dragline would be required which appears operationally impractical for oil sands mines at this time. It appears that economy of scale may reach its optimum at about 80,000 BPCD in dragline oil sands mine and between 120,000 and 150,000 BPCD in bucket wheel oil sands mine.

In Ore Body 2, the largest portion of the cost difference between sub-centre 3.1 and 3.2 of the dragline plans and cost sub-centre 3.3 of the bucket wheel plan is the higher capital and operating costs of the dragline-hopper system. Capital cost is 73% greater and operating cost 34% higher because of the fact that 5 prime excavators are employed in the dragline plan, while only 3 are used in the bucket wheel plan. A minor portion of the cost difference (3 cents/barrel) is the result of the 8% larger mass movement of the dragline plan.

In Ore Body 4 the capital costs of the 2 draglines, 2 hoppers, and overburden BWE in the dragline scheme are 48% greater than those of the 3 bucket wheels in the bucket wheel scheme, but the operating costs differ only by 3% in favour of the bucket wheels.

The comparison of prime excavators alone may not be sufficient to establish an overall trend in mining costs. A very significant portion of the mining costs is accounted for by the conveyors required (cost sub-centre 3.4) and also by the cost of the spreader (cost centre 3.5) and the cost of miscellaneous equipment (cost centre 3.6).

3.4 Transport (All Conveyors)

Ore Body 2: The difference between the dragline and bucket wheel mine plans is the doubling of many portions of the conveyor system in the dragline schemes because of the greater number of excavating machines utilized. In general, slewing systems as used in the bucket wheel schemes require fewer conveyors, and the result is lower conveyor costs. Slewing of draglines in Ore Body 2 appears impractical.

Ore Body 4: The dragline plan has less conveyor transport cost than the B.W.E. scheme since the lower bench backcasts centre reject onto the pit floor and smaller conveyors are employed. The smaller size of the dragline conveyors is the result of a surge buffering effect of the dragline hopper on instantaneous production. Compared to Ore Body 2, draglines have the advantage of reducing the number of machines and slewing, which effectively reduces conveyor lengths. Bucket wheel mine conveyor costs increase over those of Ore Body 2 because of negative economy of scale.

Ore Body 1: The conveyor system required (BWE only) has unit costs almost as high as those of Ore Body 4. A maximum amount of material must be handled per barrel of crude in this mine, and thus the widest and also longest conveyors are required. In addition, extra conveyors on the dumping side push the unit cost further upward.

3.5 Placement (Spreaders)

The spreader costs in the BWE mining scheme are higher because all centre reject must be handled, as opposed to the dragline scheme, where bottom bench centre reject is backcast directly onto the pit floor. In Ore Body 1, extra spreaders are necessary to give the mine more flexibility and avoid 3000 mm dump conveyors. Economy of scale with respect to the spreaders otherwise prevails.

3.6 Miscellaneous Equipment

The dragline plan for Ore Body 2 uses more miscellaneous equipment than the BWE plan because of the larger number of prime excavators and conveyors installed. Additional miscellaneous costs occur in Ore Body 1 BWE compared to Ore Body 2 BWE because of the extra spreaders and conveyors utilized.

COST CENTRE 4: Tailings Disposal

4.1 Area Drainage

All costs are dependent on the depth and volume of muskeg in areas where the dykes are to be constructed. The costs are site-specific and therefore, no trends are evident.

4.2 Clearing

Clearing is proportional to the area utilized by the tailings pond and, therefore, is dependent on the tailings disposal concept utilized. Ore Body 2 has the smallest area of out-of-pit tailings pond per barrel and therefore the lowest clearing costs.

4.3 Construction of Starter Dams and Overburden Dams

The construction of starter dykes is largely site-specific to the mine being developed. At the Minimum Level, the most attractive cost is possible in Ore Body 1 due to very favourable topographical influences. Ore Body 4 also has some of the same advantages but has the least favourable pit shape for in-pit starter dam construction. The least attractive dams are required in Ore Body 2. This is primarily due to the large volume starter dyke of the out-of-pit tailings pond.

In-pit starter and overburden dykes should be located to take advantage of the shape of the pit. Thus, a dyke in a "neck" of the pit would be most attractive. In the Improved Level dragline scheme of Ore Body 2, a high cost is incurred to construct the overburden dyke required for the sludge pond, since the dyke is located in the widest portion of the pit.

A comparison between the starter dyke and overburden dyke construction costs at the Minimum and Improved Level bucket wheel schemes in Ore Body 2 shows that the costs are similar. In these bucket wheel mine plans, the location and shape of the dykes are similar and the costs vary primarily due to the relative quantities and haul distances utilized. The

Improved Level has a higher unit cost but this is countered by lower quantities, and thus maintains a cost similar to that of the Minimum Level.

4.4 Piping of Wet Tailings or Conveying of Dry Tailings

The distances over which the tailings slurry is pumped is the major factor in this cost item. The pumping distance is influenced by the physical layout of the tailings pond. The quantities pumped also vary. For example, in Ore Body 4, at the Minimum Level of Reclamation, some tailings sand is pumped into the in-pit sludge pond resulting in somewhat cheaper disposal (shorter pumping distance). Likewise, differing combinations of pumping distance, elevation head and tailings quantities account for the differences between the Minimum and Improved Level pumping costs.

At the Enhanced Level, a portion of the dry tailings conveying and transport costs have been considered to be equivalent to the wet tailings disposal costs. The cost for conveying dry tailings is obtained by subtracting the conveying cost of the Improved Level Cost Sub-centre 3.4 from that determined at the Enhanced Level. Dry tailings conveying cost is proportional to conveying distance. Consequently, Ore Body 1 shows a higher cost than that in Ore Body 2 due to greater conveying distances.

4.5 Tailings Sand Placement into Dyke

Ore Body 1 shows the advantage of a favourable pit shape for tailings dyke construction. Also evident is the economy of the out-of-pit tailings pond dyke, where the ratio of dyke volume to volume stored in the pond is most advantageous. Approximately 5% of the total out-of-pit tailings pond volume is compacted tailings sand in the dykes. Ore Body 4 shows high costs which reflect the effect of a long in-pit tailings dyke; in addition, its out-of-pit tailings pond dyke is approximately 10% of the total out-of-pit pond volume. Ore Body 2 has a medium cost as a result of averaging a costly out-of-pit pond dyke and inexpensive in-pit ponds. The out-of-pit pond dyke contains approximately 16% of the total out-of-pit pond volume.

4.6 Tailings Overboarding and Sanding-in or Placement of Dry Tailings

The cost for wet tailings was related to volume overboarded, which remains somewhat proportional to the plant feed processed. Thus, the cost per barrel remains uniform throughout. In dry tailings placement (Enhanced Level only), an extra spreader in Ore Body 1 drives this unit cost up compared to same item in Ore Body 2.

4.7 Recycling of Tailings Water

Recycling of tailings water from active tailings ponds is dependent upon the pipe lengths and the number of barges required. The BWE scheme has no sludge pond at the Minimum Level, and consequently does not require the use of an extra barge returning water from the sludge pond. This extra barge is included in the dragline scheme at the Minimum Level. Ore Body 4 at the Minimum Level has a small water recycle barge situated on the sludge pond in addition to the water recycle barge in the active tailings pond. At the Improved Level, the sludge pond consists of thick sludge assumed to have no recycleable water on top.

4.8 Rehandling of Tailings Sludge

Sludge rehandle is applicable only to the Minimum Level plans as sludge pumping for the Improved Level plans is included in Cost Sub-centre 4.9. There is no cost shown for the Minimum Level bucket wheel plans for Ore Body 2, since the tailings disposal concept does not rehandle sludge. The rehandling of tailings sludge is dependent on the length of pipes, number of barges installed and pumping head. In Ore Body 4, sludge pumping is cheaper than in Ore Body 1 and 2 because most of the sludge is pumped downhill. The length of the pipe used in Ore Bodies 1 and 2 is also greater compared to that used in Ore Body 4.

4.9 Sludge Treatment

Sludge treatment costs in the Ore Body 2 dragline and bucket wheel schemes at the Improved Level are approximately $0.68 per barrel, mainly due to major costs for flocculants. Amounts of flocculants required are related to the volume of sludge to be treated. The associated sludge pumping costs are slightly less in Ore Body 4 (approximately \$0.61 per barrel), because a smaller volume of sludge (per barrel of crude) is produced from a better quality plant feed in Ore Body 4 (0.312 m^3 sludge/bbl of crude) compared to Ore Body 2 (0.377 m^3 sludge/bbl of crude).

4.10 Power Distribution

Generally, this cost item is primarily dependent on the tailings disposal concept used and the overall shape of the mine. The dragline scheme must supply power to barges on the tailings ponds which repump sludge. No sludge rehandle is being undertaken in the bucket wheel scheme at the Minimum Level. At the Improved Level in Ore Body 2, the power distribution costs are the same in the dragline and bucket wheel schemes.

4.11 Oversize Reject Disposal

In the dragline scheme of Ore Body 2, a separate oversize reject pile is used and no oversize reject is transported back into the pit. In all other ore bodies and mining schemes, oversize reject disposal cost depends on the arrangement of tailings ponds and backfill areas into which it is trucked.

4.12 Oversize Reject Disposal Road Construction

The comments of Cost Sub-centre 4.11 also apply for this cost item.

COST CENTRE 5: Establishment of Ultimate Land Use Resources

5.1 Muskeg Rehandle Loading

The primary difference between the Minimum and Improved Levels is the quantity of muskeg required for prepared soil manufacture. Prepared soil volume depends on the thickness (see reclamation criteria) and the area to be reclaimed.

More direct deposition of muskeg in Ore Body 4 at the Minimum Level is employed, thus reducing the relative costs of muskeg rehandle between Minimum and Improved Levels. It should be noted that muskeg which is taken directly from stripping operations onto the tailings pond dyke for reclamation is costed at Cost Centre 2.

Reclaimable surfaces for the bucket wheel scheme in Ore Body 2, at the Minimum Level are the smallest. There are no sanded-in ponds and a greater amount of muskeg from mine stripping is directly transferred to the reclamation site. At the Improved Level, the cost is higher due to greater prepared soil thickness and larger reclaimable area.

In Ore Body 2 dragline schemes, the Improved Level has 5% less reclaimable surface than the Minimum Level, but a greater thickness of prepared soil is required. In addition, a large portion of the muskeg required for reclamation of the out-of-pit tailings pond is costed in Cost Centre 2 as direct deposition.

5.2 Muskeg Rehandle Hauling

The combination of volume and distance determines this cost item. At the Minimum Level, muskeg is hauled from the muskeg dump to the reclamation site. At the Improved Level, the muskeg is hauled into prepared soil blending piles (see Figures 6.2-1 and 6.2-2).

At the Minimum Level in Ore Body 2, the dragline scheme costs exceed the bucket wheel scheme costs because a greater area is to be reclaimed. The muskeg rehandle hauling costs in the Improved Level dragline plan also exceed the costs in the bucket wheel plan, but in this case, both the area and the haul distance are greater.

5.3 Muskeg Rehandle Placement

The comments in sub-centre 5.1 also apply to Muskeg Rehandle Placement. The dragline scheme at the Minimum Level in Ore Body 2 includes extra cost due to second rehandle of muskeg on top of the out-of-pit tailings pond.

5.4 Muskeg Rehandle Road Construction

The roads costed in this sub-centre are shown on Figure 6.2-1 and 6.2-2. Roads on sanded-in tailings ponds are also costed here. As well, trucks hauling rehandled overburden for prepared soil manufacture use this network of roads.

Muskeg rehandle road construction decreases at the Improved Level due to the short distance between the sources of the muskeg and the blend pile.

5.5 Overburden Rehandle Loading

In the Minimum Level schemes, overburden rehandle is not required on waste dumps and backfill areas, as it is already placed during the spreader dumping operation. Overburden is rehandled only for the surfacing of sanded-in ponds or beaches and plantsite areas at the Minimum Level. At the Improved Level, overburden is loaded into trucks for transport to the blending piles. At the Enhanced Level, the cost shown is that of front-end loaders loading overburden onto the conveyor feeding the blending yard, after the top bench BWE has finished stripping overburden. Otherwise, suitable overburden is supplied by the BWE's as part of the mining operation. A significant factor on loading costs is due to the quantities of overburden required at each level: 0.20 m and 0.67 m Minimum and Improved Levels, respectively.

5.6 Overburden Rehandle Hauling

Overburden hauling costs are directly proportional to the volume and distances associated with each plan at the Minimum and Improved Levels. This item is not applicable at the Enhanced Level.

5.7 Overburden Rehandle Placement

Overburden rehandle placement costs are proportional to the costs of sub-centre 5.5. It is not applicable at the Enhanced Level.

5.8 Overburden Rehandle Road Construction

The roads costed in this sub-centre are shown on Figure 6.2-1 and 6.2-2. At times the muskeg haulage roads are also utilized for overburden haulage, and only roads required to connect overburden dumps with the muskeg rehandle road network are costed here.

5.9 Muskeg Mining, Slurry Transport and Dewatering

Economy of scale with mine size is evident, even with the extra pumping distance required in Ore Body 1 as compared to Ore Body 2. The cost is decreased from approximately $0.037 per barrel to $0.025 per barrel.

5.10 Prepared Soil Manufacture

Costs are proportional to the prepared soil manufacture scheme employed:

Minimum Level - basic field spreading of muskeg and overburden with dozers.
Improved Level - dozers scraping blend pile face.
Enhanced Level - blending yard stacker, bucket wheel reclaimer, conveyors, field stacker.

Economy of scale is evident in the Enhanced Level system, as costs decrease from $0.077 in the 120,000 BPCD mine to $0.044 in the 240,000 BPCD mine.

5.11 Prepared Soil Loading, Front-end Loaders and Trucks

Prepared soil loading costs are largely proportional to volume required for reclamation purposes. This item is applicable only for the Improved and Enhanced Levels of Reclamation.

5.12 Prepared Soil Transport, Trucks

Prepared soil transport is proportional to the volume and hauling distance. This item is applicable only to the Improved and Enhanced Levels.

5.13 Prepared Soil Placement

Prepared soil placement is proportional to the volume required. This item is applicable only to the Improved and Enhanced Levels.

5.14 Prepared Soil Road Construction

The roads costed in this Cost Sub-centre are shown on Figures 6.2-1 and 6.2-2. This item is applicable only to the Improved and Enhanced Levels.

5.15 Seed Bed Preparation, Maintenance

Seed bed preparation is proportional to area reclaimed. The bucket wheel mine at the Minimum Level in Ore Body 2 has the smallest reclaimable area. The estimates for this item are very generous, assuming re-fertilization of all reclaimed areas until Year 30, at which time all reclamation activities are assumed to stop. Planting of forest is not costed at any level of reclamation.

If the cost of forest planting in a particular mine plan is of interest, simply multiply yearly area in hectares as shown in sub-centre 5.15 (Volume III) by $4,000/hectare for commercial forest planting or $3,000/hectare for noncommercial forest planting.

The cost range summary (in $/bbl) of forest planting follows:

	COST RANGE ($/bbl)		
	Minimum	Improved	Enhanced
Commercial Forest Planting	.0053-.0153	.0078-.0159	.0077-.0114
Non-commercial Forest Planting	.0040-.0115	.0059-.0120	.0058-.0085

COST CENTRE 6: Supervision, Technical Services

The supervision and technical service staff is costed in this cost centre. Drawing No.'s BR22902-17-00, BR22902-18-00, and BR22902-19-00 are typical of the allocations assumed. If it is desired to redistribute these manpower allocations to Cost Centre 3.0, 4.0 and 5.0 in order to make comparisons which are inclusive of supervisory and technical service staff cost, this can be done simply by proportioning staff and costs. Operating labour is already included with each activity costed as part of the equipment operating costs.

6.1 Equipment Maintenance (Staff only)

Economy of scale prevails with mine size. The costs at the Enhanced Level are slightly higher than at the Minimum and Improved Levels. This Cost Sub-centre includes only staff; hourly maintenance labour is costed in the equipment operating cost.

6.2 Planning (Staff only)

Economy of scale prevails with mine size but not with level of reclamation, since the staff required at each level of reclamation is the same.

6.3 Mining (Staff only)

Economy of scale prevails with mine size. The staff, in total, is expected to change slightly between mines using wet and dry tailings. However, the redistribution occurs primarily from wet tailings disposal to conveying and spreading of dry tailings. No difference is seen between the Minimum and Improved Levels. This Cost Sub-centre includes only staff; equipment operators and other labour are included in operating cost of the equipment.

11.0 RECOMMENDATIONS TOWARDS RECLAMATION GUIDELINES FOR OIL SANDS MINES

Reclamation guidelines cannot be developed to cover all facets of an oil sands development unless there is a complete awareness of all the development possibilities, from the most minute detail to the broadest concept. Functional guidelines are only possible at the conceptual level of a project or at the routine in-pit operational level. For all practical purposes, a type of "no man's land" exists between the conceptual and site-specific levels. The setting of guidelines in this zone must be avoided, since it will stifle the development of technology and encourage operating inefficiency.

Current legislation and guidelines deal with specific problems at the operational level. Pollution control aspects of reclamation activities account for the bulk of guidelines and set, for example, specific and permissible standards for air, water, and land pollution. A standard of protection to be provided for fish, wildlife, forests, unique natural landscapes, and historical monuments can be rather precisely legislated, and should be a part of the normal operating procedures for a mining development. Negotiation of compliance at this level should be limited to only the most unusual circumstances. Enforcement of guidelines in this area must remain with field inspectors, who should be aware of the operational constraints affecting the compliance with any given guidelines. Field inspectors and company officials must work in a supportive manner to bring about the most economic, but still acceptable, solution to localized environmental and reclamation problems.

Guidelines are also required at the conceptual levels, primarily in the form of clearly and precisely stated objectives for the reclamation of oil sands developments. Such objectives must not be confused with routine site-specific requirements already cited, which are regulated and enforced by field inspectors of the various departments of the Government of Alberta. At the conceptual level, overall approval of the development plan should be given. Guidelines for the operational level need not be considered at this level, except where the overall development plan would result in numerous instances of obvious noncompliance with operational guidelines. Departments whose responsibility it is to ensure compliance with guidelines must adapt the operational guidelines to be functional within the broader guidelines developed at the conceptual level.

Over the past years, the various departments of the Province of Alberta have passed legislation and set guidelines. To date, this legislation has not been packaged in a convenient form as legislation and guidelines specific to the development of oil sands. Such a step is necessary in order to remove the confusion that presently exists with regard to the reclamation of oil sands developments.

This report provides useful information for the setting of objectives. The "trade-offs" have been demonstrated by producing development plans for three potential mining areas. Guidelines in the form of objectives must specify which trade-offs are preferred. The prerequisite to specifying objectives is that the constraints imposed by mining and tailings disposal technology be understood for the individual mine as well as well as for the entire surface mineable oil sands area.

The Consultants have concluded that the formulation of satisfactory reclamation guidelines for future development of the Athabasca oil sands is dependent on the adoption of a regional approach; to this end it is recommended that:

a. The surface mineable areas of the Athabasca oil sands be documented. A geologic data base must be developed based on exploratory drilling data. Geological interpretations must be followed by an assessment of mineability and, in turn, lead to the definition of ore bodies.

b. The entire Athabasca oil sands area be assessed to establish potential mine boundaries in terms of current and future mining and extraction economics.

c. Areas suitable for out-of-pit tailings and waste disposal be mapped and the allocation to ore bodies optimized.

d. Preliminary layouts for mines and tailings disposal systems of all known potential ore bodies be integrated into an overall regional development plan. A sequential development plan for oil sands mines (assuming that the most economical deposits will be mined first) must be matched to the expected extraction technology available at the time that a given deposit is likely to be mined.

e. The reclamation potential of all expected mines within the entire surface mineable oil sands area be ascertained and optimized with respect to basic mining economics, ore conservation, and generation of dry reclaimable land surfaces.

f. A regional end land use plan be developed as the first step toward setting guidelines for the ultimate depth of prepared soil. Soil mixtures and depths providing reasonable assurance of long term survival of vegetation, combined with low post-mining maintenance must be specified at least within area specific limits. The mixture and depth of prepared soils must reflect the quality of the materials available for reclamation, and the characteristics of the underlying materials to be reclaimed as well as other factors. Such land use guidelines would be updated periodically, possibly every 5 years.

Guidelines based on the above recommendations would be complementary to the most advanced mining, extraction, and tailings disposal technology available at any given time and, consequently, would vary somewhat within the surface mineable area of the Athabasca oil sands, reflecting the improvements in mining, extraction, and reclamation methods. Although comprehensive in a reclamation sense, the guidelines would also ensure the development of a finite resource to maximum advantage, for operators and the citizens of Alberta. In essence, the development of the Athabasca oil sands deposit would take on an optimal pattern such as is the case in the development of conventional oil reservoirs in Alberta.

Since oil sands mining, extraction, and reclamation technology is largely in the developmental stages, caution must be exercised in setting guidelines. It should, for example, be understood that once a mine begins operation under a specific set of guidelines it may be very difficult to meet the objectives of a revised set of guidelines. There must be a consensus among all regulatory agencies regarding the content and implementation of the guidelines.

12.0 RECOMMENDATIONS TOWARDS FURTHER APPLIED INVESTIGATION OF OIL SANDS MINE DEVELOPMENTS AND RECLAMATION

During the course of this study, it became apparent that there are several areas where further investigation could lead to the more efficient development of the Athabasca oil sands, and to substantial improvements in reclamation of future mines. The recommendations below do not reflect deficiencies in current practice, but rather the type of work that must be done as Alberta begins to depend more on the development of oil sands for oil production. If systematically carried out, the investigations suggested will yield practical solutions contributing to the efficient development of the Athabasca oil sands. The Consultants' recommendations are as follows:

1. Examine more ore bodies to determine if trends exist in reclamation potential other than those already established by the study. The interaction of the shape of the ore body, the mine layout and the tailings disposal concept would be further documented.

2. Study the advantages and disadvantages of raised and lowered wet in-pit tailings ponds and sludge ponds in terms of mining, hydrology, and reclamation constraints.

3. Develop a guideline for soil reclamation considering the characteristics of available materials, underlying materials and the productivity of end land use objectives. Since this guideline would be broad enough to include all combinations of materials and land uses likely to be encountered, operators could use this guideline to develop surface reclamation plans.

4. Study the feasibility of surfacing partially dewatered and treated sludge ponds.

5. Determine the feasibility and estimate the costs of incorporating sludge from mines using wet extraction processes into the dry tailings from mines utilizing dry extraction processes. The potential for the elimination of high risk and intensive-maintenance sludge

ponds as part of an integrated regional oil sands development plan may allow for a rather attractive co-existence of both types of extraction processes.

6. Assess the extent of hydraulically mineable muskeg and determine which ore bodies could possibly take advantage of this resource in order to ascertain the overall importance of the technique to reclamation practices of future oil sands mines.

7. Compare the economies of methods of producing "dry" peat to the economics of hydraulically mined peat. By using "dry" methods, it may be possible to obtain a better quality of peat from within the pit boundary.

8. Study the feasibility of a regionally integrated saline water disposal plan utilizing end-pit lakes.

9. Develop a river diversion scheme and schedule compatible with the regional development plan for oil sands mines. The regional development plan may be influenced by the schedule of stream diversions.

10. Determine, quantitatively, the advantages of regionally integrated mine and tailings disposal deelopments in the Athabasca oil sands region.

11. Develop a regional mining and tailings disposal plan to maximize the creation of dry reclaimable land.

12. Assess the economics of the mining of ore bodies within the surface mineable regional limits to suggest sequences of development based on economic priority, minimization of environmental impacts, and maximization of oil sands recovery.