# AN INTEGRATED DECISION SUPPORT SYSTEM FOR BACKFILL DESIGN

by

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**To the great loves of my life:**

**to my parents**
**and to my fiancee:**
**Caroline Sui**

# ABSTRACT

Backfill design is a multi-level, data/information/expertise intensive procedure involving various professional fields. During the life cycle of mining design and operation, a series of decisions are made based on the knowledge and personal experience. The success of mining design depends largely on knowledge available, and how knowledge is managed and processed. This thesis presents a conceptual backfill design rationale in a systematic approach in terms of basic information needs and data flow. Based on this representation, an integrated decision support system for backfill design is introduced. The underlying system is based on the integration of traditional databases, expert systems, hypermedia, and logical programming concepts consisting of the following components: 1). user interface, which creates the model of tasks and the application environment; 2). hypermedia-based reference manual, which supports non-linear access to backfill design reference manual of various formats; 3). expert systems which solve certain tasks based on heuristic rules; and finally, 4). knowledge base management system which provides an efficient approach to manage and manipulate massive data of previous backfill operations. The integration of these components is achieved through a blackboard architecture. The early test of the prototype system shows that the basic objectives have been achieved to provide fast information access and certain decision makings.

# RÉSUMÉ

La conception des remblais est un procédé intensif de plusieurs niveaux, données/information/expertise, impliquant divers domaines professionels. Durant le cycle de vie de la conception et de l'opération d'une mine, des décisions sont prises en se basant sur la connaissance et l'expérience personnelle. Le succés de la conception des mines dépend largement de la connaissance disponible et comment cette connaissance est traitée et gérée. Ce mémoire présente, d'une manière systématique, la logique derrière la conception de remblais dans les termes des besoins de l'information fondamentalle et de la propagation des données. En se basant sur cette représentation, un système intégré de support de décisions pour la conception des remblais est introduit. Le système mentioné est basé sur l'intégration des bases de données traditionelles, des systèmes experts, hypermedia et les concepts de la programmation logique. Il est composé des élèments suivants: 1) L'interface avec l'utilisateur, qui crée un modèle des tâches et de l'environnement de l'application; 2) Un manuel de référence, basé sur la technologie hypermedia, qui supporte un accés non-lineaire aux manuels de référence de conception des remblais en différents formats; 3) Un système expert qui résoud certaines tâches en se basant sur des règles heuristiques; et finallement, 4) Un système de gestion des connaissances qui fournit une approche efficace pour gérer et manipuler les données sur les opérations antérieures. L'intégration de ces composants est accomplie à travers l'architecture d'un tableau. Les premiers tests du prototype prouvent que les objectifs élémentaires ont été atteints dans le but de fournir un accés rapide à l'information et la prise des décisions.

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# INTRODUCTION

## 1.0. INTRODUCTION

Waste rock from mining operations has been used to fill openings for hundreds of years. However, the use of waste rock or other materials in order to fill openings for support purposes is a relatively recent development necessitated by large scale mining methods, and facilitated by modern milling practices. The development of the flotation process late in the 19th century and early in the 20th century had monumental implications for both mining and milling practices[1]. During the past twenty-five years, or so, three factors:

- Cement addition to fill,
- Environmental pressures, and
- Resource conservation pressures

have had a direct impact on the development of backfill technology. The following professional fields are related to the technology of mine fill in underground mines[2]:

| | |
|---|---|
| • Mining engineering | • Mineral processing |
| • Rock mechanics | • Soil mechanics |
| • Environmental engineering | • Cement technology |
| • Pozzolan chemistry | • Mineral chemistry |
| • Industrial engineering | • Geology |

As a relatively new mining technique, the approach to backfill design and evaluation in practical mining operation is still not beyond the trial and error stage, although some rules and rationales have been established through recent research. The traditional backfill design is a multi-level data/information/knowledge intensive procedure involving various professional fields. During the life cycle of mining design and operation, a series of decisions are made based on the knowledge and personal experience. The success of mining design depends largely on knowledge available and how knowledge is managed and processed. Basically, knowledge exists either explicitly in various sources such as reference manual, literature, research reports etc., or implicitly in some successful designed mining operation and experience. Mining designers need to refer to those standard design manual or other explicitly stated knowledge source, of course. But more often, decisions are made by drawing the analogy from case to case, and personal judgment. Successfully designed mining operations are usually used as strong arguments to support other mining projects. Therefore, mining conditions are never identical in the real world. New problems arise with new mining project starts. Mining designers often found themselves oscillating between design principles and personal judgment when facing crucial decision makings. For instance, conclusions reached by analytical or numerical method are not trusted until certain safety margins are heuristically adjusted. When inconsistency occurs from various knowledge sources, compromised decisions have to be made to respect the influences of each knowledge source.

In the traditional mining design, a large amount of time and money is spent to bring various information together. Even though the information is available, the manual process and access to information base is also very time consuming and inefficient. To simulate this kind of decision making process, this thesis proposes a computer-based integrated decision supporting system for mining design in the context of object-oriented programming, expert system, knowledge base management system, and hypermedia based system. The main efforts are given to backfill design and evaluation. In this framework, explicit knowledge and implicit knowledge are distinguished and processed using different technologies. The cooperation among different knowledge sources is achieved by a blackboard architecture.

## 1.1. OBJECTIVES AND CONTRIBUTIONS

The aim of this research is to systematically investigate the basic information requirements and process of backfill design based on a profound understanding of backfill operation. From the backfill engineering point of view, this research presents a backfill design rationale at conceptual level, which defines basic scope and stepwise backfill design procedure in pre-feasibility study level. From computer system analysis point of view, this research proposes an automated or semi-automated approach to the information storage and process, and therefore provides basic knowledge and information service to backfill designer for various decision makings, and hence improves the efficiency and accuracy of decision making involved in backfill design.

Accordingly, the research objectives of this thesis are:

1. Define the information scope and requirement of backfill design;
2. Investigate the feasible model and related technologies to meet with the requirement based on a computer system;

To achieve these objectives, the following goals have to be met:

1. Identify the application scope by defining standard backfill design rationale at conceptual level and the basic specifications of the system requirements;
2. Investigate feasible models and architecture of a computer system to integrate expert system, knowledge base management system, and hypermedia system as decision support tools for backfill design;
3. Conduct the conceptual modeling of the application environment;
4. Implement the prototype of the designed system.

This research program opened a new area to apply computer technology to mining design. Data intensive application of computer technology is no longer restricted within the traditional business management, employment information management, air flight scheduling etc.. On the other hand, the massive transfer of merging computer technologies mentioned in this thesis to mineral industry provides more efficient tools to solve engineering problems like backfill design. Applications of the proposed technology is not limited to backfill design alone. Other similar data intensive engineering problems are also considered as potential application areas. The research also demonstrates the potential need for further development of computer ability to meet with more complex data intensive engineering applications.

## 1.2. THESIS ORGANIZATION

This thesis is divided into twelve chapters. Chapter 1 states the problems of backfill design and its data intensive feature. Based on the statement, this chapter defines the objectives of the research program and illustrates contributions of the thesis to solve the problems. Chapter 2 presents an overview of backfill design and its recent development. A complete backfill design rationale is outlined with the effort to present the basic information needs involved in backfill design within the framework of context diagram. The analysis follows a top-down approach starting from high level information requirements. Chapters 3 to 7 discus the main techniques of backfill design. At lower level, information requirement and process are represented by data flow diagrams. Chapter 3 defines the following backfill material properties and the main factors that affect these properties along with a brief review of basic techniques:

- Mechanical property;
- Hydraulic property;
- Environmental property;

Chapter 4 illustrates the techniques related to backfill material preparation, which includes various equipment for dewatering operation and some typical applications. Chapter 5 presents basic concepts of fill transportation. The main efforts are given to hydraulic transportation system and the main concerns of design issue. Chapter 6 discusses the backfill placement problem, which includes the stope preparation, backfill placement, bulkhead design and monitoring.

Chapter 7 presents the main results of Quebec backfill operation survey. Based to the survey, the notion of capital cost and operation cost is defined and subclassified to various cost items. In addition, a simple regional cost estimation model is established. The data collected through the survey are compiled and input to the database.

Chapter 8 defines the basic components of integrated decision supporting system backfill design. Basically, the integrated decision supporting system consists of four models:

1) User interface,
2) Expert systems,
3) Knowledge base management system,
4) Hypermedia base system.

Functions of each component are defined at conceptual level. A blackboard architecture to integrate various technologies into a single environment is presented. Also in this chapter, techniques of conceptual modeling are presented.

Chapter 9 demonstrates the application of experts system to mining method selection. A rule-based expert system is implemented using CLIPS expert system development shell, which can either work as a stand alone tool for mining method selection, or support dynamic links within hypermedia base system. A decision tree approach is discussed in details followed by a concrete example.

Chapter 10 demonstrates the hypermedia based system to implement a reference manual for backfill design. The hypermedia system provides a non-linear access to database of various formats. This feature meets with most information needs of backfill design.

Chapter 11 introduces the knowledge base management system and its application to hydraulic transportation system. A concrete example is given to illustrate the basic techniques and how it helps to solve the pump selection problem.

Chapter 12 summarizes important conclusions and discusses future developments.

# CHAPTER 2

# BACKFILL DESIGN RATIONALE

## 2.0. INTRODUCTION

Waste rock from mining operation has been used to provide unit support and to fill openings for hundreds of years. Pillars constructed of rock (sometimes quarried), or concrete, or timber, have been used continuously in mining up to present times[3], [4]. During the twenty-five years, or so, three factors

1. Cement addition to fill;
2. Environmental pressures; and
3. Resource conservation pressures;

have a direct impact on the development of backfill technology. In general when contemplating backfill design, the following aspects should be investigated:

1) Geology of ore deposit, dimensions of ore body, dip, ore grade...;
2) Physical and mechanical properties of ore and surrounding rock mass;
3) Environmental requirements;
4) Fill material resources;
5) Mining method, production capacity and operation schedule;
6) Fill strength requirement analysis;
7) Determination of quantity of fill constituents;
8) Fill preparation system and facilities;
9) Fill placement system and related equipment;
10) Overall economic analysis;

To define an appropriate backfilling method, the following criteria are proposed:

1) Backfill technology should be reliable and guarantee safety and continuity in mining operations.
2) Backfill capacity should be optimized based on technical and economic analysis.
3) Fill preparation and the placement system should be simple, efficient and special attention must be given to the aspects of quality and quantity control.
4) Stope preparation should be minimized. and fill facilities utilized efficiently.

5) The backfill operation must be economic, i.e. the benefit produced by using backfill should be higher than that gained with other mining systems.

In bulk mining with backfill, inter-chamber pillar recovery operation is closely related to stability of surrounding fill, which is governed by stope size and mechanical properties of the host rock. Cemented fill of various solid composition has been employed as an artificial support with considerable popularity in recent years. The most widely used cemented fill consists of classified tailing, or rock, sand and cement. Results of practical studies show that cement content in a fill and its slurry density are essential factors affecting fill stability and economy of backfilling. The uncertainty in fill design, based solely on the theoretical or numerical modeling techniques, without experimental or practical input, may result in fill block failure or excessive consumption of cementing material. In cyclic backfilling, the filled orebody is utilized mainly as a working platform and so, the cement content required of this platform is generally higher than that in delayed backfill. Hence, the fill strength must be properly designed not only for pillar recovery in subsequent mining operations, but also for heavy equipment.

In order to quantify the factors affecting fill stability and to optimize economic effect, it is essential to consider a rationale and practical design approach upon which the operators can effectively manage backfill technology. Basically, the design should consist of determining fill composition and the water needed for fill preparation to produce an acceptable mix having certain physical and mechanical properties i.e. strength and backfill cost estimation. The following sections describe the backfill operation design process starting from backfill mining methods and stope geometry considerations to backfill economy from both the backfill engineering and computerization point of view. The main goal of this rationalization is aimed to develop a conceptual model for a decision supporting system of backfill design based on the detailed analysis on the backfill technology available and practical operation. The ultimate results of the research will establish a solid understanding and framework of backfill design to cooperate various technologies available in computer engineering such as expert system, knowledge base management system, hypermedia system and object orientation etc. At the higher level, context diagram approach is used to specify the basic functions and data flow of underlying system.

## 2.1. THE GENERAL SCENARIO OF BACKFILL DESIGN

In 1989, Scoble, M. et al, presented a backfill design rationale from engineering point of view[5], which defines, in detail, every design step and the related engineering problems.

Based on this designation, the backfill operation design procedure generally proceeds through the following steps:

1. Mining method specification based on geological and environmental conditions.
2. The backfill purpose specification to satisfy the basic criteria of the mining operation induced by certain mining method.
3. Defining of the target properties of fill materials based on the backfill purpose.
4. Defining the operation system to make the backfill material available to meet with the target properties and finally pour to the mining voids, which includes:
   1). backfill material preparation;
   2). backfill material transportation;
   3). backfill material placement;
   4). backfill operation quality control and environmental monitoring. (Information monitored in this step will be feedback to modify the backfill system.
5. Economical evaluation of backfill system.
6. The documentation and implementation of backfill mining operation.

As a backfill engineer and evaluator, the following aspects have to be taken into account as long as the backfill operation is involved:

1. The geological condition of the mine site;
2. The backfill techniques available currently in operation;
3. The historical experiences and commonly accepted rules of backfill design parameters applicable in similar mining conditions;

A large amount of data, information, and options are involved in these considerations. Meanwhile the access to the information is rarely sequential and non-linear cross-reference is common. Therefore a complete storage of the information and efficient access to the information source is the key to the success of the backfill operations design and hence, the economical profit as well[6], [2]

The backfill design rationale is the approach of representing the knowledge of design procedure from both the designer and computer point of view, by which the fundamental design procedure is outlined. Precise and concise representation of design function hierarchy will lead to an efficient communication between computer and mining designer. The backfill design rationale from designer point of view could be summarized as following six steps (Shown in Figure 2-1).

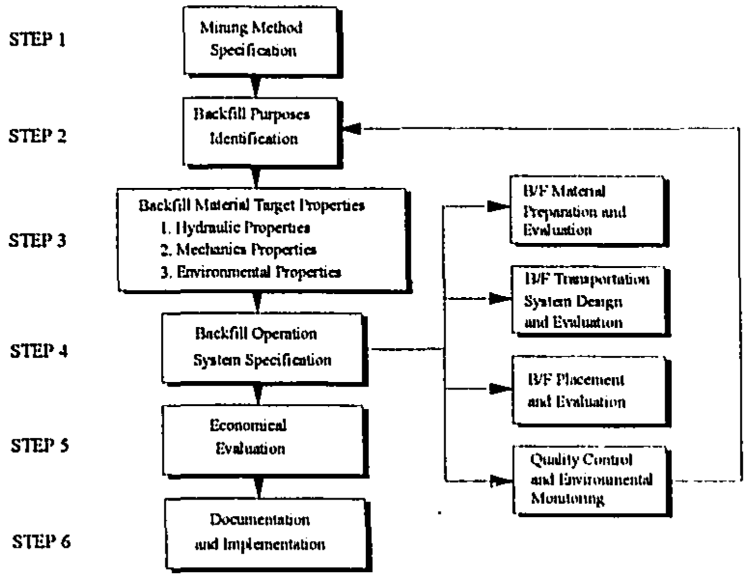



Figure 2-1 The B/F Design Procedure Rationale

From computer system point of view, this procedure can be viewed as an operation on a computer program which consists of an information system combining with various supporting tools and user's interfaces to guide the user to fulfill these six steps on computer. The basic functions related to backfill design are illustrated in Figure 2-2:

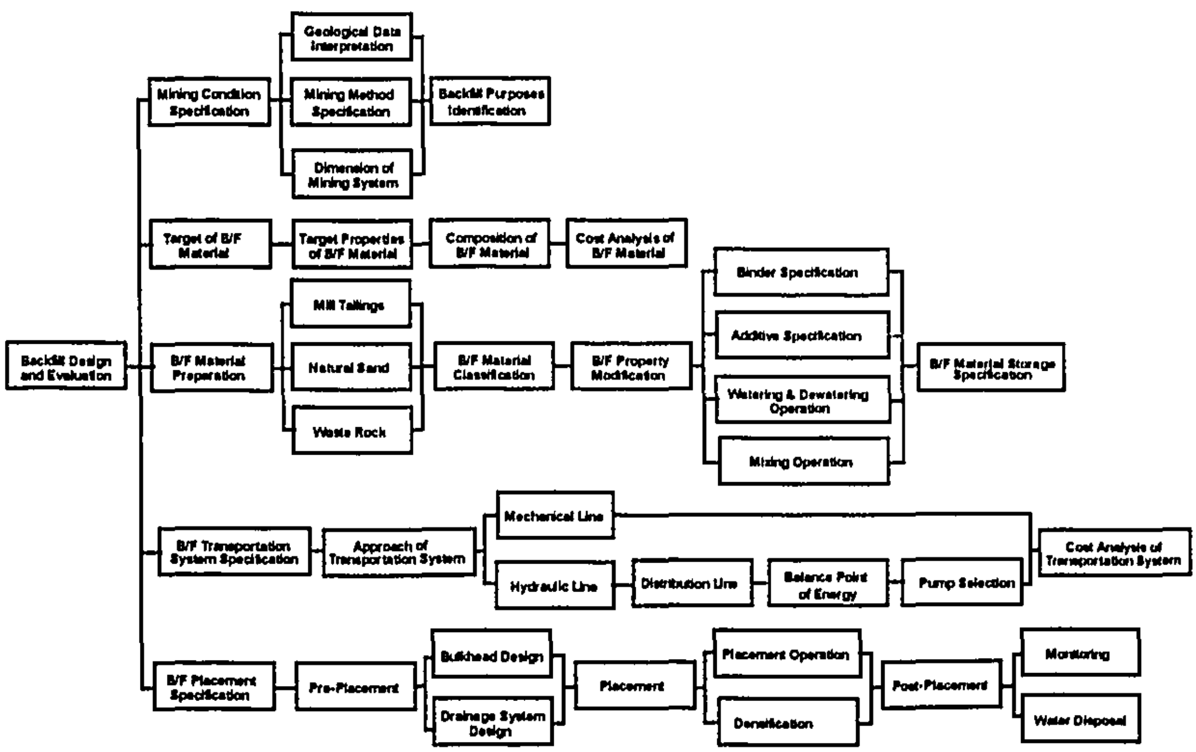



Figure 2-2 Basic functions of the backfill operation

The rationale presented by Scoble, M. focuses on basic tasks of backfill design in general. Less attention has been paid to address the issues of task solving, information needs and data flow during the process. Having recognized backfill design is a data/knowledge intensive process, we now need to further define the design rationale to reflect the basic information needs, data flow and task solving approaches. To be able to have a profound understanding of the system, certain tools are needed to decompose each unit for a systematic specification and rationalization.

## 2.2. TOOLS OF SYSTEM ANALYSIS

To be able to identify the functions and data flow of backfill design in terms of computerization, the system analysis is essential. From the system analysis point of view, it is the application of a system approach to problem solving using computers. The ingredients are system elements, processes and technology. This means that to do system work, one needs to understand the system concepts and how these elements are related and operated as a system, and then design appropriate computer-base systems that will meet the system requirements. It is actually a customized approach to the use of the computer for problem solving. To be able to achieve the goal, the following objectives have to be met:

1. Define outline of the design procedure;
2. Identify the information flow during design process;
3. Define the process description;

Over the past 20 years, a set of tools has been developed to assist the system analyst in understanding how current information systems work and to design new, improved systems. In the following discussion, the specific tools: context diagram, various data flow diagrams and three process descriptions[8] (1. decision tree, 2. decision table and, 3. structured English) will be used to analyze the backfill design system. The selected tools for backfill design system analysis are listed in table 2-1:

Table 2-0 Selected systems analysis tools

| |
|---|
| I. Context diagram - outside users computer system |
| II. Data flow diagrams<br>A Level 0 data flow diagram<br>B Level 1 data flow diagram<br>C. Level 2 data flow diagram |
| III. Process description: A. Decision trees; B. Decision tables, C. Structured English |

These steps form the main conceptual parts of the analysis and general design phase. The analysis and general design phase consists, first, of a graphical model illustrating the flow of data and information between the components of the overall system, including users and any other parties involved. This representation is referred to as the context diagram. Each part of the context diagram is further subdivided by utilizing data flow diagrams. The graphic representation using data flow diagrams helps visualize the flow and processing representing the functions of the system that simulates real backfill design process. Each process is treated at a general level, level zero, and subsequently divided into detailed level processes, levels 1 to N. This representation is in hierarchical format. At the lowest level, the processes are described using one or a combination of techniques known as decision trees, decision table, and structured English. The data flow diagrams show the processes involved in solving a problem. In the subsequent sections, each component of the analysis and general design phase, shown in Table 2-1, is described in more detail along with the rationalization of the backfill design system. These tools provide the systematic methods to represent the knowledge and basic requirement of the application domain. Understanding of the application domain and basic requirement of the backfill design are fundamental for system analysis and rationalization. In these thesis, the main research related to the understanding of backfill design is conducted through literature search and mining survey. The literature review provides us with a good understanding of the backfill design and basic backfill technology. The mining survey, in the other hand, give us the practical understanding in terms of real world operation. Based on the survey, certain globe and regional model can be achieved for computerization to some substantial level. The full report of the literature will be given in the later chapter. For the time being, simply assume that the application domain is well understood.

## 2.3. THE CONTEXT DIAGRAM OF BACKFILL DESIGN

The rationalization process starts from the context diagram representation of the backfill design system, which deals with mainly the basic requirement and operations involved. The context diagram is a useful tool to tie the user domain to the computer system. The interaction between the user and the computer system is a dialog by various links to different tools. One can interpret this stage as a communication protocol between the user needing information and advice, and the computer system as the decision-making medium. The context diagram is a more explicit representation of the functions involved (see Figure 2-3).

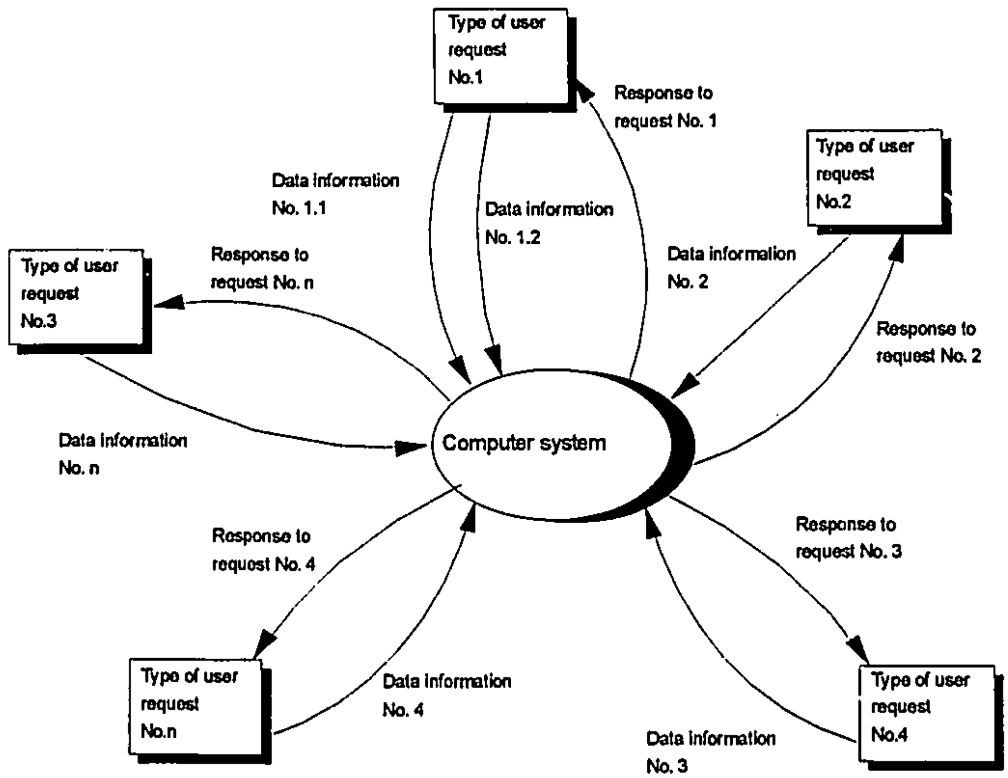



Figure 2-3 General model for a context diagram

The context diagram defines the context, or scope, of the system. Figure 2-3 shows a general model for a context diagram. The diagram shows the flow of data information between the computer-based system and the external entities, in this case the requests posed by the user engaged in a consultation with the system. The convention used in the systems analysis methodology is to depict the system to be designed as single circle in the center of the diagram, and the interacting external entities depicted as squares. The information flow connecting the entities to the system, and vice-versa, must be labeled to show the data content and direction of flow. At this point, the user requests are representative of the general purpose expected from the system. Similarly, information and responses are characteristic of the consultation-advice nature of the system. The data flow labels should maintain a general meaning, only descriptive of the general information required by the system and the expected responses from the system. In this manner, the context diagram establishes, in a global nature, the scope of the system being analyzed and designed. According to this representation, the basic scope and requirement of backfill design process can be represented as the Figure 2-4a:

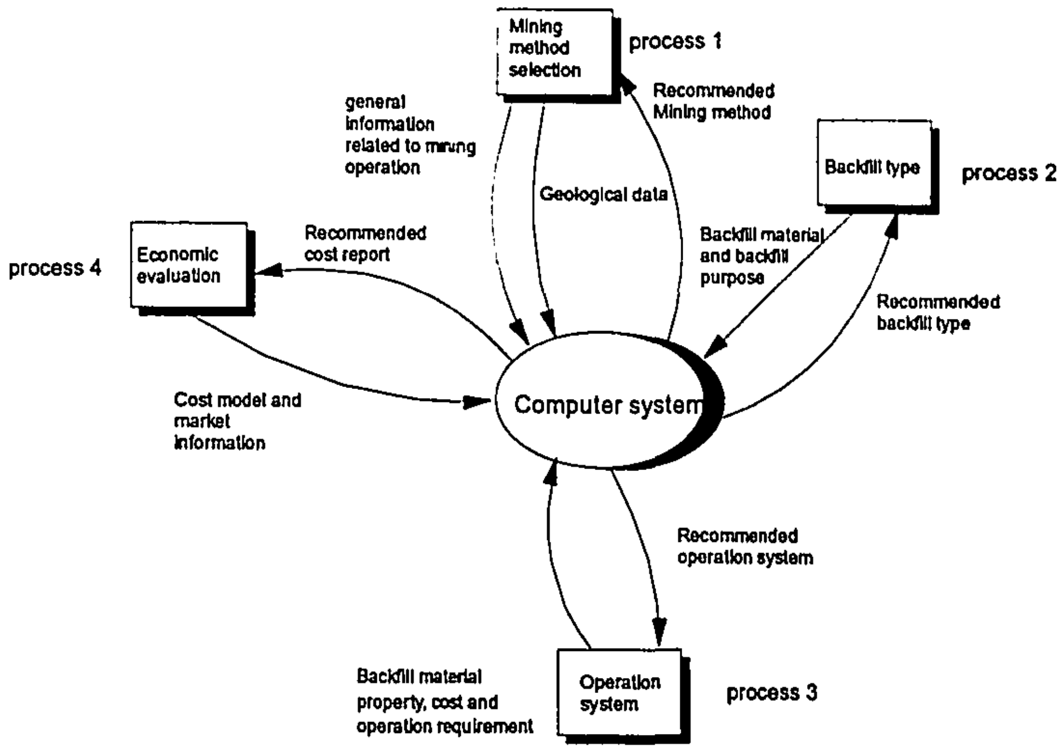



Figure 2-4a Context diagram for backfill design system

In this representation, four basic requirements are defined for the system, among which the operation system can be further defined as the following subsystem (Figure 2-4b):

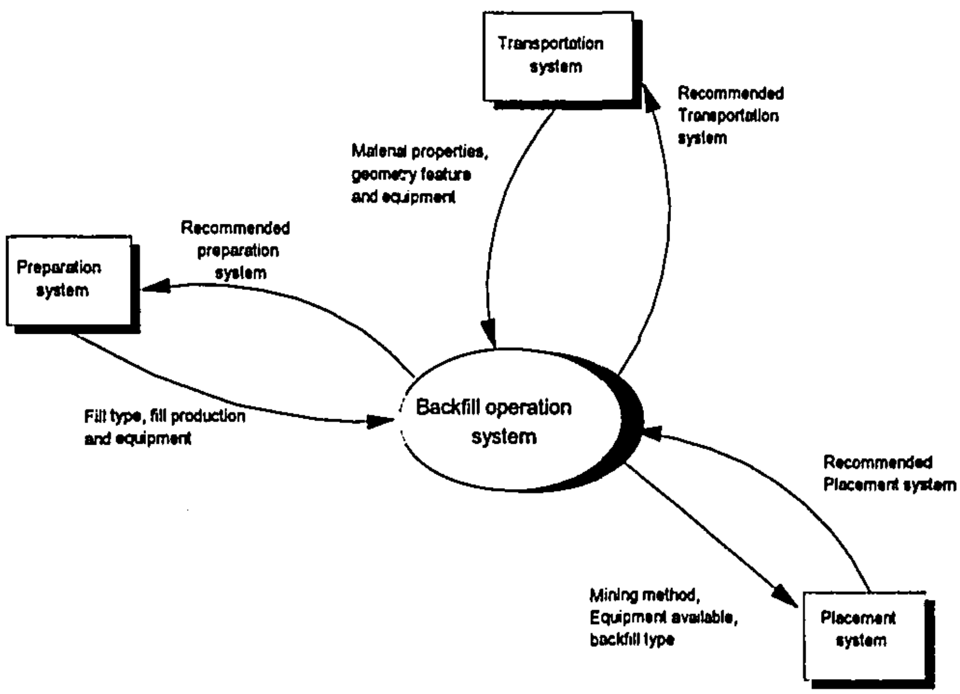



Figure 2-4b The context diagram of backfill operation system

### 2.3.1. The mining method

The specification of mining method is the first decision encountered in the backfill design, which is largely depended on the geological information of the ore deposit and the productivity. In the practical mining activity, most of mining methods related to backfill design are inherited from the previous mining operation and overall mining developing system. Through the literature review, the mining methods related to backfill operation can be classified as table 2-2

Table 2-1 The classification of mining methods [9]:

| **Delayed backfill** | **Cyclic Backfill** |
|---|---|
| Vertical Crater Retreat | Overhand Cut and Fill |
| Sublevel Stoping | Underhand Cut and Fill |
| Shrinkage | AVOCA |
| Blasthole Stoping | Cut and Fill with Post Pillar |
| Room and Pillar | |

Each of these mining method has its application conditions attached. In general, the delayed backfill is applied to the situation where the rock conditions are relative stable, and the mass high-efficient production is required. Stope recovery is normally fulfilled by delayed backfill except that rock conditions are not favorable. On the other hand, the cyclic backfill technique is usually applied to the situation where rock conditions are relatively poor and the low ore loss operation is required. Examples of using cyclic backfill can be easily found in pillar recovery. Information needed to select certain mining technique are listed in the table 2-3.

Table 2-2 The information related to the selection of mining method:

| Mining object | Stope or Pillar |
|---|---|
| Geological condition | 1) Rock type 2) Unit weight of rock<br>3) Mineralogy 4) Grade of mineralogy<br>5) Dip of rock 6) Rock mass quality<br>7) Uniaxial strength 8) Tensile strength<br>9) Elastic modulus 10) Cohesive<br>11) Internal friction angle |
| Mining operation productivity | • Efficiency<br>• Productivity<br>• Market price |
| Primary Stress | • Major principal stress,<br>• Intermediate principal stress,<br>• Minor principal stress; |

Once the certain mining method is selected, the dimension of the mining object (stope or pillar) can also be determined by various approaches developed through rock mechanics researches. Following is the methods that have been used to predict the optimum the dimension of certain mining objects: finite element, boundary element, and other empirical formulas for specific mining cases. It is important to indicate that, in the real mining practice, there is hardly solid consensus on the selection of mining methods and dimension of mining object among the mining experts. Therefore, the previous experience and personal expertise plays important roles. Most decisions made in this process were based simply on experience and analogy drawn from case to case. So a knowledge base system to manage and simulate this feature of decision making is in the center of the decision supporting system which provides the general information of the previous mining experience with a backfill designer.

### 2.3.2. Backfill type

Backfill type is classified according to its constituent and properties. In the practical mining operation, the main backfill materials are waste rock, natural sand, and mill tailings or the mixture. The addition of other binding agents and hydraulic properties will lead to the further classification. Shown on Figure 2-5 is the tree structure of the classification of fill types:

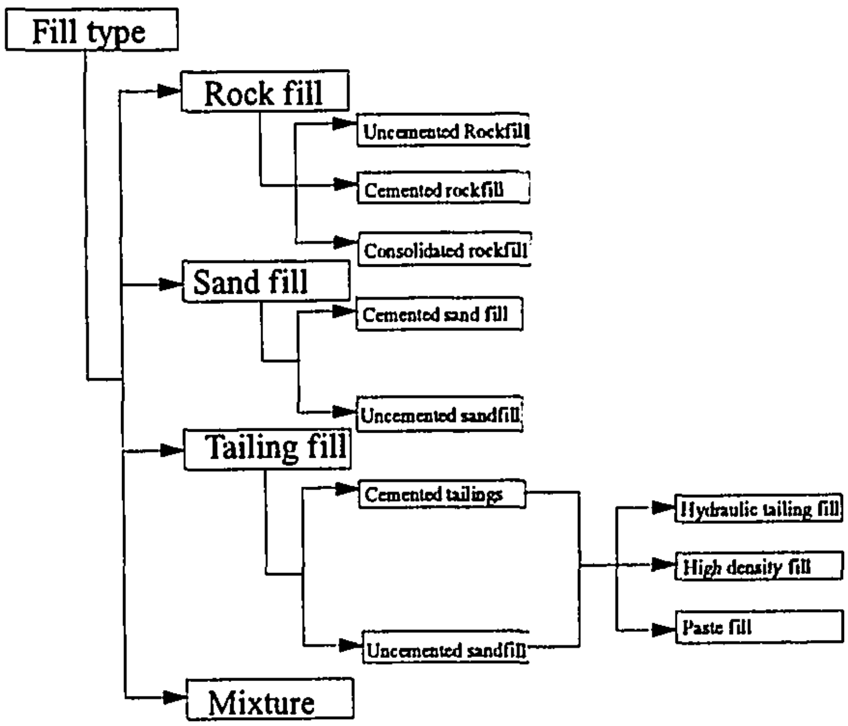



Figure 2-5 The classification of fill types

The parameters influencing the selection of the backfill type are listed in table 2-4:

Table 2-3 The parameters related to backfill type:

| | |
|---|---|
| 1. Backfill material available | Rock, sand, tailings |
| 2. Backfill purpose | 1). Pillar recovery<br>2). Sill pillar recovery<br>3).Working platform<br>4). Ground support<br>5). Waste disposal |
| 3. Strength requirement | High, medium, no requirement |
| 4. Hydraulic properties of fill | Permeability, size distribution etc. |
| 5. Economical consideration | high cement cost for high strength requirement |
| 6. Transportation system | Hydraulic, pneumatic, truck or conveyer |

1). Fill material available for backfill operation: The fill material utilized is classified into three groups: inert material (which is the major part of the fill), the binding agent, and chemical additives. The inert materials commonly used are mill plant tailings, sand or gravel, waste rock, and slag. Binding agents, such as Portland cement or flyash are applied in backfill technology to improve the mechanical properties of fill, i.e. strength. Chemical additives such as flocculent, accelerator, retarder are employed to improve the fill permeability, flowability, and consolidation. Sand from surface alluvial basins is widely used in mine backfill, and its particle size is generally less than 2 mm. Waste rock from mine development or quarry must be crushed to meet transport requirements. The maximum size for pipeline transportation is less than 1/4 the pipe diameter. In the case of hydraulic transportation, this means about 60 mm, whilst aggregates up to 30 cm can be transported by truck or conveyor.

2). The backfill purpose: From the technological point of view, there is a number of possible reasons that backfill operation is used in underground mining. The backfill type is largely dependent on the backfill purpose. In order to optimize the backfill system, it is important to identify the main backfill purpose to be served by the backfill material. The purpose identified will determine what material properties have to be considered in design procedure. These properties are termed as the target properties. The target properties include hydraulic, mechanical and environmental properties. The target properties may be predicted on a site-specific basis such as required mechanical strength to avoid failure of a particular backfill face, or based on commonly accepted standard values such as a required hydraulic percolation rate to permit effective drainage.

3). The strength requirement: In the case of ground support, the strength requirement of the fill material may decide whether cement or other additives should be added to meet the target property.

4). Hydraulic properties: The hydraulic property of fill material influence the hydraulic behavior of fill slurry. The permeability of slurry depends on the size distribution of the fill material, which will in turn determine whether or not full tailings or classified tailings should be used as the fill material, or whether or not paste fill can be employed.

5). The economical consideration: Cemented fill always costs more money than uncemented fill, while waste rockfill costs more than tailing. The principle is always select one which serves the purpose with less cost.

6). The transportation system: The transportation system selected or adopted also plays a role in the determination of the backfill type.

The following is the general characters suggested for hydraulic backfill, paste backfill and rockfill.

**A. Hydraulic Backfill**

Conventional hydraulic fill includes tailing, sand and/or rock, mixed with a binding agent (optional but commonly used) and water. Preparation can take place on surface or underground, and placement is at a pulp density of less than 70% by weight.

**Technical Advantages**

1) Relatively simple to install, operate, maintain and requires minimum technical supervision.
2) All constituents are controlled at fill station which secures the fill quality and mixture density.
3) Desliming technology to increase the percolation rate to 100 mm/hr. can be achieved simply by hydrocyclones.
4) Pumping can normally be avoided by optimizing the pipeline lay-out.
5) Tailing, as mill waste, is readily available in most mines and its utilization can reduce surface waste disposal.

**Technical Disadvantages**

1) Excess water needs to be dewatered from stope and pumped to surface; the permeability is a critical design criteria.
2) Strength of filled body in stope can be reduced by cement marbling (the segregation of cement from the inert fill due to the excess water).
3) Slime from stope drainage requires time (and cost) consuming clean up.
4) Bulkhead dewatering facilities construction and fill curing process can interrupt the mining operation (this is particularly evident with cut and fill mining).

**B. Rockfill**

Waste rock from underground developments or surface quarry is dumped into raise then distributed by truck or conveyor to the stopes. When cemented rockfill is needed, the cement slurry is introduced by separate pipelines and mixed with waste rock prior to placement.

**Technical Advantages**

1) Waste rock is used to backfill underground openings, reducing waste disposal on surface, especially if there is an open pit associated with the mine.
2) Simple preparation system.
3) Relatively high strengths can be attained when waste rock is cemented.
4) Stope dewatering can be avoided.

**Technical Disadvantages**

1) When rock is used as a hydraulic fill, it must be crushed, screened and prepared in the form of a slurry which greatly increases the capital cost.
2) Quarried rock will require crushing, and its transportation and surface production and haulage costs can be significant.
3) The voids in rockfill must be filled by introducing fine material and cementing agent to ensure the rockfill competence.
4) Coning of placed rockfill results in the segregation of coarser material to stope sides and reduces the ability to tight fill stopes.
5) Any tailings produced are only partially utilized and surface disposal must be considered.

**C. Paste fill**

Paste fill has an appearance akin to "toothpaste" and possesses different flow behavior compared to hydraulic fill. Such fills have a higher pulp density, between 75 and 85% by weight, depending on the grain size distribution. They contain total tailings and perhaps include sand or waste rock. The material can be transported from surface and does not require in-situ dewatering. Cement may be added at preparation or immediately prior to placement.

**Technical Advantages**

1) Contains less cementing agent than hydraulic fill under the same strength pre-requisites.
2) Tailings can be almost fully utilized as backfill, so surface disposal can be remarkably reduced.
3) There is little or no excess water to drain off when placed in a stope, hence no need for bulkhead.

**Technical Disadvantages**

1) Requires positive displacement pumps for transportation and the associated high pressure in the pipeline system.
2) Superior dewatering facilities are needed to enable the concentrations required for paste flow to be obtained without loss of fines.
3) Higher level of technology requires more skilled supervision and accurate quality control.
4) Liquefaction studies may be required to ensure safety.
5) Technology is new and unproved in the Canadian Mining industry.

From the economical point of view, the following remarks can be drawn:

1. Rockfill has significant advantages in mines, especially when development waste is utilized as rockfill since it is virtually cost free. Mines with an open pit on site or nearby consider rockfill as an excellent method of waste disposal. Hydraulic fill or paste can be used to fill voids and tight fill within a stope.
2. Hydraulic fill technology has been widely used in mining operations but its application should be limited in modern backfill design. There are opportunities to improve existing hydraulic fill operations by optimizing the solids concentration which can lead to substantial savings in cement consumption, a large proportion of operating costs.
3. Paste fill is the state-of-the-art fill technology and holds tremendous long-term potential in mining. The application of paste fill could significantly reduce the cyclical nature of mining, improve ground conditions, speed up the production and greatly reduce environmental costs. Research and practice have shown the significant advantages of paste fill, although it is still in its infancy. A comprehensive study is necessary to design a proper fill system. Presently, a major loop test facility is required to enable mine backfill to be carefully tested to determine the pressure gradient.

### 2.3.3. Operation system

The backfill operating system is a series of mechanical procedures to make the original material available for backfill to meet with the specific fill properties, fill supply and demand from the mine and finally placement into the mining spaces. It covers the following aspects:

1. Backfill Material Preparation
2. Backfill Transportation Systems
3. Backfill Placement

### 2.3.3.1 Backfill preparation specification

Before being poured into open stopes, the solid material selected for backfill has to be prepared in a fill station to meet with the operating requirements, which include the following aspects:

1. Fill should be prepared in accordance with the mechanical property requirements generated from underground operations, and the cost has to be optimized;
2. Solid concentration of slurry must be optimized for transportation and favorable for backfill performance. The commonly used concentration of slurry range between 70-75% of solid by weight;
3. The content of fine particle in tailing or sand must be adequate to render the resultant fill inherently safe with regard to possible liquefaction and to reach the optimum drainage time after placement;

Fill sources are generally mill tailing, natural sand and waste rock. In addition, some binder agent, such as Portland cement, slag, flyash etc. are also used to modify the performance of the fill to satisfy the designed material requirement. Shown in Figure 2-6 is the general flow sheet of the preparation station:

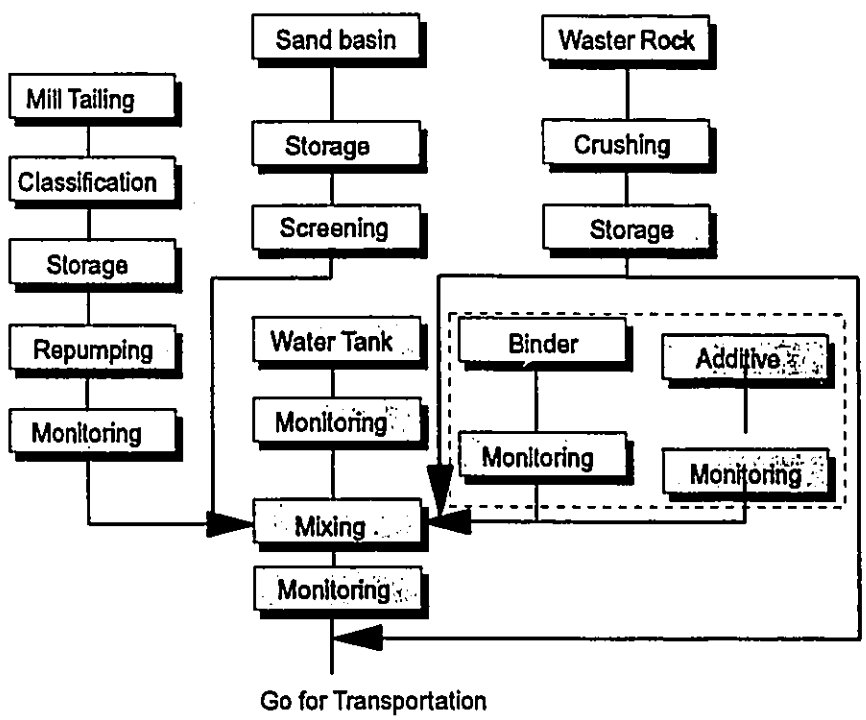



Figure 2-6 The flow sheet of preparation station[10]

In case of the backfill without binder or additive, the related devices enclosed in the dash line are discarded.

Mill tailings used for backfill are usually classified or full tailings which might be mixed latter with binder depending on the backfill purpose. The backfill preparation designation for tailing fill covers the following aspects as shown in table 2-5:

Table 2-5 The specification of mill tailings preparation:

| | |
|---|---|
| 1) Dewatering and desliming | a. Hydrocyclone classification<br>b. Mechanical classifier<br>c. Spinner centrifuge |
| 2) Tailing storage | a. The type of storage silo<br>b. Volume of storage silo<br>c. Repumping system of storage silo |
| 3) Mixing operation | Mixer selection |

The sand preparation goes through the similar procedure as tailings except that, instead of being deslimed, the sand are normally screened to eliminate the coarse particle.
The specification for rock preparation includes the following (Table 2-6).

Table 2-6 specification for rockfill preparation:

| | |
|---|---|
| 1) Rock crushing design | a) Water tank type<br>b) Tank volume<br>c) Water supplying machinery |
| 2) Rock screening design | a) Maximum particle size<br>b) Machinery selection |
| 3) Rock storage design | Same as tailing storage |
| 4) Mixing operation | a) Mixing tank volume<br>b) Mixing type<br>c) Mixing machinery |

Once backfill materials are mixed with binding agent and additive, the operation proceeds to the transportation operation.

### 2.3.3.2 Backfill transportation system specification

The selection of transportation system is crucial to the overall system, which determine how fill material is prepared and placed. Following transport systems have been reported:

1). Hydraulic transportation system;
2). Pneumatic transportation system;
3). Truck transportation system;
4). Conveyor transportation;

So far the hydraulic transportation system has been widely used in backfill operation and large numbers of researches have been conducted and the related results have been published. It is obvious that the rules established for hydraulic transportation system have made it possible that the design procedure can be computerized for feasibility study and economical evaluation. Unfortunately, the researches and publications dealing with other transportation systems are not likely to enable the computerization to any substantial level for feasibility study and economical evaluation. So in this thesis, the main efforts are put on the hydraulic transportation system. More researches and experiments are needed to complete specification for the other transportation system.

The determination of transportation system depends on the following:

- Backfill material;
- Position of ore deposit;
- Topography of mining site;
- Equipment available;
- Economical consideration;

As stated earlier that this rationalization discusses the hydraulic system only, so once the transportation system is selected, the next question to answer is: does it need pump to balance the energy loss, if yes, what kind of pump should it be? The selection of pump depends on the following factors:

- Geometry of ore deposit: Deep of shallow;
- Pipe diameter;
- Size distribution of fill material: coarse or fine;
- Viscosity of material: flowability or not;
- Gravity: heavy or light;
- Flow velocity;
- Concentration of slurry: low, high or paste;

The selection of pump for certain transportation system counts on the energy balance between gravity and resistance pressure loss for certain flow velocity. When potential gravity can not overcome the resistance, pump is needed. The specification of pump includes at least the following aspects:

1). Type of pump: reciprocating pump or centrifugal pump;

2). Capacity of pump: cubic meter per hour;

3). Head: PSI;

4). Cost

#### 2.3.3.3 Backfill placement operation

The backfill placement is the last step towards the completion of entire backfill operation. The literature survey indicates that the approaches used for placement have strong effect on the strength and stiffness of the backfill stowed and play an important role in backfill behaviors. So the placement methods available for each individual mine are carefully tested and compared with each other to reach the best possible benefit both from the economical and technological point of view. The backfill placement design includes the following aspects:

- Pre-placement Operation
- Placement Operation
- Post-placement Operation

1. The designations included in the pre-placement operation vary with the type of stope, which can be broadly classified as follows:

1) Preparation in cyclic stope production prior to backfilling, which requires an initial or basic preparation followed by cyclic adjustments and maintenance.
2) Preparation in non-cyclic or open stopes prior to backfilling, in which the preparation activity is delayed until the stope depletion. It is then commenced as one continuous operation to completely backfill the stope.

In general, stope preparation can be divided into 3-steps, as follows:

a. Installation of stope and backfill dewatering systems
b. Installation of monitoring systems
c. Installation of bulkhead

The drainage system design deals mostly with the hydraulic transportation system with water as fluid.

2. The designations included in the placement operation are the followings:

1) Placement Machinery Specification
2) Densification Method Specification
3) Placement Operating Specification

In many cases, the placement uses the same equipment as in transportation systems, such as the pneumatic transport systems. But still lots of operating systems use different equipment and systems for transportation operation and placement operation respectively. In this case, the placement machinery should be specified specifically. The densification may be involved in placement operation if it is considered as a feasible approach to reach better backfill quality. The followings are possible approaches published so far in literature:

a. Stope Dewatering Prior to Placement
b. Vibrating Compaction
c. Explosive Charge Compaction

The placement operating specification concerns the sequence of placement. The recent development of backfill techniques increasingly implies that there are great potential benefits in the mixture of different type of fill pouring by a specific sequence, which makes the backfill operation more flexible and easier to meet with the backfill material requirement in term of fill resources. For instance, if tailing itself is not sufficient for the total backfill requirement, the crushed rock or alluvial sand may be used as part of backfill material. In this case, the alternatives are either mixing the tailing, cement, sand, or crushed rock together as the fill, or pouring the different fill separately. The specification of the placement sequence includes the following aspects:

a. Steps of Placement
b. Sequence of Placement
c. Fill Amount of Each Steps

3. Finally, the post-placement operation deals mainly with:

1). Backfilling Water Disposal
2). Post-backfill Monitoring

The approaches used to handle the backfill water are normally to incorporate the backfill water system with the ground water disposal system as a whole, and take into account the backfill water in the pump station design.

The post-backfill monitoring includes the followings:

a. Bulkhead identification, type and location.
b. Date, dimensions and details of construction.
c. Dimensions of the stope and bulkhead drift.
d. Degree of saturation of fill in each stope.
e. Height of fill in each stope.
f. Rate of fill placement and its density.
g. Water input and output by stope.
h. Any unusual or abnormal behavior of the bulkhead system.

Shown in Figure 2-7 is the structure of placement operation design.

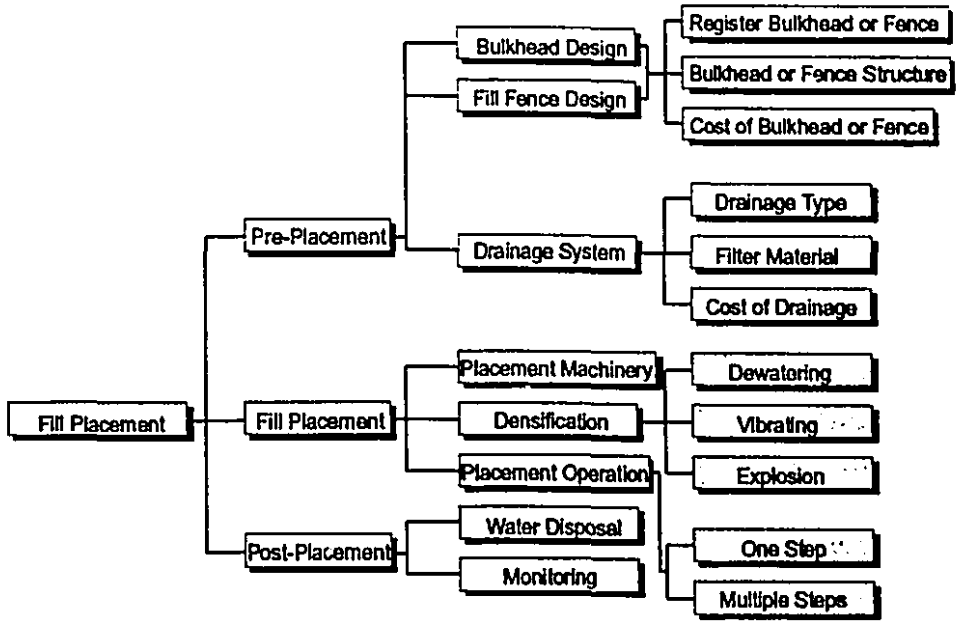



Figure 2-7 The Structure of the Placement Operation

### 2.3.4. Cost estimation

Backfill cost is defined as the direct cost related to backfill operation which includes capital cost and operation cost. Theoretically, the capital costs are the total cost of all installation, equipment and labors and the operation cost are total cost of labor, material and maintenance. But at the feasibility study level, it is impossible to predict every aspect of the capital cost and operation cost in details. Therefore, certain statistic model can be established for cost estimation. One interesting model uses nomogram to illustrate the relationship of different backfill systems. But these nomograms are still too general, some regional models are needed to predict more specific case. The cost model used in this thesis is based on the mining survey conducted within Quebec area. A detailed discussion is presented in the following chapter.

## 2.4. DATA FLOW DIAGRAMS

The data flow diagrams are structured models one step further in detail than the context diagram. The diagrams simplify the logical understanding of the complex system being designed. Each of the external entities shown in Figure 2-**Error! Bookmark not defined.** provides input information and, in turn, receives responses from the system. Through modeling, subsequent changes to the system can be realized without major modifications to the scope of the system.

In applying these models to the design of a computer-based system, the external entity is represented by type of user request. The data flow diagrams mark the interchange of information between the user and the system and, at a lower level, between processes and data stores. Each process indicates the function or operation performed by the system on the incoming data flow. Data stores hold information for later use by processes. Data collectors and routers collect and disperse common information, respectively, to facilitate the information interchange among processes. These simple modeling conventions permit uniformity in understanding the system logic. The Figure 2-4 is expanded into the data flow diagram shown in Figure 2-8a and Figure 2-8b:

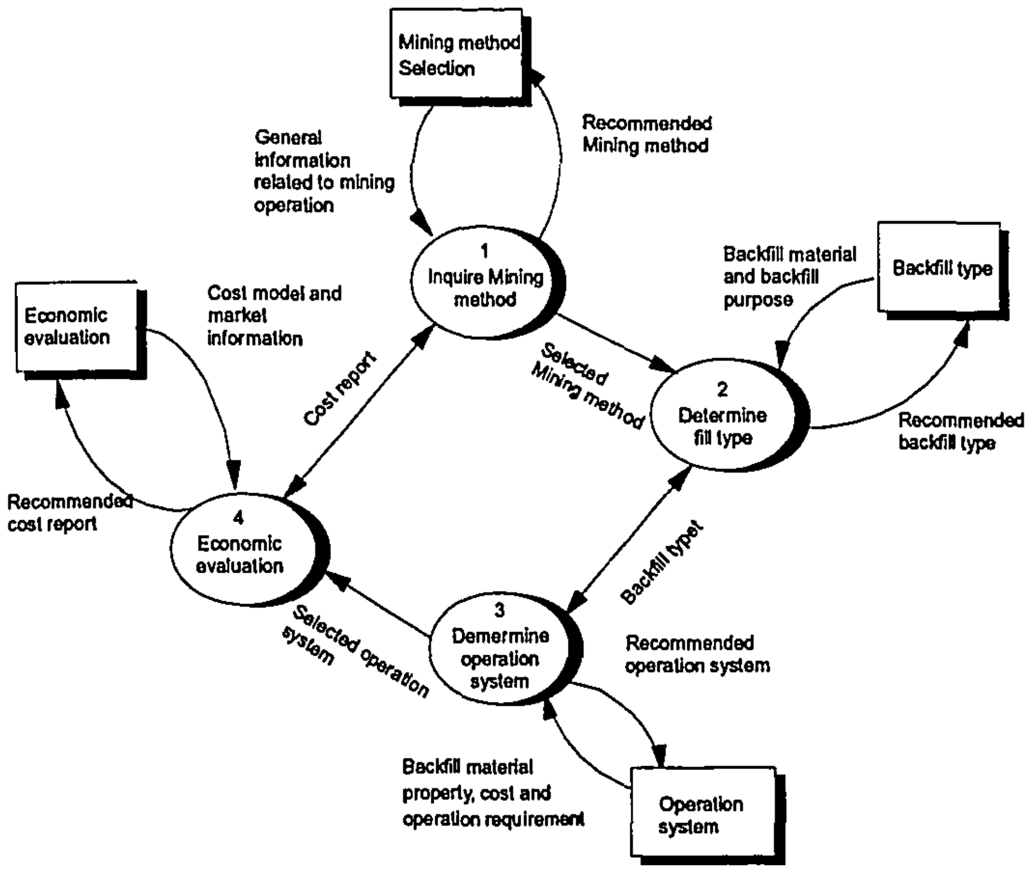



Figure 2-8a The Level-Zero data flow diagram for backfill design system

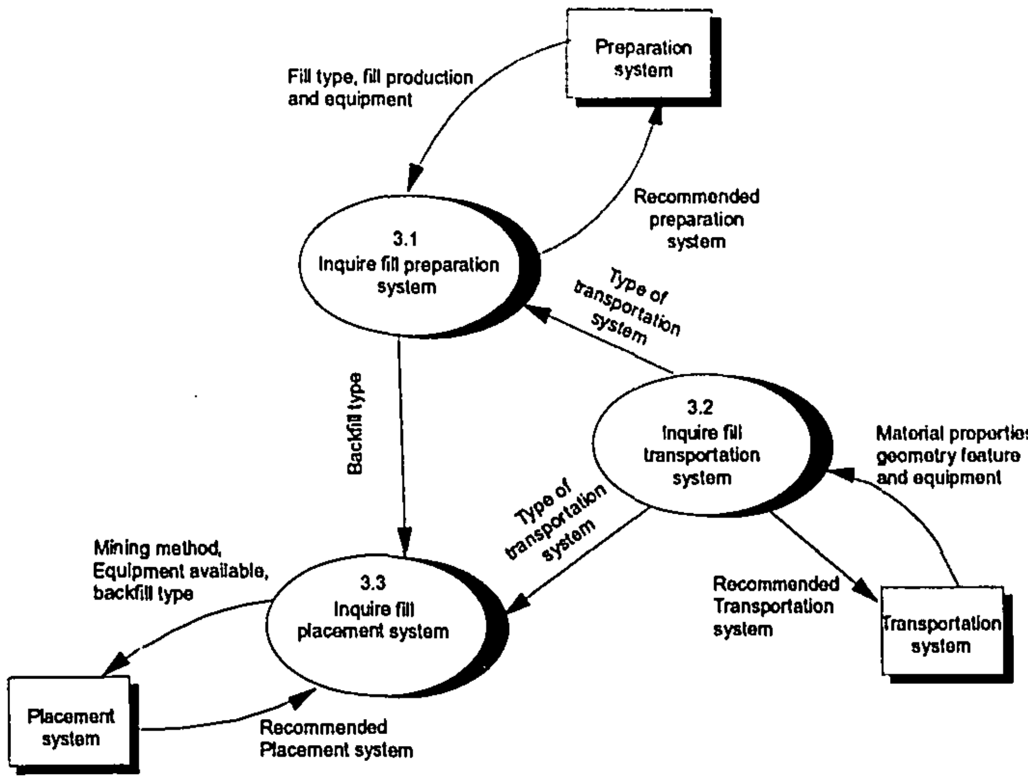



Figure 2-3b Level-Zero data flow diagram for backfill operation subsystem

This diagram is referred to as level zero diagram because of the higher level nature of representation. Subsequent lower levels are further descriptions of functions performed by the system. The user's requests are processed according to the actions specified by each process. All actions or functions are interconnected among the processes representing the system logic. In backfill design, the process 1 requests information about general geological information related to mining operation and helps the user to select a suitable mining method. The process 2 requests information about backfill materials available and backfill purpose, and then response with the fill type. Process 3 deals with the specification of operation system which includes: 1). material preparation operation; 2). transportation system operation; and 3). placement operation. In this analysis, the efforts focus on the transportation system in term of system functionality. However, information flow of material preparation operation and placement operation is investigated in terms of data storage and retrieval. To select certain transportation system, the process requests material properties, operation requirements, equipment available etc. and then response with the recommended transportation system. Process 3.2 requests the information related to the transportation system, hydraulic properties of fill slurry, size distribution etc. either from the user or from other processes and response with the answer whether pump is needed and what is the proper pump for the system. Finally, process 4 asks the cost model to estimate the overall cost for backfill system and the feasibility conclusion. The incoming information is processed, and a recommendation is delivered back to the user and to other internal processes. The diagram illustrates logical functions rather than the manner in which the system would be written to implement each function. Through this process, five basic processes have been defined as stated above. The further decomposition of processes to lower level models will be based on this framework. A complete system design requires lower level representations for each of the processes, which will be presented after a integrated architecture is discussed.

# CHAPTER 3

# FILL MATERIAL PROPERTY AND SPECIFICATION

## 3.0. INTRODUCTION

As identified earlier, the backfill design proceeds through six steps, each of which involves certain technologies and expertise. In review of literature and industrial reports, the specification of backfill system deals with the following aspects:

1. Mining Technique Specification;
2. Backfill Material Specification;
3. Backfill Operation System Specification;
4. Cost estimation;

Corresponding to the context diagram figure 2-4, the mining technique specification is related to the mining method selection; backfill material specification to fill type selection and the backfill operation system specification to the transportation system selection and pump selection. The cost estimation will be generated by system as a independent function. This chapter discusses basic principles with regard to backfill material selection and the main factors affecting the behaviors of fill slurry. This information has to be captured some way to serve as knowledge bases for backfill designer, which will be presented as a hypermedia reference manual system later.

## 3.1. BACKFILL MATERIAL SPECIFICATION

The specification of backfill material includes the following aspects:

1. Backfill material resource specification
2. Backfill material constituent specification
3. Backfill material property specification
4. Backfill material cost specification

The backfill material resource specification includes the following aspects:

1). Backfill type
2). Backfill material resource

3). Backfill material production requirement
4). Backfill material potential production

The material constituent specification include the following aspects:

1). Constituent of fill material
2). Percentage of each constituent
3). Size distribution of backfill material
4). Unit weight backfill material

Illustrated in figure 3-1 is the level-1 representation of process 2 (see figure 2-8a):

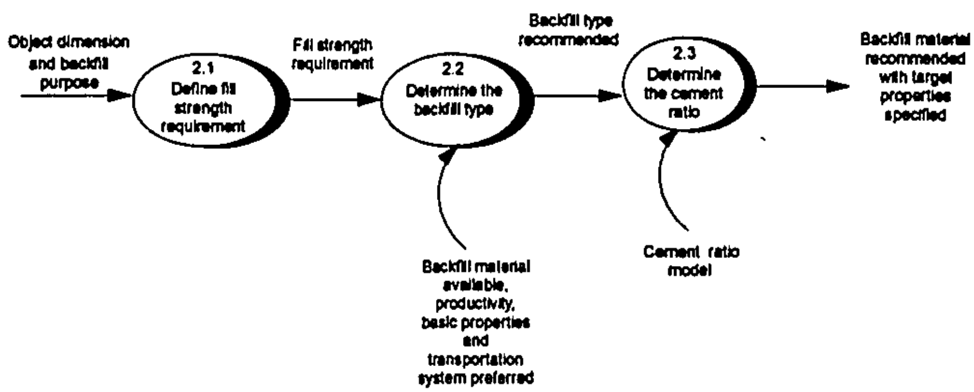



Figure 3-1 The level-1 data flow diagram of process 2

The process requests the information of the object dimension generated from process 1 and asks the backfill purpose identified from backfill designer to define the fill strength requirement. The material target properties depend on the purpose of backfilling in mining operation.

The following are the main parameters and the basic models with regard to the fill type specification. The complete discussion of fill properties and the main factors affecting the fill properties will be described later.

### 3.1.1. Fill Strength Requirement

**1. Delayed Backfill**

If the backfill purpose is for ground control or pillar recovery, then the uniaxial compressive strength of fill material is the major parameter. The basic requirement is that the fill must be stable, as a free standing wall during pillar recovery. In general, the compressive strength required is up to 5 Mpa, even 7 Mpa depending on stope size and host rock conditions. The higher strength requirement will be considered to be unreasonable and some modification of stope size or mining method are demanded to ease the strength requirement. A three-dimensional analysis used to determine necessary unconfined compressive strength, which assumes the fill block failure mode, is shown in Figure 3-2. This approach is developed on the basis of model tests and described in details by Mitchell et al[11]., Nantel, Lecuyer[12] and Arioglu[13]. The unconfined compressive strength of the fill ($\sigma_c$) is computed as follows:

$$\sigma_c = \frac{(\gamma L(H - \frac{1}{2}W \sin 2\alpha)(F \tan\alpha - \tan\phi))}{2M(L \tan\alpha + (F \tan\alpha - \tan\phi)(H - \frac{1}{2}W \tan\alpha)\sin 2\alpha)} \qquad \text{3-1}$$

Where: $\sigma_c$ = compressive strength of fill after 28 days of curing, MPa.

- $\phi$ = angle of internal friction, 30 - 45 degree
- $\alpha$ = angle of failure plane in fill block, 45 + ($\phi$/2) degrees.
- $\gamma$ = unit weight of fill, tonnes/m$^3$.
- M = a constant relating to the ratio of cohesion/compressive strength (i.e. $M = S_o/\sigma_c$). For cemented rock fill, M = 0.18. For cemented tailing fill and sandfill, M = 0.35[14].
- W = width of fill block, m.
- L = length of exposed fill, m.
- H = height of exposed fill, m.
- $H_E$ = H - ½W tan $\alpha$, the effective height of fill block, m.
- F = 3 to 5, the safety factor for stability of the backfill. It depends on ground conditions, blasting technique, height of exposed fill and pillar recovery technique.

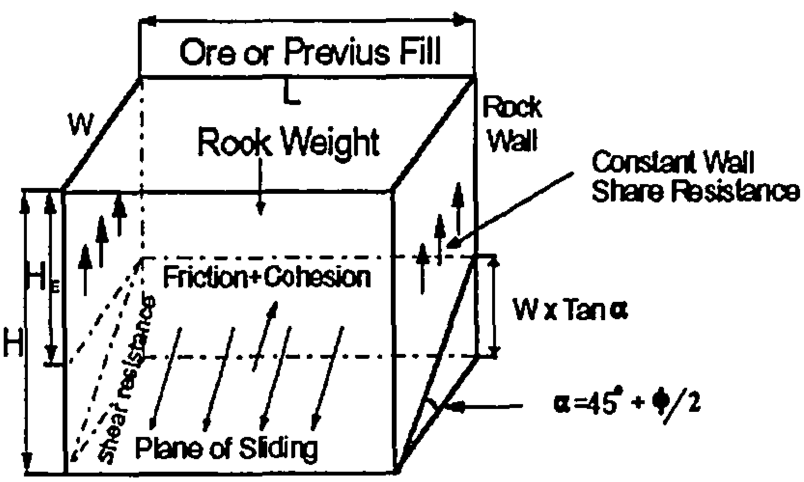



Figure 3-2 Schematic of failure model for a confined fill block

Other numerical models have also been used to predict the fill strength requirement, among them are the finite element, boundary element and other empirical equations. Once the strength requirement is calculated, the process asks for the information of backfill material available, fill productivity, basic properties of source fill material and transportation system preferred to determine the fill type. This process is illustrated as process 2.2. in figure 3-1. The process 2.3 evaluates the material available and try to match the fill with strength requirement generated earlier, permeability requirement and backfill purpose to determine whether binding agent like cement is needed. If yes, the process asks for the cement ratio prediction model to calculate the cement ratio of fill material, and finally recommends the backfill type and cement ratio.

**2. Cyclic Backfill**

In cyclic backfilling systems, fill in each operation cycle, works as a platform for mining equipment. The major factors determining traffic stresses are: load transmitted by the wheel, area of load influence, number of load repetitions, speed of vehicle, contact area of the tire, number of tires in the assembly, spacing between axles (see Figure 3-3). The general bearing capacity for fill is defined as follows[15].

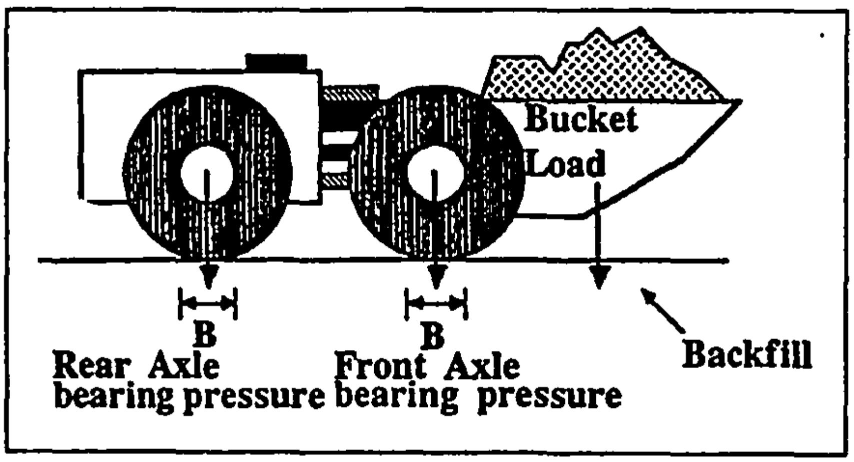



Figure 3-3 Bearing Capacity Design Criteria

$$Q = (1.3\, C\, N_c) + (0.4\, \gamma\, B\, N_\gamma) \qquad 3\text{-}2$$

Where: Q = surface bearing capacity

c = cohesive strength of backfill

$\gamma$ = bulk density of backfill

B = width of square footing at surface contact position

$N_c$ = cohesion component bearing capacity factor

$N\gamma$ = bulk density bearing capacity factor

$N_c$ and $N_\gamma$ are the bearing capacity factors of Terzaghi's theory, they depend only on the angle of internal friction $\phi$ of the backfill and are non-dimensional coefficients which characterize the bearing capacity of the fill. The values of $N_c$ and $N_\gamma$ are available from experimental nomogram. As a general rule, in cut and fill mining, the 28 day compressive strength required is lower than 1 Mpa,

### 3.1.2. Mathematical Model for Fill Strength

A number of tests with various fill constituents has been carried out to determine the parameters affecting fill strength. The effects of the compositional variables on the material strength have been analyzed and incorporated into an empirical model. The unconfined compressive strength (in Mpa) of a desired composition can be estimated according to the following relationship[16].

$$UCS = e^{\left( P1 + \left( P2\left(\frac{OPC}{W} + P3\frac{PFA}{W} + P4\frac{PBFC}{W}\right)\left(1 + P5\frac{CT}{NCS} + P6\frac{CW}{NCS}\right)\right) + P7\frac{NCS}{W}\right)} \qquad 3\text{-}3$$

where: P1 = -2.15 P2 = 5.65 P3 = 0.67 P4 = 1.60

P5 = -0.07 P6 = -0.34 P7 = 0.21

OPC = Ordinary Portland Cement

PFA = Pulverized Fuel Ash

W = Water

CT = Classified Tailings

CW = Comminuted Waste

NCS = Non-cement Solids (all material masses in same unit)

PBFC = Portland Blast Furnace Cement

Limiting ranges: Water/Cement : 2-10

Total solids/water: 2-5

Within the given limits, this model allows the estimation of the 28 day compressive strengths of cemented backfills, cured at 30°C and 100% humidity. It must be remembered that such a design equation is not intended as a definitive strength predictor, but rather as a general guide to the strengths which may be expected from materials of certain composition. Laboratory testing should not be omitted.

### 3.1.3. Fill Quantity

The quantity of fill is determined by using the following equation:

$$G = K \times \gamma \times V \qquad 3\text{-}4$$

where: $G$ = quantity of fill needed per year, tonnes/year.
$V$ = volume of stopes to be filled per year, $m^3$/year.
$K$ = coefficient concerning with fill loss = 1.02.
$\gamma$ = unit weight of fill, tonne/$m^3$ = $\gamma_t + C_o$
$\gamma_t$ = weight of fill (tailing) in $m^3$ of cemented backfill, tonne/$m^3$.
$C_o$ = weight of cement in $m^3$ of cemented fill, tonne/$m^3$.

An alternative is the replacement factors determined from the Quebec Mines Survey. Two factors $N_R$ and $N_T$, for rockfill and tailings (and sandfill) respectively were calculated and are shown below:

**Rockfill:**

$$N_R = \frac{\text{(SOLID WASTE ROCK DENSITY)} \times (0.71)}{\text{(ORE DENSITY)}} \qquad 3\text{-}5$$

**Tailings (or Sand):**

$$N_T = \frac{\text{(SOLID TAILINGS OR SAND DENSITY)} \times (0.64)}{\text{(ORE DENSITY)}} \qquad 3\text{-}6$$

The backfilling rate is closely related to the mining system and production rate and should be designed to meet operation requirements.

### 3.1.4. Water Demand

**1. Tailing Fill**

Water requirement for cemented tailing fill (W) is estimated from the following expressions:

$$W = W_1 + W_2 + W_3 \qquad 3\text{-}7$$

where: $W_1$: weight of water for repulping saturated tailing in storage tank, tonne/year.

$$W_1 = \left(\frac{1 - C_f}{C_f} - \frac{1 - C_B}{C_B}\right) G_w \qquad \text{3-8}$$

$W_2$: weight of water for mixing tailing with cement, tonne/year.

$$W_2 = \left(\frac{1 - C_g}{C_g}\right) G - \left(\frac{1 - C_f}{C_f}\right) G_w + 0.23.G_c \qquad \text{3-9}$$

$W_3$: weight of water for pipeline flushing, tonne/year.

$$W_3 = K_o \,.\, Q_m \qquad \text{3-10}$$

G: weight of fill, tonne/year.

$$G = G_w + G_c \qquad \text{3-11}$$

$G_w$: weight of tailing (or rock or sand), tonne/year.

$G_c$ : weight of cement, tonne/year.

$C_B$: solid concentration of saturated tailing = 82%.

$C_f$: solid concentration of slurry discharged from silo = 72%.

$C_g$: solid concentration of slurry placed underground = 65 - 85%.

$K_o$: volumetric ratio of flushing water to slurry = 0.04.

$Q_m$: volume of slurry placed underground, m³/year.

Hence:

$$W_1 = \left(\frac{1 - C_f}{C_f} - \frac{1 - C_B}{C_B}\right) G_w + \left(\frac{1 - C_g}{C_g}\right).\, G - \left(\frac{1 - C_f}{C_f}\right).G_w + 0.23.G_c + K_o \,.\, Q_m \qquad \text{3-12}$$

The ratio of water to fill is expressed as follows:

$$\frac{W}{G} = \left(\frac{1 - C_f}{C_f} - \frac{1 - C_B}{C_B}\right)\frac{G_w}{G} + \left(\frac{1 - C_g}{C_g}\right) - \left(\frac{1 - C_f}{C_f}\right).\frac{G_w}{G} + 0.23.\frac{G_c}{G} + K_o \frac{1}{C_g} \qquad 13$$

where: $\frac{G_w}{G}$ = ratio of tailing to cement fill by weight.

**2. Rockfill and Sandfill**

The water needed for <u>cemented rockfill or sandfill</u> can be estimated by using the following expressions:

$$W = \left(\frac{1 - C_g}{C_g}\right) G + 0.23.G_c + K_o \frac{G}{C_g} \qquad \text{3-14}$$

In equation (15), the ratio of water to rockfill or sandfill is expressed as follows:

$$\frac{W}{G} = \left(\frac{1 - C_t}{C_t}\right) + 0.23.\frac{G_c}{G} + K_t \frac{1}{C_t} \qquad 3\text{-}15$$

When rockfill or sandfill is not transported by pipeline, the ratio of water to solid fill is expressed by the following equation:

$$\frac{W}{G} = \left(\frac{1 - C_t}{C_t}\right) + 0.23 \,.\, \frac{G_c}{G} \qquad 3\text{-}16$$

### 3.1.5 Fill Quality

There is little data regarding the quality control of the in-situ backfill with respect to its physical and mechanical properties and this is certainly an area that requires further investigation. In most cases optimization of the preparation and transportation of the existing backfill will result in a higher pulp density, higher strength, and a reduction in the backfill cost.

### 3.1.6 The permeability

From hydraulic point of view, the permeability is an important measurable property. The general accepted percolation rate is 100 mm/h. To reach this, the fraction of 10 $\mu m$ or less particles in classified tailings is usually less than 20% of total mass by weight.

## 3.2. PHYSICAL - MECHANICAL PROPERTIES OF FILL

Mechanical properties of fill placed underground change remarkably depending on its composition and preparation procedure. Factors affecting mechanical properties of fill have been analyzed and their relationships, determined by laboratory research and in-situ monitoring, are as the following:

1. Specific gravity of fill solids
2. Void ratio and porosity
3. Moisture content
4. Particle size
5. Consolidation characteristics
6. Compressive strength
7. Internal friction angle and cohesion
8. Percolation rate, permeability.

### 3.2.1 Specific gravity of fill solid

Specific gravity of fill solid, $\gamma$, is defined as the weight of solid in an unit volume, and varies in a wide range depending on fill material. Practice indicates, most cement applied fill possesses specific gravity, $\gamma$, from 2.6 to 4.0 $t/m^3$.

### 3.2.2 Void ratio, porosity and relative density

Void ratio, e, of a fill material is defined as the ratio of the volume of voids, $V_v$, to the volume of fill solids, $V_s$,

$$e = \frac{V_v}{V_s} \qquad \textbf{3-17}$$

Porosity, n, of a fill is defined as the ratio of volume of voids, $V_v$ to volume of total fill mass V:

$$n = \frac{V_v}{V} \qquad \textbf{3-18}$$

Backfill practice indicates typical void ratios as 0.25 to 0.75 for hydraulic sandal and 0.50 to 0.85 for rock fills. The relationship between n and e is expressed as follows:

$$n = \frac{e}{1+e} \qquad \textbf{3-19}$$

The porosity, n, is in the range of 0.42 - 0.48 for hydraulic fill, 0.35 - 0.42 for high densified backfill.

Relative density, $D_d$ is a useful parameter to define the status of a backfill between maximum and minimum void ratio states, expressed by the following equation:

$$D_d\,(\%) = \frac{e_{max} - e}{e_{max} - e_{min}} \times 100 \qquad \textbf{3-20}$$

Where: $e_{max}$ = void ratio in loosest state

$e_{min}$ = void ratio in densest state

e = in situ void ratio

### 3.2.3 Moisture content

Moisture content can be defined on either a dry - weight basis or a total - weight basis. Dry fill placed by car or conveyer generally has much less moisture than hydraulic fill. Since most fill is prepared and placed in form of a slurry, the moisture content is a certain fraction of the total weight of slurry, and expressed by percent of weight[17], [18]:

$$\text{Moisture content} = \frac{\text{moist mass - dry mass}}{\text{moist mass}} \% \qquad 3\text{-}21$$

For hydraulic fill and high density fill, after being placed, the moisture content is in the range 15 - 22% depending on the permeability of the filled body and local underground conditions.

### 3.2.4. Particle size

#### 3.2.4.1 Tailing and sand fill

An accurately determined particle size analysis is probably the most useful information available on any fill material, especially for comparison with similar fills to aid preliminary design parameters.

Tailings and deslimed tailings fill size analyses presented in Figures 3-4 and 3-6 are broadly representative of practice covering a range of countries and ore types. Size analyses of a series of natural sands either used as fills or considered for such use are presented in Figure 3-7

In order to qualify the distribution characteristics, a number of indices has been used to summarize the graphical representation, the most widely used index is the coefficient of uniformity, $C_u$:

$$C_u = \frac{D_{60}}{D_{10}} \qquad 3\text{-}22$$

The second index is the coefficient of curvature $C_c$.

$$C_c = \frac{[D_{30}]^2}{[D_{60} - D_{10}]} \qquad 3\text{-}23$$

Where: $D_{10}$ = diameter of 10% passing size

$D_{30}$ = diameter of 30% passing size

$D_{60}$ = diameter of 60% passing size

Well graded material contains equal representations of all size fractions with $C_u$ values of 4 to 6, and $C_c$ values of 1 to 3[19]. For paste backfill a broad size distribution is often desired; since there are aggregates and slimes together the coefficient of uniformity can be very high, 2-500.

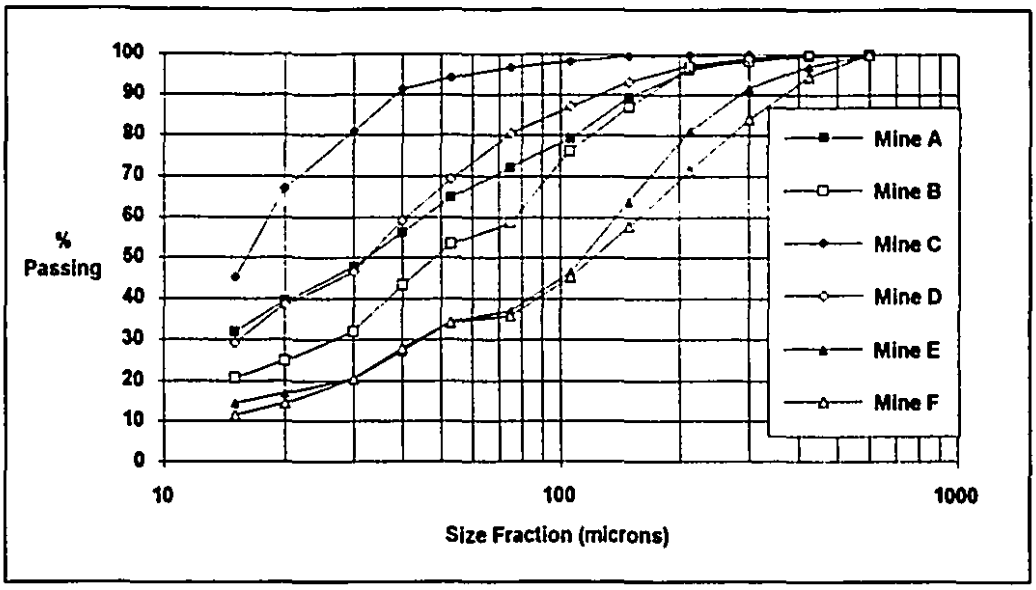



Figure 3-4 Tailing Size Analyses [Thomas, 1979]

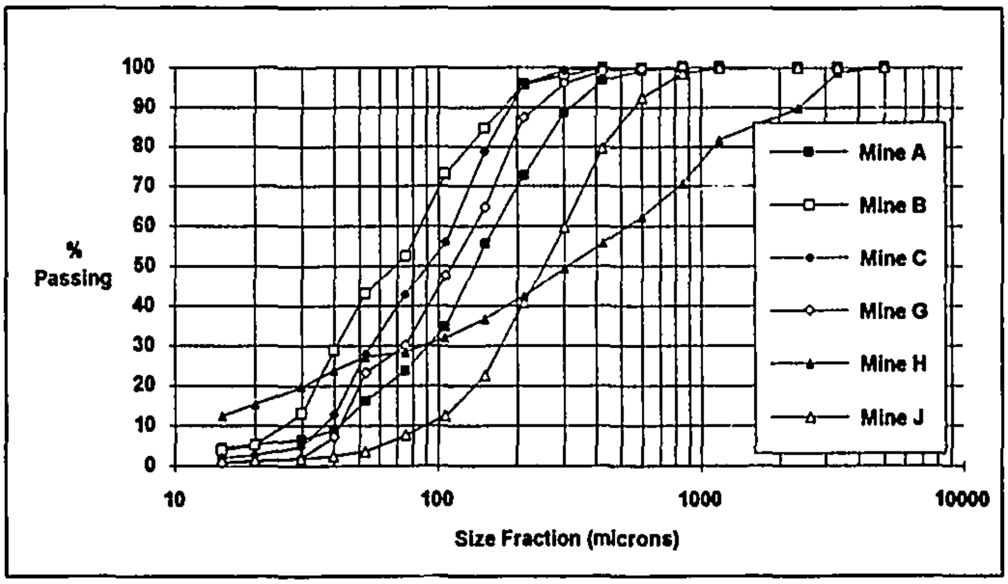



Figure 3-5 Deslimed Tailing Size Analyses [Thomas, 1979]

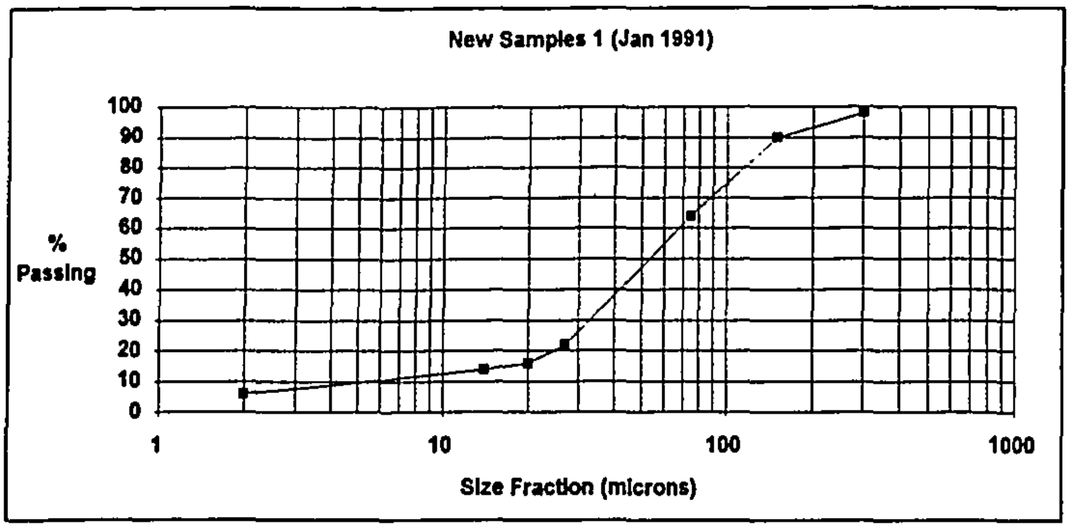



Figure 3-6 Mobrun Mine, Quebec - Tailing Size Distribution

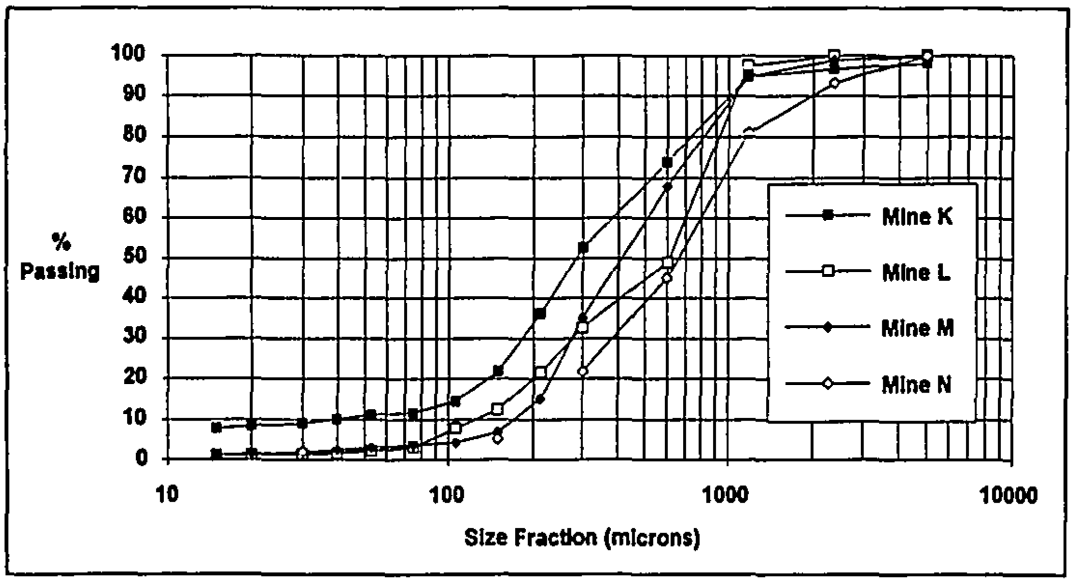



Figure 3-7 Natural Sands Size Distributions [Thomas, 1979]

#### 3.2.4.2 Rockfill

The main types of rockfill are waste rock, gravel, smelter slag or ore processing reject. The maximum grain size is generally limited in accordance with transport means, for pipeline transportation rock must be crushed to smaller than 1/3 diameter of pipe, in case of transportation by conveyor, maximum grain size can be much bigger, at Kidd Creek, rockfill size is up to 15 cm, and at Mount Isa, the top size reaches 30 cm.

### 3.2.5 Consolidation characteristics

Consolidation characteristics are determined in an oedometer, with a cylindrical sample typically 18 mm thick and 76 mm diameter. Loading is usually through a dead-weighted lever arm, though a more recent development employing fluid pressure is proving very suitable for use with fills. The term "consolidation" is in strict application confined to compression where rate is controlled by water movement from the sample.

### 3.2.6 Compressive strength

Compressive strength is usually of application only for cemented fills since these are usually the only fills exposed in such a way as to allow a fill failure related to fill compressive strength to occur.

### 3.2.7 Percolation Rate, Permeability

Permeability is the most important measurable property of fills. The procedure used is based upon the constant-head permeability test and invariably features a sample held in a glass or perspex tube with a porous base; hydraulic gradient of near unity is used. Test equipment and test procedure are not standardized from mine to mine, or country to country. The only standard factor is the answer required, a percolation rate of 100 mm/h, (for hydraulic fill). The definition of permeability is derived from Darcy's law, as:

$$K = \frac{Q \cdot L \cdot \eta}{h \cdot A \cdot \gamma} \qquad \text{3-24}$$

where: $K$ = permeability of porous medium, $m^2$.

$Q$ = rate of flow of fluid through porous medium, N/sec.

$L$ = length of porous medium, in direction of flow, m.

$h$ = static pressure differential across porous medium, m.

$A$ = cross-sectional area of porous medium, normal to flow direction, $m^2$.

$\eta$ = absolute viscosity of fluid flowing, N sec/$m^2$, and

$\gamma$ = unit weight of fluid flowing, N/$m^3$.

This expression for permeability is valid only while Darcy's law is valid, though this is usually the case during testing and dewatering of fill materials. Also, Darcy's Law inherently improves in range of validity as fill fineness increases. Percolation rate and permeability are in fact directly related, as described below. If the rate of flow Q be expressed in $m^3$/sec instead of N/sec.

$$K = \frac{q \cdot L \cdot \eta}{h \cdot A \cdot \gamma} \text{ since } Q = q \cdot \gamma \qquad \text{3-25}$$

For a unique solution at constant temperature, $\eta/\gamma$ is constant and the equation becomes:

$$K = \frac{q \cdot L}{h \cdot A} \qquad 3\text{-}26$$

where the units (m/sec) of k (permeability coefficient) is the same as those of percolation rate. Percolation rate is defined by the equation:

$$V_p = \frac{q}{A} \qquad 3\text{-}27$$

where: $V_p$ = percolation velocity, m/sec.

q = rate of flow, $m^3$/sec , and

A = cross-sectional area of sample normal to flow direction, $m^2$.

Paste backfill has an advantage over hydraulic fill since the permeability is expected to be very low. A distinct cost benefit is the elimination of bulkheads. The paste should carry enough water to enable flowability but drainage water should be minimal.

## 3.3. FACTORS AFFECTING PROPERTIES OF FILL

### 3.3.1 Strength characteristics of uncemented fill

#### 3.3.1.1 Shear strength

Strength of uncemented fills has two components:

1) An apparent cohesion which results from surface tension forces in capillary water and disappears at dryness and saturation, and
2) Interlocking of grains, the extent of which is independent of moisture content except insofar as pore pressures within fill water can reduce effective confining pressures.

Apparent cohesion is an extremely important property of uncemented fills, which is dependent upon:

- particle size,
- particle shape,
- overall particle grading, and
- packing density,

The importance of grain interlocking and hence the magnitude of the internal friction angle also depend upon grain shape, overall particle size and packing density's shown in Table 3-1.

Table 3-1 Effect of particle angularity and grading and packing density on friction angle.

| Shape and grading | Loose | Dense |
|---|---|---|
| Rounded, uniform | 30° | 37° |
| Rounded, well graded | 34° | 40° |
| Angular, uniform | 35° | 43° |
| Angular, well graded | 39° | 45° |

### 3.3.1.2 Compressibility and load bearing capacity

#### 1. Compressibility of tailing fill

Nicholson and Busch reported the compressibility of backfill under the low-confinement and the high-confinement conditions (Figure 3-8). Curve 3 represents the triaxial stress conditions frequently employed in testing of fills. Curve 2 represents earth pressure at rest condition where lateral strain is not allowed and lateral stress development and axial stress development are related through the coefficient of earth pressure at rest. Most practical fill situations lie between Curves 2 and 3, some lateral strain being possible and lateral stress not remaining constant but increasing with increasing axial stress.

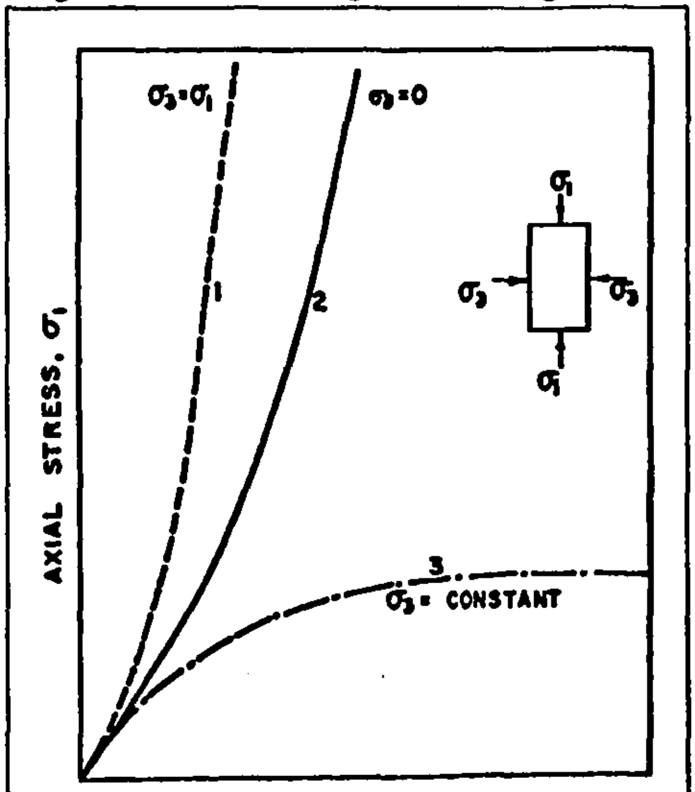



Figure 3-8 Axial Stress-strain Curves for Granular Material Subjected to Various Confining Stress (after Hendron, 1963).

The stress-strain characteristics are presented in Figure 3-9[20], which very clearly indicates the substantial load bearing advantage that milled waste then has over the other uncemented fills.

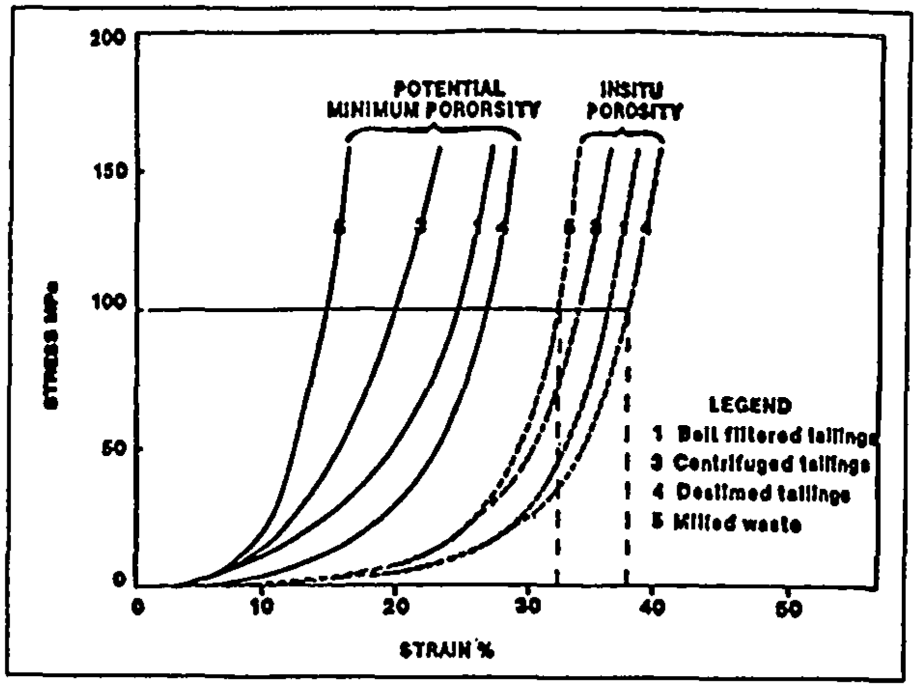



Figure 3-9 Stress-strain Curves for Various Fills Tested at their Observed Underground In-situ Porosity and at their Potential Minimum Porosity as Defined by the Placement Properties Test.

Effect of cement: The addition of cement to a backfill such as dewatered tailings cause the fill to develop a significantly stiffer stress-strain response in the low strain region. This is illustrated in Figure 3-10 in which stress-strain curves are given for cemented and uncemented tailings prepared to the same porosity. The very substantial increase in early loading stiffness is well demonstrated by curve 2 in Figure 3-11.

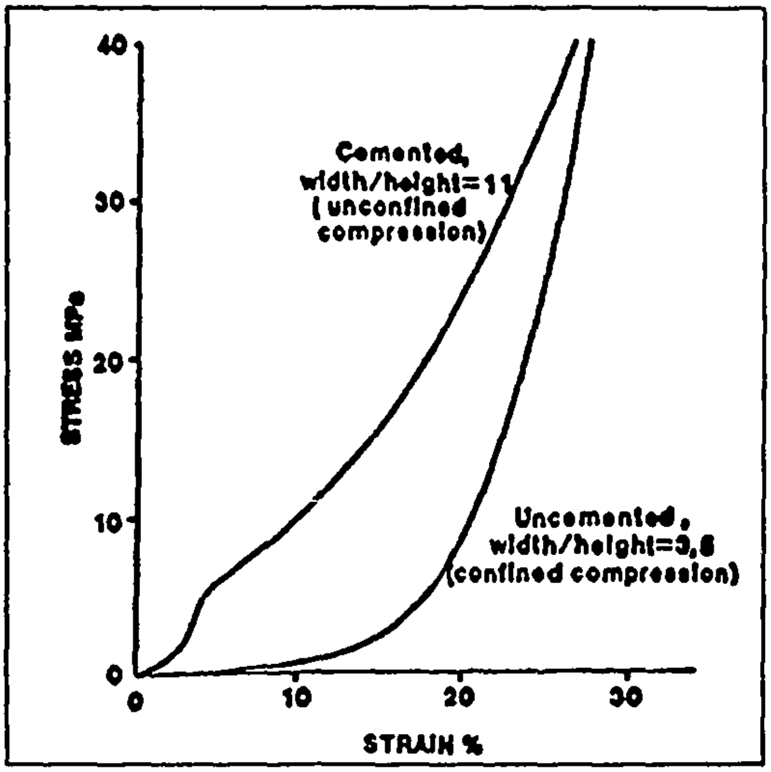



Figure 3-10 Stress-strain Curves for Cemented and Uncemented Tailings Prepared to the same Porosity. [Stewart 1986]

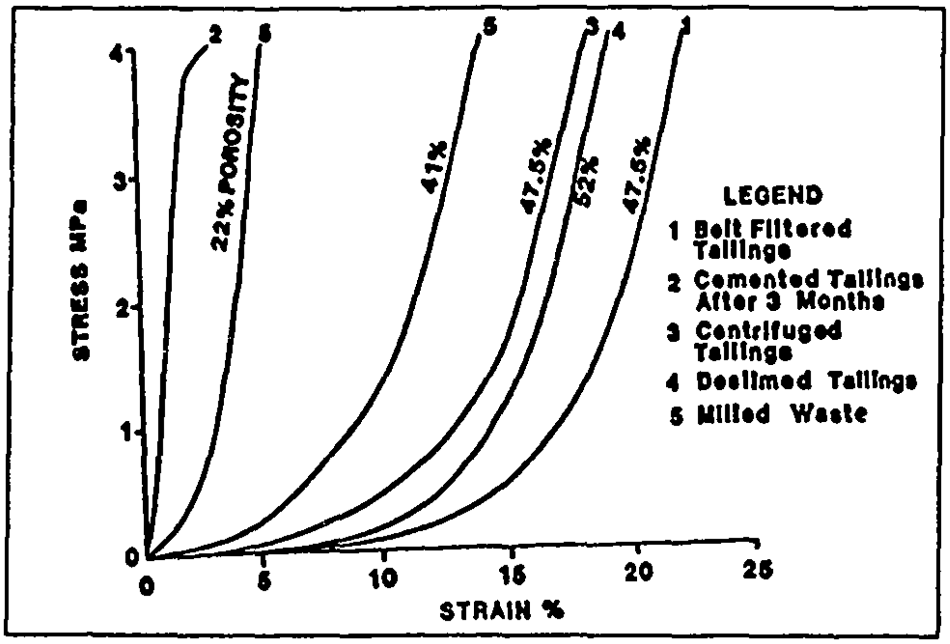



Figure 3-11 Stress-strain Curves, in the Low Stress Region, for Various Backfills

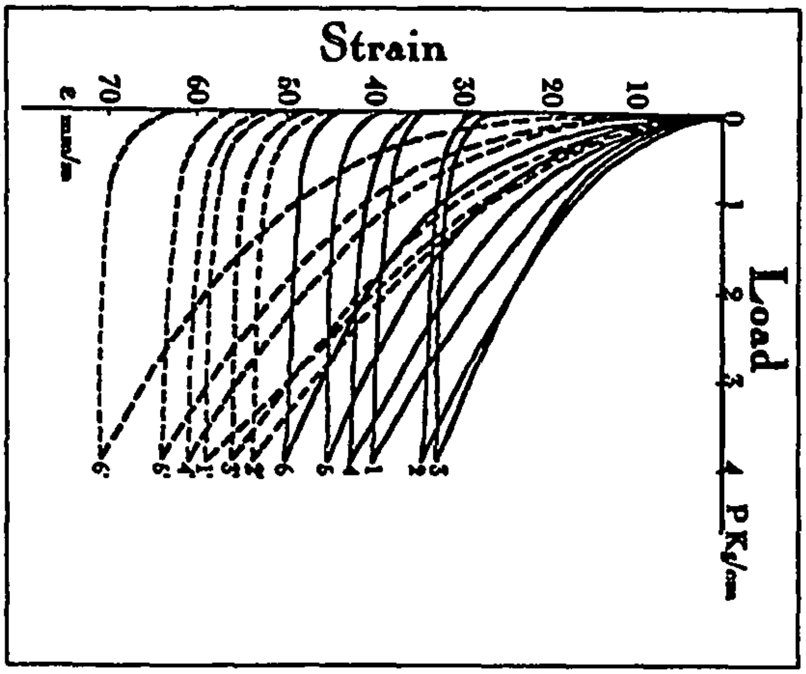



Figure 3-12 Compression Curves of Tailings with Different Particle Diameters[21]

Key to Figure 3-12 Solid line - dry tailings, Dotted line - saturated tailings

1. $d_1$ = 0.455mm (d = average particle size) 2. $d_2$ = 0.203mm
3. $d_3$ = 0.113mm 4. $d_4$ = 0.086mm
5. $d_5$ = 0.061mm 6. $d_6$ = 0.045mm

Effect of particle size. The compression tests carried out with different tailing fill, indicate, that particle size affects the compressibility remarkably. The loading and unloading curves are illustrated in Figure 3-12. The solid line indicates the dry tailings and dotted line saturated tailings.

Effect of moisture content. From Figure 3-12 it is obvious that for the same particle size of tailing there is a clear difference in compression curves between dry and saturated sand. The modulus of saturated sand, E, is between 85-115 kg/cm², but of dry sand is 130-170 kg/cm², that means deformation of dry sand is smaller than saturated and explains the need for good dewatering to ensure that the optimum strength is obtained.

### 2. Compressibility of rockfill

The same rule governs the compressibility of crushed rock, it closely relates to grain size composition, porosity, water content and confinement. Curves in Figure 3-13 show the compression characteristics of crushed dolomite with porosity ranging 32-43%. The compression rate varies between 22-29% under a pressure of 15 MPa.

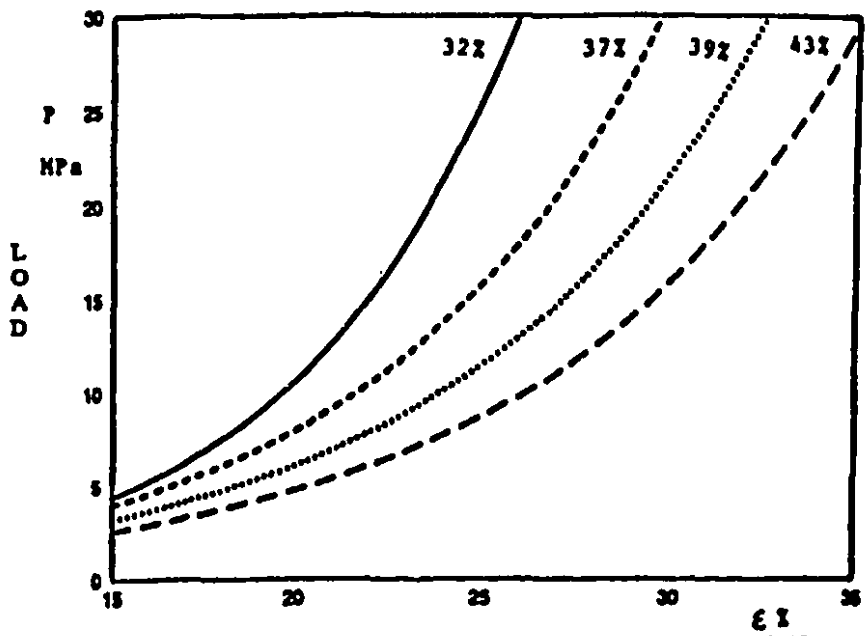



Figure 3-13 Compression Curves of Dolomite (crushed)[22]

### 3.3.2 Strength Characteristics of Cemented Fill.

### 3.3.2.1 Compressive Strength of Cemented Tailing Fill.

The main factors affecting compressive strength are summarized as follows.

Effect of cement content Portland cement is the most universally used, the most predictable, reliable, convenient, and the most expensive fill additive. After a comprehensive study on compressive strength of cemented tailing, the experimental data are plotted in Figure 3-14, which summarizes the relationship between compressive strength and cement content, curing time and grain size composition[23].

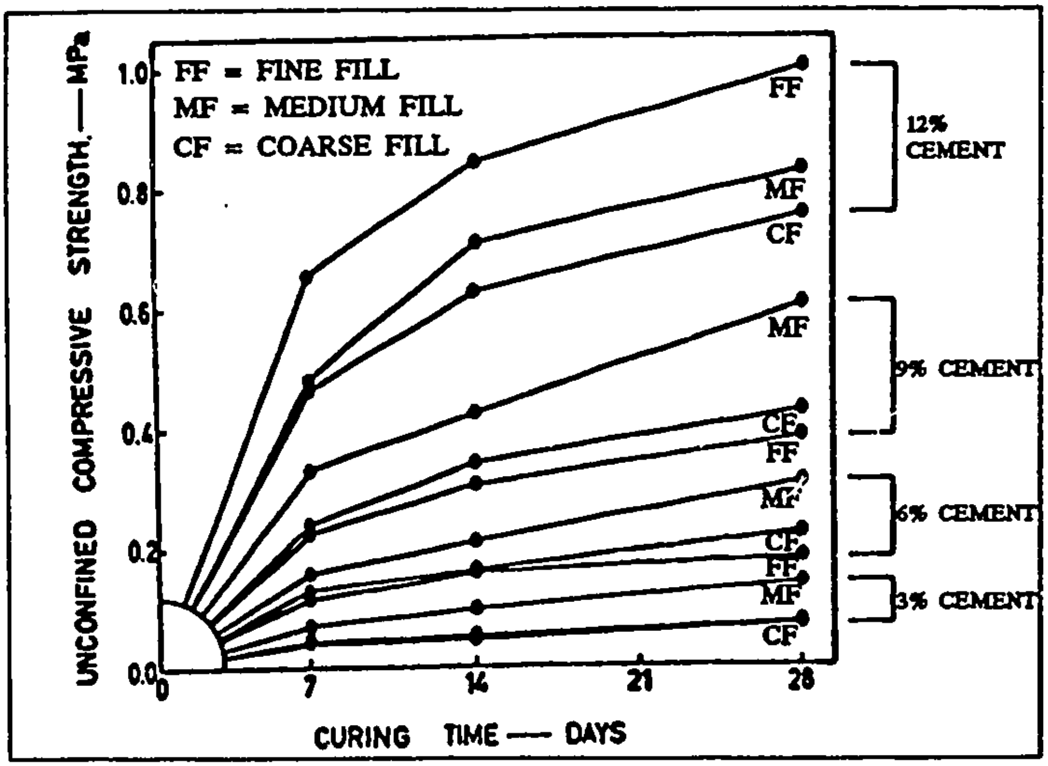



Figure 3-14 Pancontinental Ltd., Tests-curing Curves to 28 days for Three Fill Grades (coarse, medium and fine) and Range of Portland Cement Addition Level [Thomas, 1981]

Figure 3-15 shows the maximum compressive strengths of compositions comprising full tailing with increasing amounts of cement, the water/solids ratio of samples were kept constant at 0.39, after 90 days this increase is no longer significant.

Effect of backfill type Figure 3-16 shows a test of three compositions of fill all containing 10% cement. The difference in strength between cemented backfills derived from belt- filtered tailings or cyclone - classified tailings is a function of their ultra - fine (-10 micron) particle fraction. Considerably higher strengths can be achieved if cement is added to well-graded comminuted waste (coefficient of uniformity, $C_u = D_{60}/D_{10} = 35$), which has a lower placement porosity than tailing-based backfills.

Effect of water/solid ratio Ideally the water/solids ratio in cemented backfill should be as low as possible. A low water content of the backfill material enhances the intrinsic bonding strength of the cement in addition to minimizing the place porosity and reducing segregation of the fines. Figure 3-17 clearly demonstrates the advantage of operating at a high slurry density, as the strength increases more than threefold over the test range.

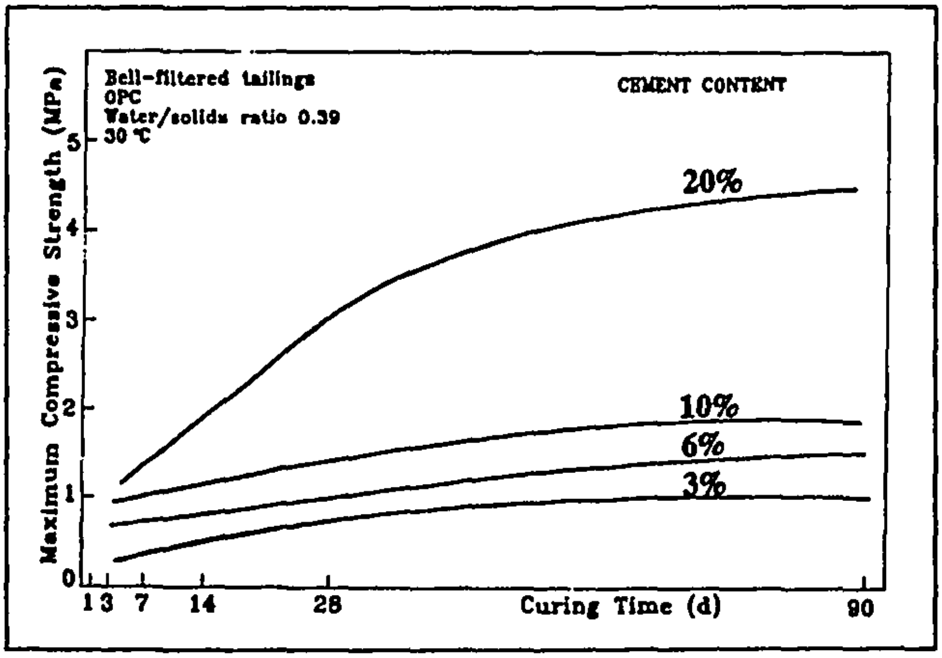



Figure 3-15 Maximum Compressive Strengths of Compositions with a Range of Cement Contents. [Lamos, Clark, 1989]

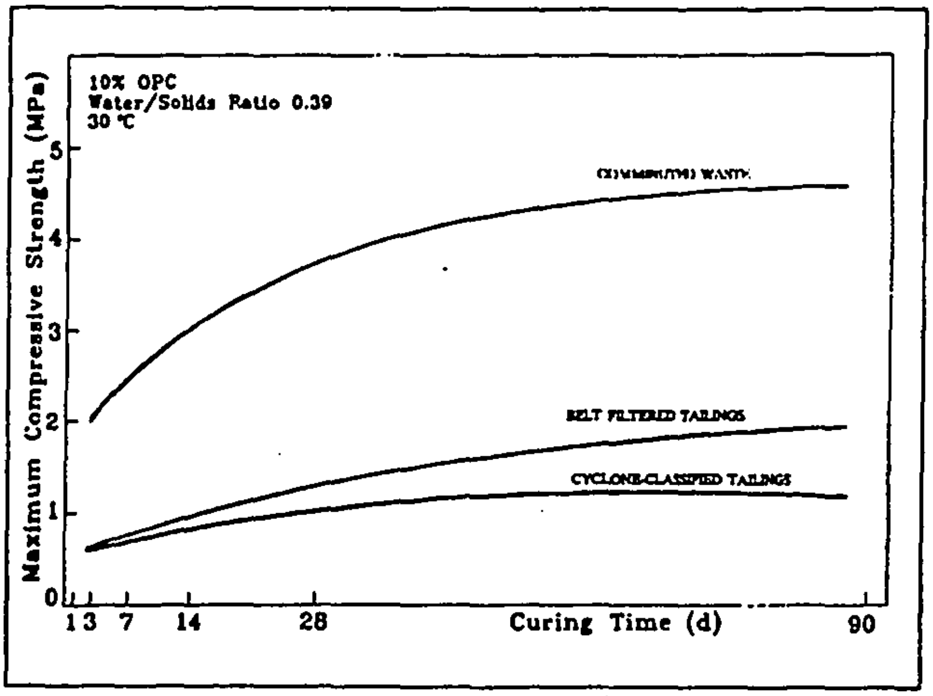



Figure 3-16 Maximum Compressive Strengths of Compositions Comprising Different Backfill Types. [Lamos, Clark 1989]

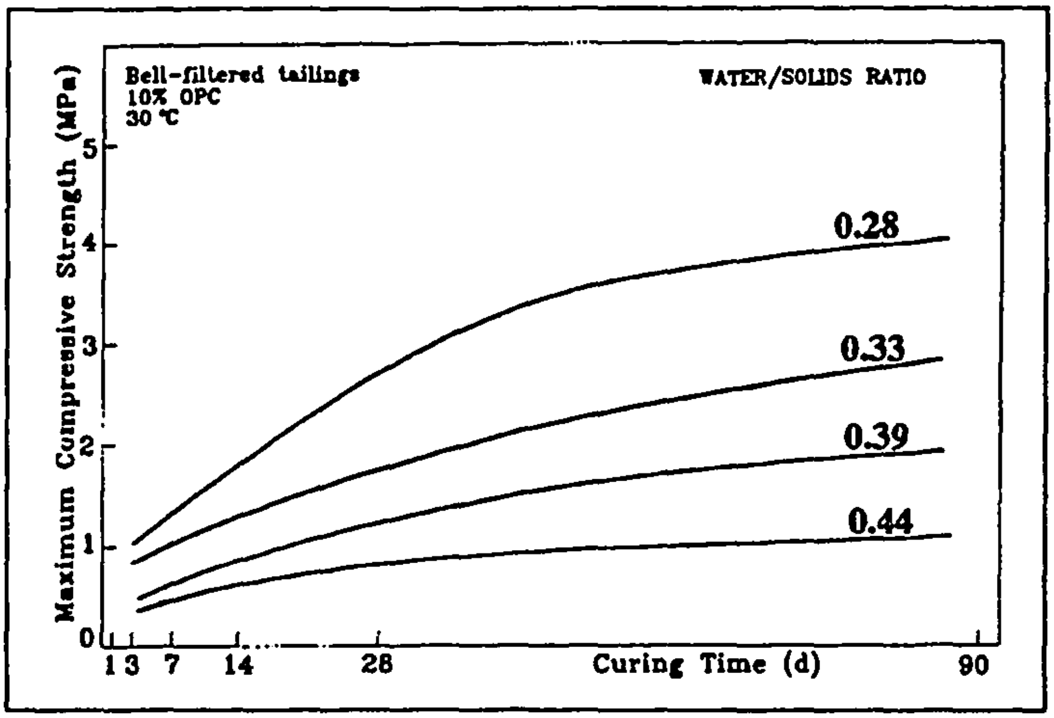



Figure 3-17 Maximum Compressive Strengths of Samples Prepared to a Range of Water/solid Ratios. [Lamos, Clark 1989]

Effect of cement type. Test involving different binders have been reported. The results are shown in Figure 3-18. Backfill containing ordinary Portland Cement (OPC) is stronger than that containing a blend of OPC and pulverized fuel ash (PFA), and as the proportion of PFA increases so the strength of the cemented backfill decreases. At the same concentration, Portland Blast furnace cement (PBFC) produces a much stronger fill material than OPC.

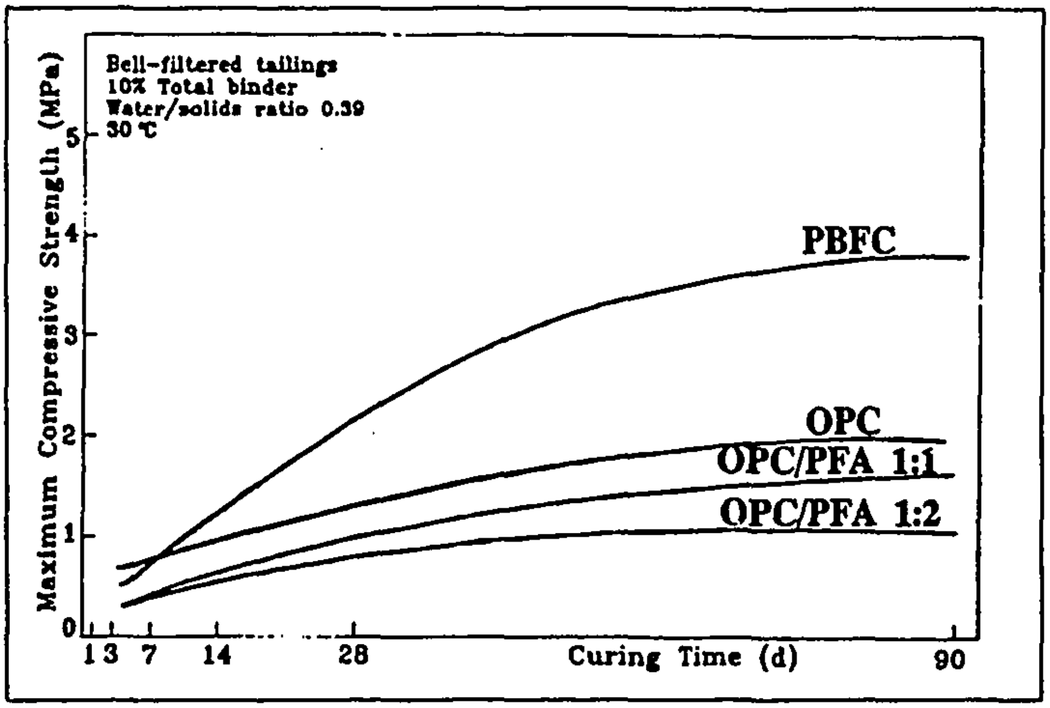



Figure 3-18 Maximum compressive strengths of compositions using different binders[24]

### 3.3.2.2 Compressive strength of cemented rockfill (CRF)

The factors that affect the strength and behavior of CRF are summarized below:

Cement content: The cement content greatly affects the strength of CRF mixtures. CRF can be considered similar to a weak concrete mixture. Arioglu[25] investigated the variation of CRF strength properties with cement content and concluded that the compressive strength, indirect tensile strength, cohesion and elasticity increases with increased cement content. The relationship connecting the two parameters was derived from a statistical assessment of experimental values [Arioglu, 1983]:

$$\sigma_c = A \cdot (\alpha)^n \qquad \textbf{3-28}$$

where: $\alpha$ = water/cement ratio by weight = $W_w/W_c$

$W_c$ = content of cement. Kg/m³

$W_w$ = content of water. $Kg/m^3$

A = constant = 80.268

n = constant = -1.536

Yu[26] also reported that the compressive strength of CRF at 28 days cure from a Kidd Creek fill sample showed a relationship to the cement content by:

$$Q_u = 1.5\, e^{0.25c} \quad \text{for } 5<c<25 \qquad 3\text{-}29$$

where: $Q_u$ = uniaxial compressive strength (MPa)

c = Portland Cement content by weight % of -4 cm aggregate

Water/cement ratio (% mixing water). The water/cement ratio is of the greatest importance in strength considerations of CRF. Knissel and Helms[27] has shown that for the same size, age and cement content, CRF have different compressive strength which vary with the moisture content. This is illustrated in Figure 3-19. A closer look at Figure 3-20 shows that the optimum compressive strength can only be obtained at one moisture content (the optimum moisture content).

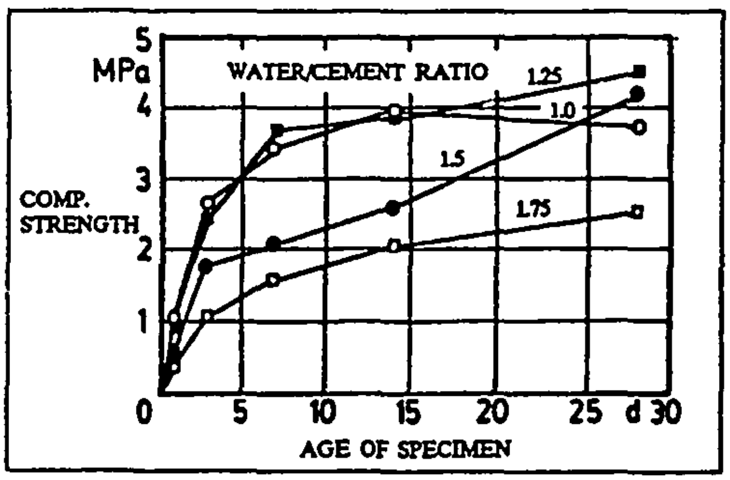



Figure 3-19 Uniaxial compression strength of mixtures with different w/c - ratios versus age of specimens (Cylindrical specimens, h=100 mm, h/d=2).

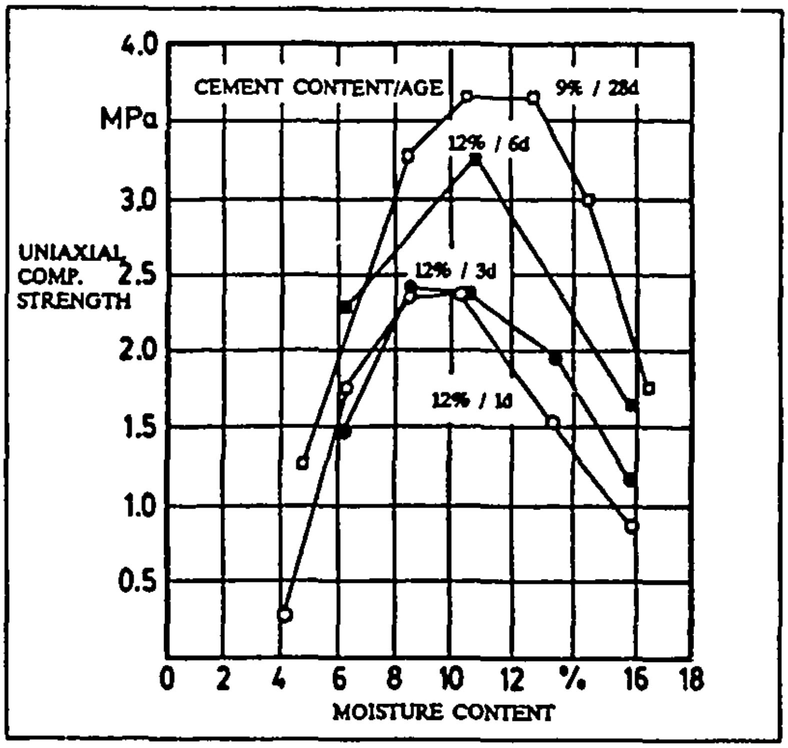



Figure 3-20 Variation of compressive strength with moisture content for the same size, age and cement content (Knissel & Helms, 1983)

3) Coarse aggregate/fine, aggregate/cement ratio. The strength properties can be optimized if the right properties of coarse aggregate, fine aggregate and cement are combined. At Kidd Creek Mines, sand was added to CRF to act as a void filler and reduce segregation. The increase in sand content, however, led to a decrease in the fill strength (see Figure 3-21). Also at the Cannon Mine (Wash., U.S.A.), the testing program proved that the strength of the fill depended on the proportion of coarse aggregate to cement. Figure 3-22 illustrates results of the tests and shows that coarse aggregate/fine aggregate/cement ratio of 65% / 29% / 6% by weight gave the highest uniaxial strength. The strength of CRF also varies with the size of aggregate (Jeremic, 1987). Figure 3-23 illustrates the variation of strength with aggregate size for model rock aggregate pillars.

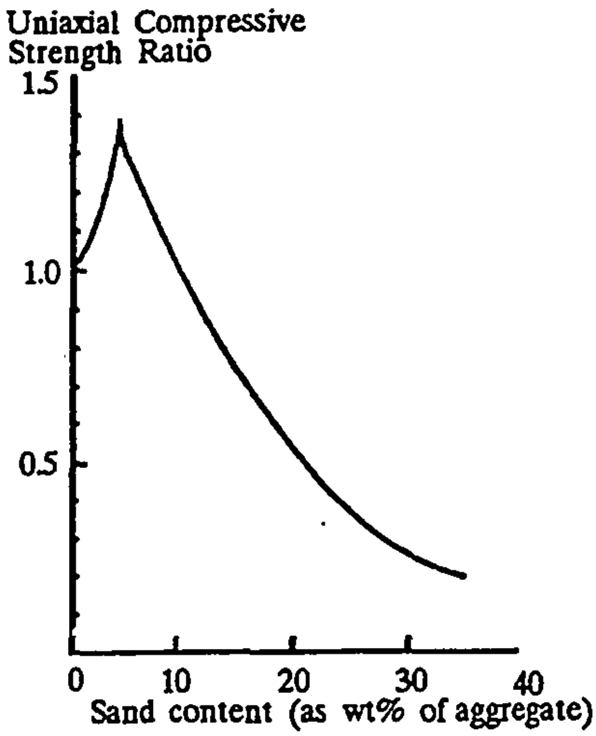



Figure 3-21 Compressive Strength vs. Sand Content

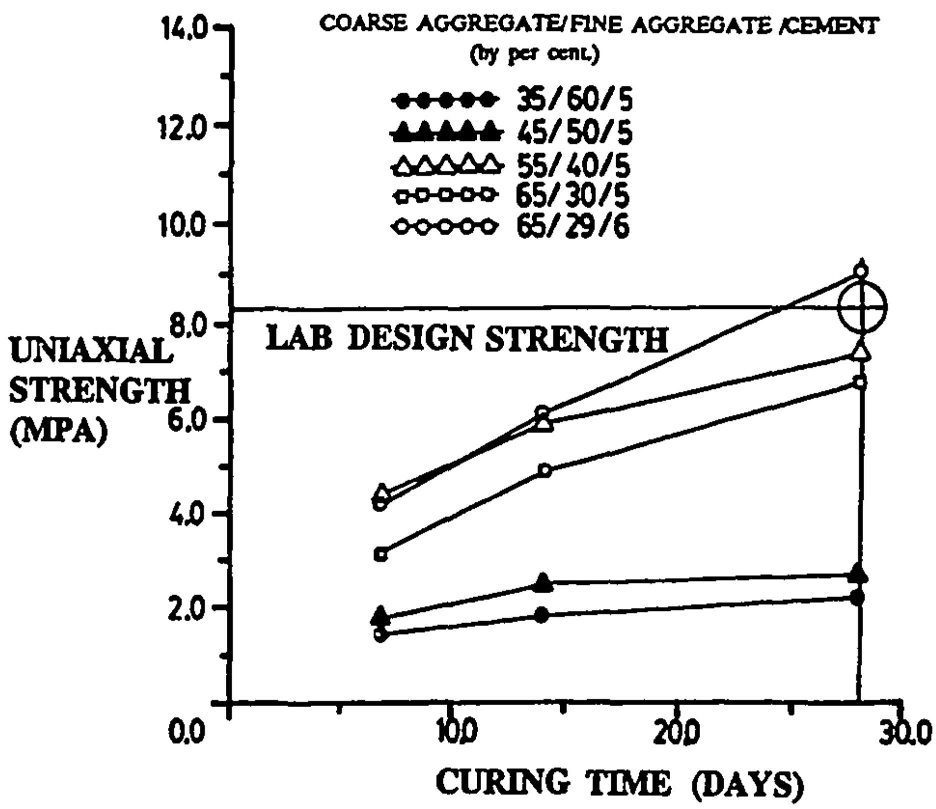



Figure 3-22 Effect of coarse aggregate content on backfill strength (Brechtel, Hardy, Baz-Dresch & Knowlson, 1989)

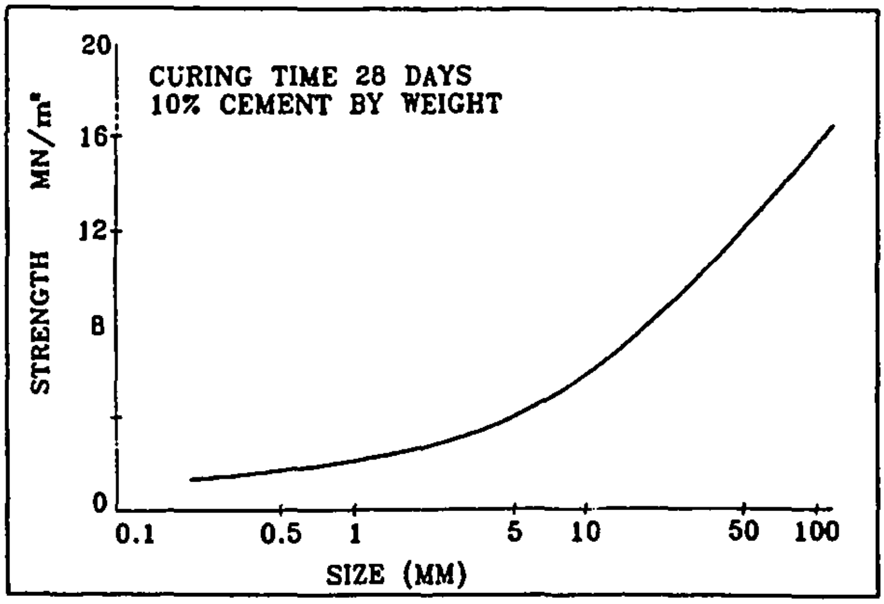



Figure 3-23 Variation of fill strength with size of aggregates (Jeremic, 1987)

### 3.3.3 Binder Alternatives

Cemented fills introduce a new flexibility into mine planning and design; however, cost considerations limit potential applications. The operating cost can be reduced by replacing Portland cement with a less expensive binder alternative. Iron blast furnace slag, power station fly ash and various natural slags and ashes have long been known for their pozzolanic properties. E. Douglas et al[28] reported the experiments with slags from different sources. In a recent study, made by P. Hopkins and M. Beaudry in 1988, five binder alternatives were tested[29]; Strength developments of cemented rockfill were tested using flyash at different proportions with cement. The experiments were performed for Kidd Creek, Bousquet and Page-Williams mines[30]. In addition, Anhydrite ($CaSO4$) Anhydrite has also been used as a monolithic packing material in coal mines but they have potential, as research at McGill University has shown[31], for hard rock mining.

### 3.3.4 Permeability (Percolation Rate)

Permeability (percolation rate) is an extremely important fill property. The universally accepted percolation rate value is 100mm/h and this is probably valid since most authors quote the percolation rate close to (preferably slightly above) 100mm/h. Percolation rate is closely related to grain size composition, particularly fine particles and cement content in the fill.

#### 3.3.4.1 Effect of size composition on percolation rate

Bates and Wayment[32] conducted some 135 routine percolation rate tests on actual mine fill materials from seven different mines. Their test results indicated that they could satisfactorily calculate percolation rates using the empirical multi-variable equation.

$$\mathbf{Ln\,(P_{20} \times 25.40) = 11.39147 + 2.853422\,ln\,(e \times D_{10}) + 0.1747436\,e \times C_u - 178.8039\,D_{10} \times D_{60} + 311.7034\,(D_{60})^2} \qquad \mathbf{3\text{-}30}$$

Where: $(P_{20} \times 25.40)$ = percolation rate at 20°c, mm/h

Ln = natural logarithm,

e = void ratio

$D_{10}$, $D_{50}$= 10 per cent. finer, 50 per cent. finer grain size, mm

$C_u$ = coefficient of uniformity ($D_{60}/D_{10}$ particle sizes)

Provided test samples satisfied the following conditions, (ranges expanded by Thomas, 1965)

1)e: between 0.430 and 1.080,

2)$D_{10}$: between 0.003 and 0.105mm,

3)$D_{60}$: between 0.0053 and 0.240mm,

4)$C_u$: between 1.77 and 22.0

Based on test results, Mitchell proposed an empirical equation to calculate percolation rate of tailing fill. Which is given as [33]

$$\mathbf{P = 5000\,\frac{(D_{10})^2}{e^{\left(\frac{0.45-n}{0.16}\right)}}\,Cm/h} \qquad \mathbf{3\text{-}31}$$

Where: $D_{10}$ - effective grain size, mm.

n - porosity of tailing fill.

### 3.3.4.2 Effect of cement on percolation rate

Addition of Portland cement greatly reduces permeability of hydraulic fill. Mitchell reported that the addition of small quantities of cement to classified hydraulic backfill will not alter the initial porosity significantly. Cementation will, however, decrease the percolation rate due to the formation of cement gel in the void space. The data in figure 3-24 shows typical limits of this effect for two backfills with similar uncemented percolation rates. Herget[34] reported that the effect of cement on the drainage behavior of a material where a material which had a percolation rate of 3.7 cm/hr showed a percolation rate of less than 1cm/hr after two days with a cement addition between 3-12% (Figure 3-25).

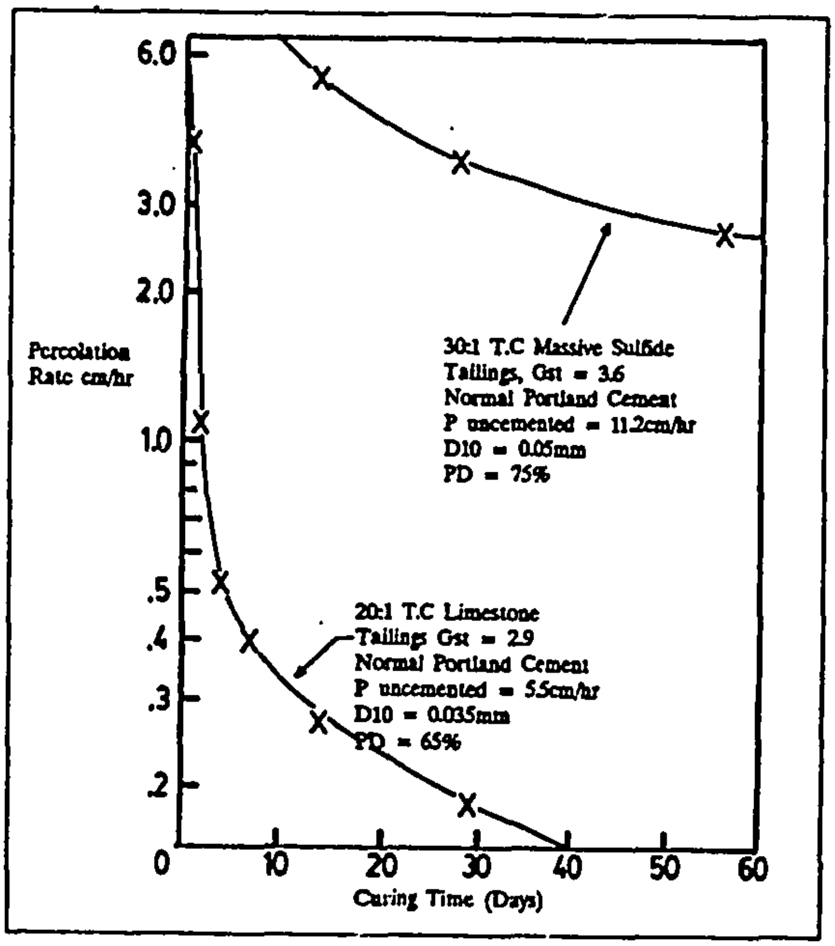



Figure 3-24 Percolation in cemented tailings. [Mitchell 1979]

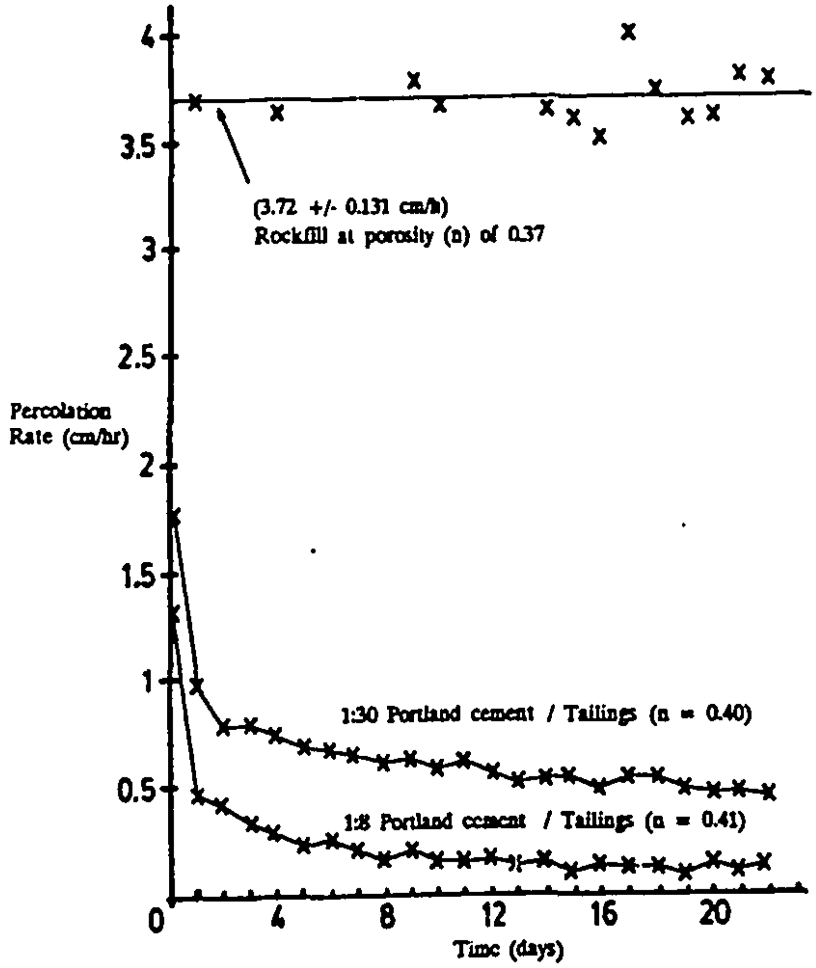



Figure 3-25 Change of percolation rate with time at 100% saturation. [Herget 1981]

### 3.3.4.3 Effect of fine particles

The percolation rate has a very close relation with the content of fine particles in tailings. With the increase in content of fine particles, the percolation rate drops quickly. The factors influencing the percolation rate would be concluded into three parts: Porosity (particle shape included), constituent of particles, and composition of mineral. The relationship among percolation rate, the content of fines, and porosity has been analyzed by data from laboratory tests using the tailings from four gold mines which is shown in Figure 3-26.

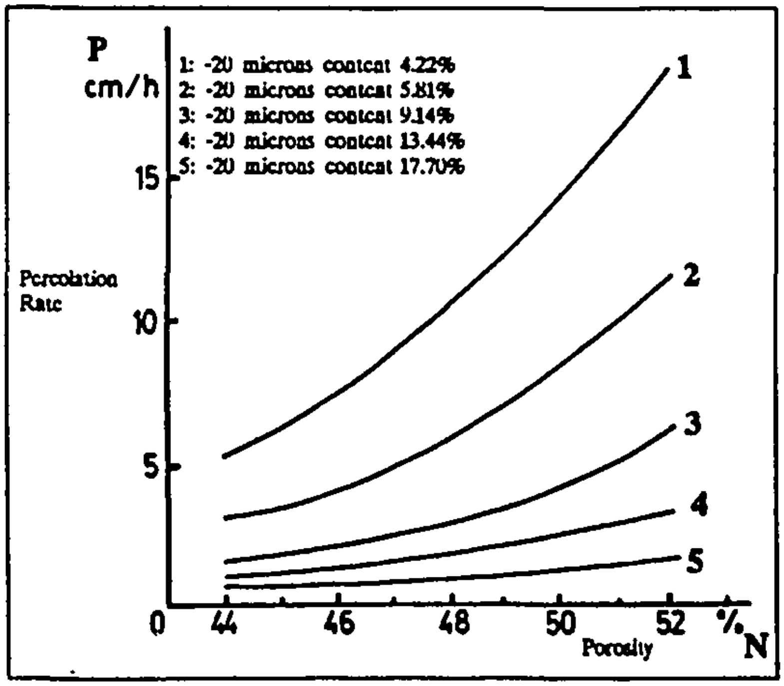



Figure 3-26 P-N Curve of Tailings [21]

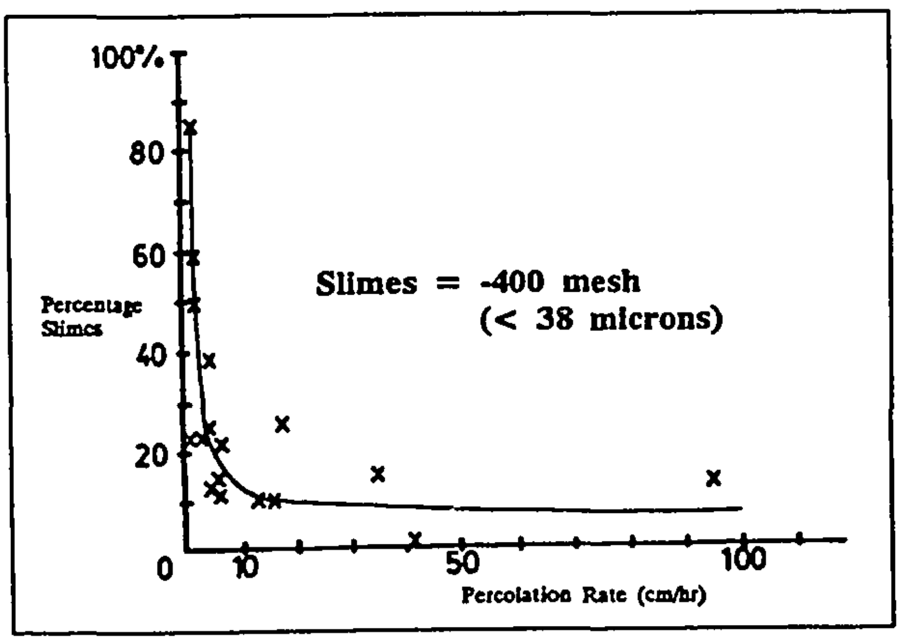



Figure 3-27 Relationship between percentage slimes and percolation rate. [Herget, 1981]

It is clear that the permeability is reduced as the percentage of fines increases. Slimes are generally called the size fraction of -400 mesh or grains with a size less than .038 mm [Herget. 1981]. Figure 3-27 shows a plot of percentage slimes against percolation rate from various standard percolation rate tests. The percolation rate drops significantly as the slime portion increases above 10%.

### 3.3.5 Liquefaction Potential of Paste Backfill

Liquefaction is a phenomenon wherein the shear resistance of a mass of soil (backfill) decreases when subjected to monatomic, cyclic or dynamic loading at constant mass. It can occur in saturated tailings, sand, silts and quick clays. When liquefaction occurs, the body will behave as a fluid with a mass twice as that of water, resulting in very high hydrostatic pressures on retaining walls and great potential danger should the soil be able to 'flow'.

#### 3.3.5.1 Evaluation of Liquefaction Potential

a) An evaluation procedure based on the concept of "steady state of deformation" was proposed by C.S.J. Poulos et al[36]. On liquefaction the mass undergoes very large unidirectional shear strain - it appears to flow - until the shear stresses are as low or lower than the reduced shear resistance.

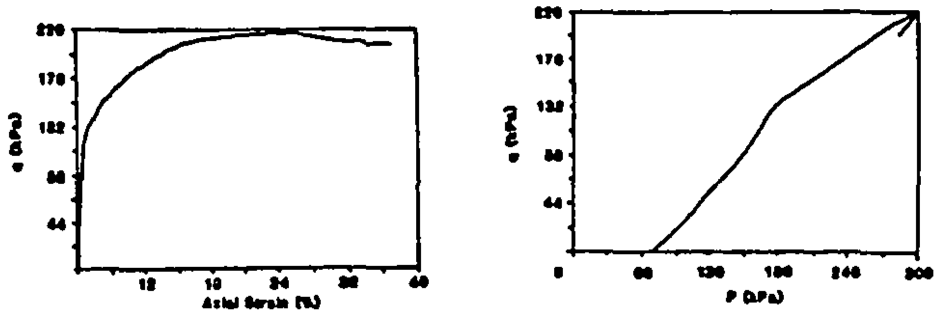
b) A second procedure is based on the cyclic stress or strain conditions likely to be developed in the field by a proposed design earthquake. These stresses and strains are then compared to those observed to cause liquefaction in representative samples in laboratory tests. Such results can provide results permitting an assessment of the soil behavior under field conditions [37].

The first procedure suggested by Poulos[38] et al has been widely employed; where a fill mass sustaining static shear stress greater than the steady state undrained shear strength is susceptible to liquefaction. If paste backfill is potentially unstable, it is also necessary to consider those possible seismic events that could cause liquefaction.

#### 3.3.5.2 Liquefaction Evaluation Analyses

According to the suggested method from Polous et al, (1985), a series of consolidated undrained triaxial tests was conducted on reconstituted and undisturbed samples at various void ratios. Typical results of tests on reconstituted samples are presented in Figure 3-28[39].

These tests were carried out on saturated samples with void ratios ranging from 0.68 - 0.98 and different cement:tailings ratios, (1:30, 1:40, 1:50).



Stress-strain. Stress path.

Figure 3-28 Paste backfill sample at 30% water content and 69kPa pressure.

## 3.4. CONCLUSIONS

Physical mechanical properties of fill inherently relate to type of fill, its grain size composition and additives blended in fill mixture. A number of laboratory tests and in-situ investigations have been devoted to determine the inter-relations of basic parameters affecting of backfill and as a result of those activities, some empirical correlations interpreting quantitative relationship between factors and property indexes have been developed. As investigations and experiments indicate that the extremely important factors are, grain size and size distribution, type and amount of cementing agent, water content and curing time. Variation of any of those mentioned factors would result in significant change of mechanical properties of backfill. Since the potential material for backfill is of great variety, so the measurement of fill properties has become necessary to prove a real suitable fill.

The results obtained in experiments and investigations presented in this chapter are good references but not concrete solution for a given condition. To improve properties of fill mixture and backfill economy, further study with specified material is always encouraged.

# CHAPTER 4

# PREPARATION OF BACKFILL

## 4.0. INTRODUCTION

The fill preparation process is the first part of operation system specification which generally comprises fill composition and mixing solids with water to form a slurry before being transported and then placed in stopes. Solid materials selected for backfill must be prepared in fill stations according to the target properties of the fill and operation requirements.

Following the discussion of context diagram analysis, figure 4-1 illustrates the level-1 data flow diagram of process 3.1 (shown in figure 4a):

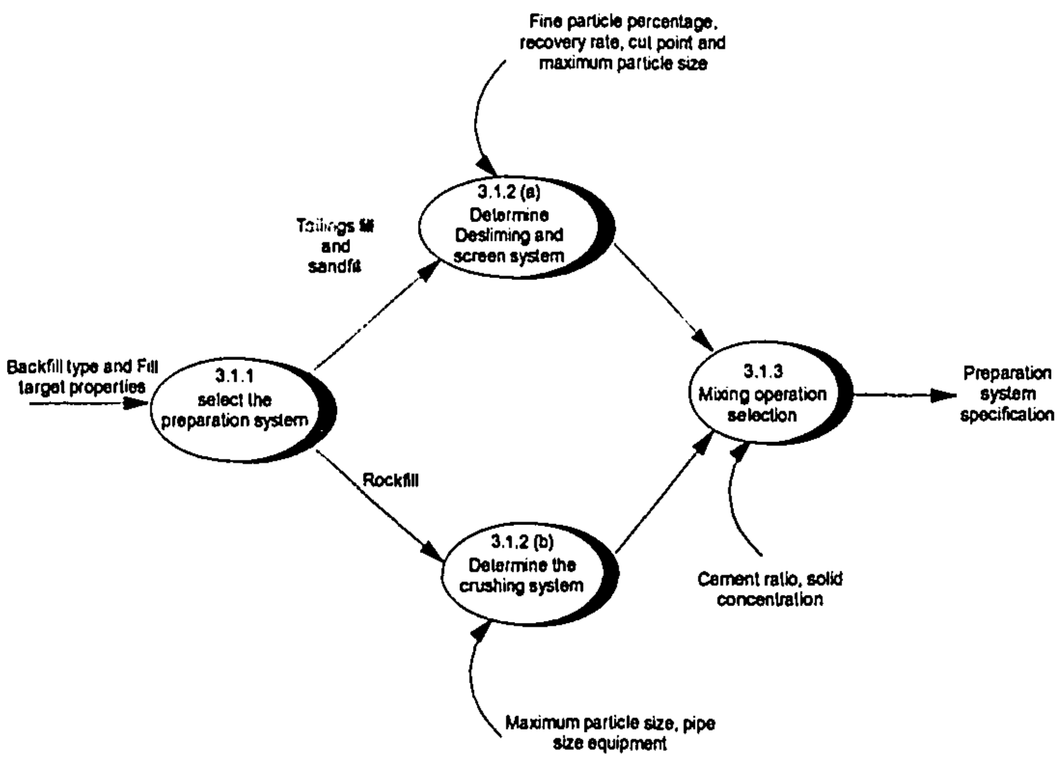



Figure 4-1 The level-1 data flow diagram of process 3.1

The process requests the information of backfill types and target properties of fill material specified in process 2 to determine the preparation system (shown in figure 4-1 as the process 3.1.1). The process may result in two possible conclusions: 1). rockfill preparation; and, 2). sandfill or tailings preparation, each of which requests different processes. For tailings or sandfill, process 3.1.2(a) requests information about fine particle percentage, recovery rate, cut point and maximum particle size to determine the desliming operation and screen operation. For rockfill, the process 3.1.2(b) requests information about maximum particle size, possible pipe diameter and handling equipment to determine the crushing process. Finally, the process 3.1.3 requests information about cement ratio and preferred solid concentration to determine the mixing operation. The overall process will produce the basic specification of backfill preparation operation. In the following section, the details of technology involved in the process presented in figure 4-1 will be presented.

## 4.1. CONSTITUENT AND SOURCES OF BACKFILL

Fill materials are generally mill tailings, sand, waste rock and binding agent, such as ordinary Portland cement, slag and flyash. Mill tailing comes from ore processing plants and then is classified before being stored in surge bunker.

Sand and waste rock can also be used for backfill, especially in the case of inadequate tailings, or deficiency in its required mechanical properties. Sandfill usually comes from surface alluvial sand basin by truck or railway. Rockfill is commonly provided from underground development, or nearby open pit or quarry. Smelter slag and flyash from either smelter or coal burning plants have also been widely utilized in backfill in recent years. This has brought an effective environmental control of the latter materials.

Some additives, such as flocculent, accelerator, retarder, are used in small amounts to perfect the fill mechanical properties. These additives are supplied by a chemical plant as chemical products. They are relatively expensive and therefore their use is limited at the present time.

## 4.2. PREPARATION OF BACKFILL

### 4.2.1. Preparation of tailing fill

Mill plant tailing is a fine solid material with average particle size about 100 μm. It can be used as a deslimed tailing or full tailing depending on percolation requirement. Tailing fill can be divided into six categories based on fine particle content (-10 μm) as follows[40]:

Table 4-1 Tailing fill Properties

| Mass fraction (%) (minus 10 μm) | Category |
|---|---|
| 0 - 10 | Very coarse |
| 10 - 20 | Coarse |
| 20 - 30 | Medium |
| 30 - 40 | Fine |
| 40 - 50 | Very fine |
| >50 | Extremely fine |

Classified tailing containing fines of 10 μm less than 20% by weight is commonly applied in backfill which has a percolation rate above 10 cm/h. The fine tailing is used only for paste fill, where its density reaches above 80% of solid by weight.

**4.2.1.1. Centrifugal Methods of Dewatering and Desliming of Tailings:**

Tailings from ore processing generally contains 30 - 40% by total weight fine particles of size 10 microns. This reduces the efficiency of stope drainage for hydraulic fill and results in possible liquefaction. Therefore it must be deslimed to reduce the presence of fine solids in tailings to less than 20% by weight. Desliming is usually performed using hydrocyclones, which may operate in various circuit combinations, or by a classifier (thickener) depending on size limitation and requirements of tailing recovery.

A. Hydrocyclone

Hydrocyclones were introduced in the 1950's as a popular classifying device and is now probably the most popular mechanical classifying method used in mining today (see figure 4-2). The hydrocyclone is physically small and mechanically simple with relatively low capital and operating costs. Tailings are pumped directly from a mill plant or disposal pond to hydrocyclones in a slurry with density of between 30 - 40% solids by weight. Solids are thrown to the outside of the cyclone and as the primary vortex travels down the conical shaped cyclone it is throttled and solids escape through the bottom, a secondary vortex carries the clarified flow in a tighter and faster flow up through the center of the cyclone. This acts as a second filtering system which increases the particle recovery. The fine particles are rejected as overflow to a disposal pond and the coarser particles are discharged as underflow with a density of above 50% by weight.

The final 'cut point' is generally defined by the acceleration of the secondary vortex. The 'cut point' is determined by means of a Tromp Curve, (Figure 4-3). Tc, the cut point, is defined as the point at which particles have an equal chance of going either with the overflow or the underflow is also shown in Figure 4-3.

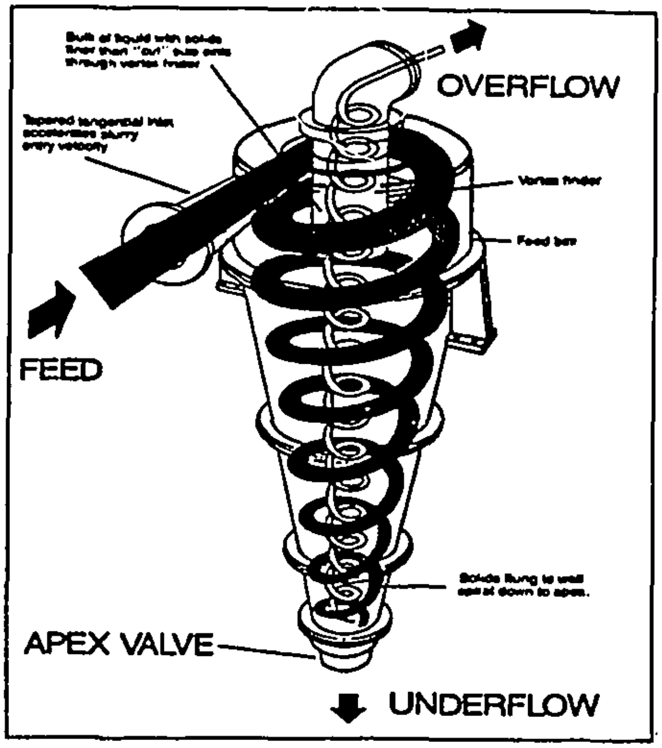



Figure 4-2 Hydrocyclone [41](Linatex™)

The cut-point can be shown to be theoretically dependent on the square root of the diameter.

$$d_c = \sqrt{\frac{\eta}{g.(\rho_s - \rho_l)}} \cdot \sqrt{\frac{X}{\lambda}} \cdot \frac{\sqrt{D}}{\sqrt[4]{H}} \qquad \textbf{4-1}$$

Note that the cut point is proportional to the fourth root of the pressure drop across the cyclone. Hence fine separation requires small hydrocyclones and it is found that in commercial applications the capacity dictates that many cyclones must be connected in parallel.

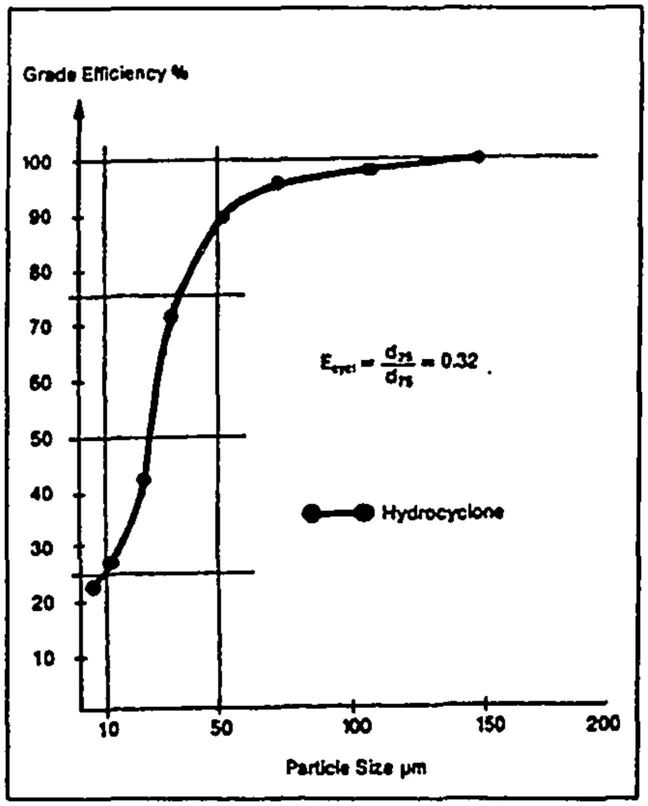



Figure 4-3 Tromp Curve (Linatex ™)

Grain size distribution of tailings classified by various cyclones may be different depending on the desliming process. For example, at Mount Isa, the two - stage cycloning of tailings at wet fill station produces hydraulic fill with a pulp density ranging from 68 to 72% solids by weight, depending on characteristics of the concentrator tailings[42]. The relevant size distribution is shown in Figure 4-4. This is suitable for both hydraulic and cemented fill. The specification for a backfill product meeting the requirements of support, drainage and fines retention is not unique, and a change in emphasis on each of these requirements results in a different specification of 2% to 7% is considered to provide acceptable backfill by various mines.

To optimize preparation of classified tailings in backfill, comprehensive research work on classification of mill tailings has been carried out at the South African Chamber of Mines Research Organization[43].

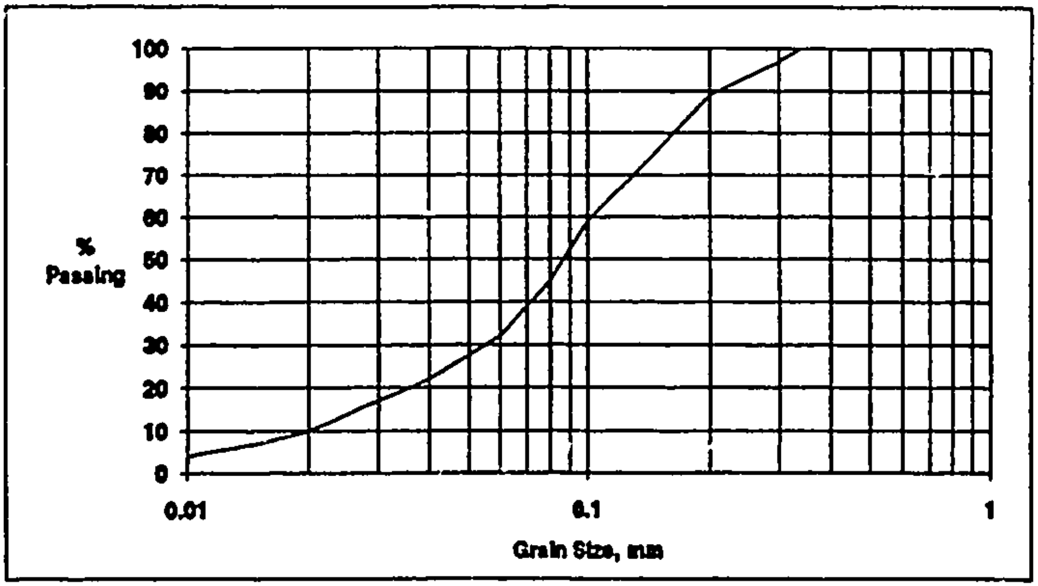



Figure 4-4 Grain size distribution of Mount Isa deslimed tailings

Test-work on small diameter hydrocyclones (100-150mm diameter) was performed. The performance of these small cyclones have been extensively studied and a computer model developed to describe the performance of these cyclones in various circuit combinations and under different operating conditions. The results showed much less variation in backfill recovery with respect to feed density and spigot diameter for the two-stage operation compared with the single-stage operation. The circuits used are shown in Figure 4-5.

Their principal disadvantage is that they are relatively inefficient in terms of accurate particle sizing, particularly in allowing fine slimes to be misplaced into coarse underflow stream. For fill preparation, where high throughput desliming with a sharp separation is required, this leads to the necessity of using large numbers of very small cyclones operating in parallel from a common feed manifold. Consequently it causes the problem of supervising and ensuring each cyclone in the cluster receives the same feed in terms of pulp density and size distribution.

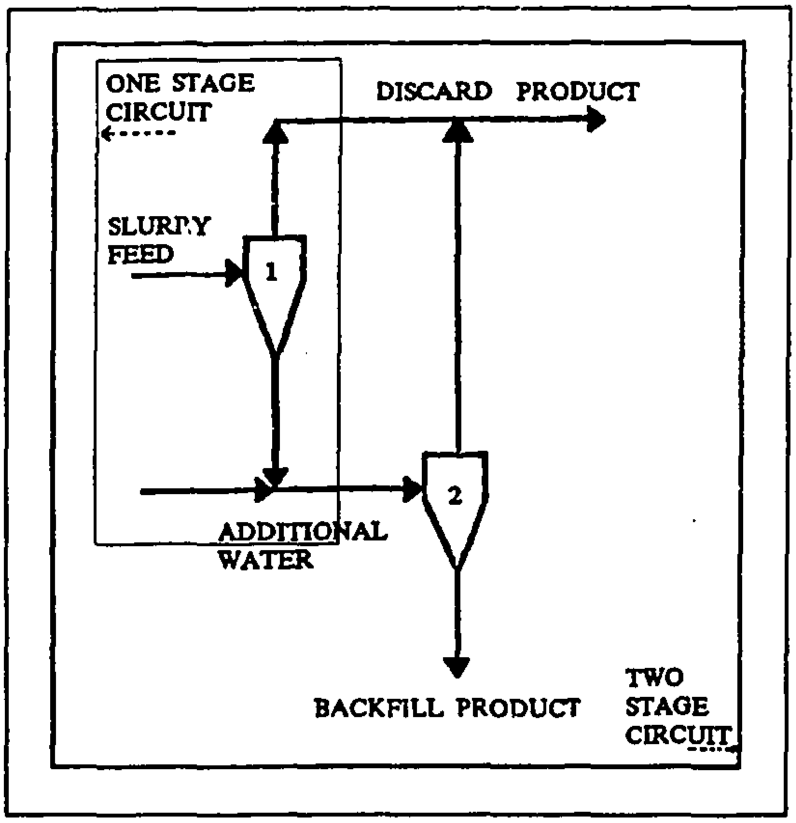



Figure 4-5 Single-and Two-Stage Hydrocyclone Circuits

B. Classifier/Thickener

Although cyclones are widely used for fill preparation, alternative methods of classifying mill tailing, such as various forms of mechanical classifiers, may be worth considering. Their advantages are in: (a) desliming application (particularly for coarse to very coarse tailing), (b) solids recovery to underflow (with fewer misplaced fines) leading to higher classification efficiency, and (c)lower operating costs compared with a multiple cyclone/feed pump arrangement, especially where very abrasive material is being classified. The principal disadvantages are: (a) higher capital cost, (b) larger floor area requirement, and (c) inability to achieve very fine cutpoint of small diameter cyclone required to deslime medium to extremely fine tailing. Feed to mechanical classifiers is around 30% by weight solids, and the deslimed underflow can be up to 80% by weight solids.

C. Separator

The Linatex Separator (Figure 4-6) differs from the normal cyclones in that the overflow is siphoned from the cyclone case allowing extra suction. The overflow is discharged through a rubber diaphragm which is kept pinched by the suction effect. The effect is to improve the separation and, more important in commercial applications, provide a uniform underflow density with varying feed rates.

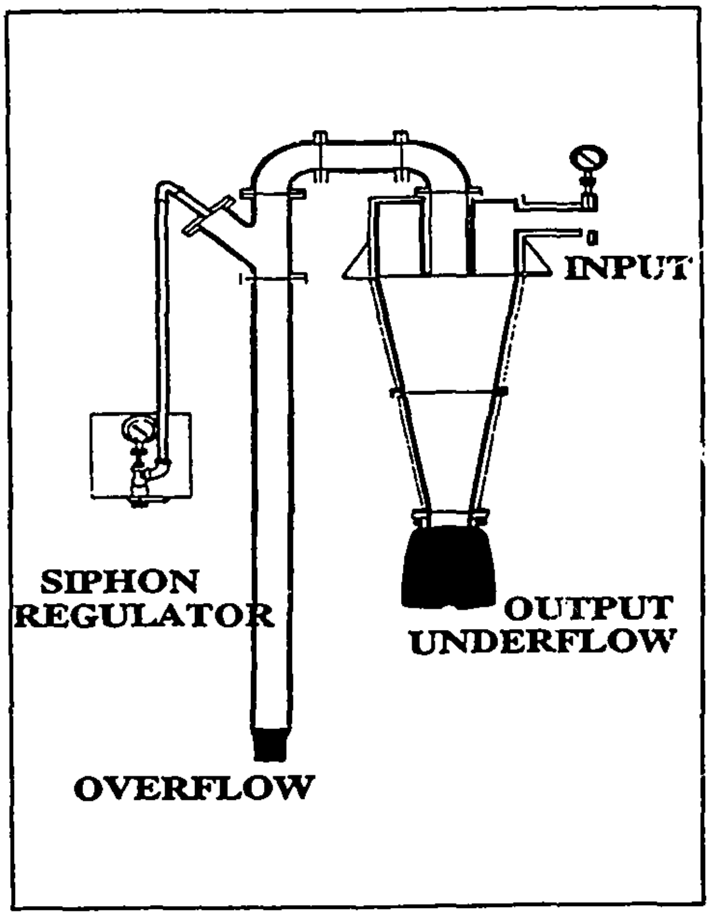



Figure 4-6 Linatex™ Separator (Linatex)

D. Centrifuges

The basic centrifuge is a horizontal hydrocyclone that mechanically rotates to induce the separation of solids and liquid. The rapid centrifugal acceleration is 5000-8000 times greater than the acceleration provided by gravity alone. There are two basic types of centrifuge applicable to the mining industry; 1). solid bowl centrifuge (counter current and concurrent design), and 2). screen bowl centrifuge. For the purpose of high concentration fill preparation, continuous solid bowl counter current models offer the maximum settling distances and quiescent unhindered sedimentation. (Shown in figure 4-7)

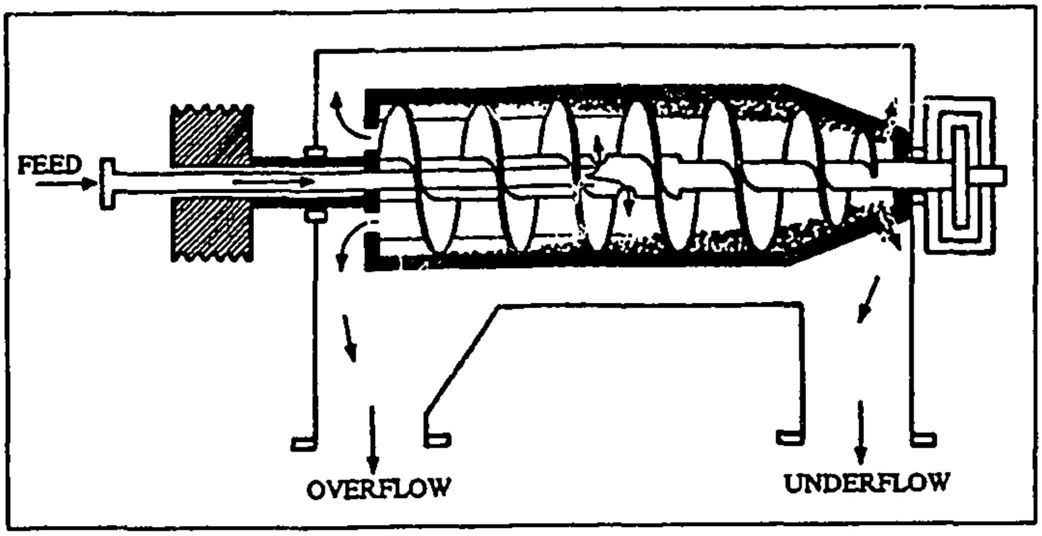



Figure 4-7 Continuous Sedimentation Centrifuge

E. Tailspinner centrifuge

Designed by Joy in the 70s, the tailspinner is basically a centrifuge which has been specifically manufactured for use in underground mines. The tailspinner enables hydraulic fill to be dewatered at the stope entrance to a very high concentration, i.e. a paste, (76 - 84 % by weight) where it is mixed with cement before final placement thus avoiding in-situ dewatering. The general arrangement for processing the fill, under both surface and underground conditions, is illustrated in Figure 4-8.

Slime is supplied from the rotary drum filters and is repulped to the required density between 1.65 and 1.70 gr./ml. The waste product containing particles of less than 44 microns is pumped to a waste disposal dam. With most of water and fine fraction now removed, the underflow material emerges as backfill consisting of 78% solids by weight and 58% +44 micron at a relative density between 1.9 and 1.98. The placement pump is a double acting hydraulic driven concrete placer type pump. It is rated at 8m³/hour at a pressure of 100 bars and has a maximum operating distance of 300 m.

According to Dome Mines Ltd. report, there are five tailspinners in operation on backfill production. Advantages are the mobility and sturdiness of the design. Disadvantages are the low capacity, low solids recovery with certain feed and a poor maintenance record.

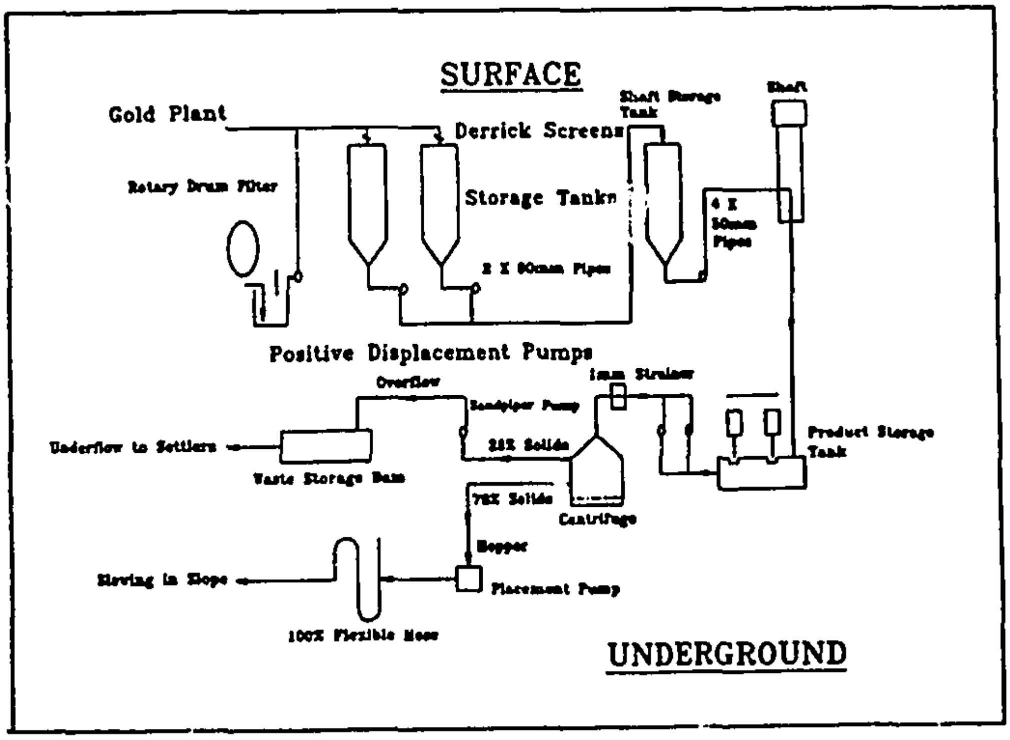



Figure 4-8 Schematic of a Dewatering Slime System [Close & Klokow, 1985].

### 4.2.1.2. Filter Methods of Dewatering and Desliming Tailings

Filter methods involve either mechanical or vacuum methods to separate solids and water via a filter cloth of some sort. Their use in backfill is uncommon due to the large capital costs involved. Filters are normally used in final dewatering of a cash product, i.e. ore concentrate. Recently, advances in the design of disk filter have made feasible underground filtration for backfill preparation more especially in paste production where concentration must be high and controllable. It is preferable to dewater a product beyond the paste design requirements to enable storing of material as a dry product and reliquefy to arrive at the correct concentration.

A. Pressure Filters and Belt Filters.

Pressure filters are large expensive items of machinery that are able to dewater most slurries up to concentration of 90 per cent by weight (shown in Figure 4-9). Various companies supply pressure filters involving hydraulic and mechanical pressure over filter cloth enabling clear filtrate and very high solids recovery.

The first continuous horizontal belt filter was the Landskrona filter installed at SUPRA, Sweden. This unit is still in operation and modern day belt filters are direct descendants of this first machine. The belt filter concept allows the cake formation and dewatering zones to vary in size as required for different requirements. More importantly it allows the separation of filtration zones and multiple cake washing with separate discharge of mother and wash filtrates which was the initial application of the Belt Filter. Performance in terms of solids recovery for both types of filter is excellent with some able to extract a clear filtrate from a 15 micron slurry. However, the capacity of all these machines is low, generally up to 15 tones per hour. The machines are also often large, cumbersome, and involve complicated mechanisms, and are expensive.

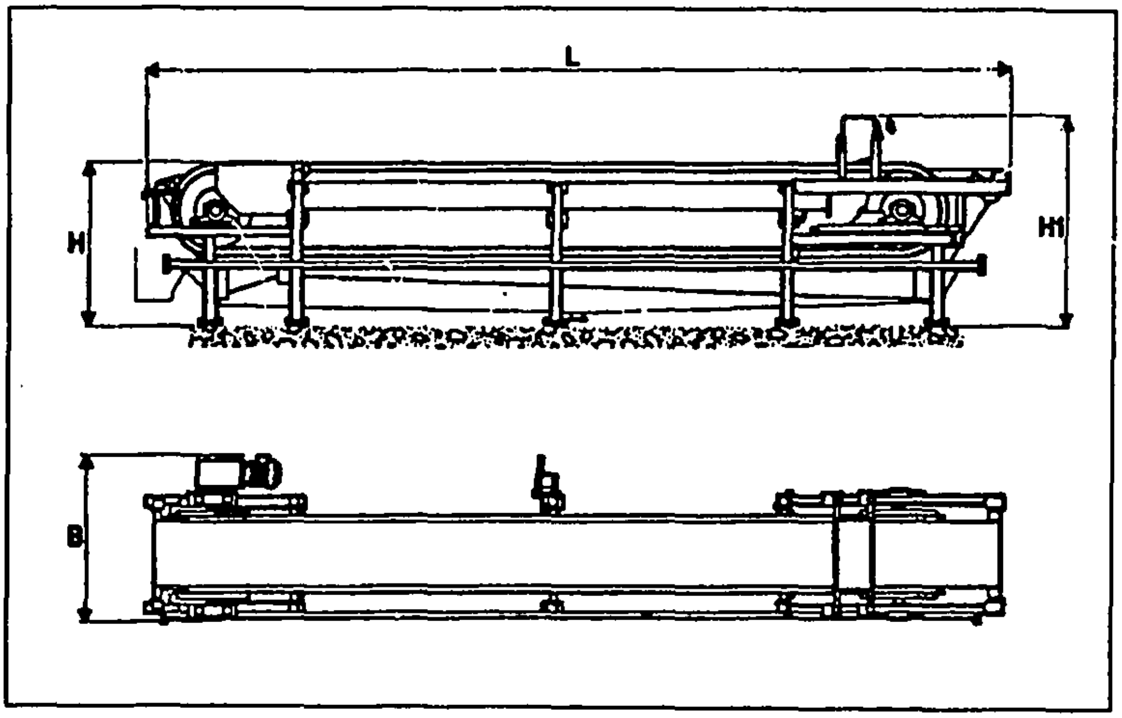



Figure 4-9 Belt Filter [44]

B. Disk Filters.

In the late 1980s a disk filter specifically for the production of "high density backfill" was developed[45]. The design is similar to the Dorr-Oliver disk filter shown in Figure 4-10. There is minimal maintenance required with cloths being the only consumable with nominal costs. The control and sealing disk should last two years with normal maintenance of 1/2 hour per day. There are 16 IMS disk filters currently in operation in South Africa. A very general guide to capacity is approximately three tons slurry dewatered per hour per square metre of filter cloth. Surfaces filters offer up to $84m^2$ and underground filters, $12m^2$.

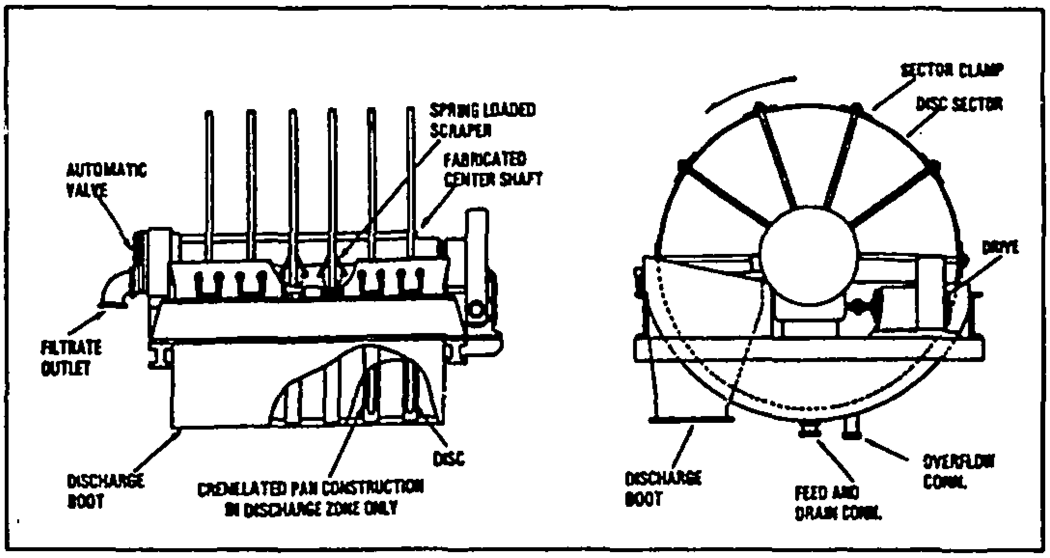



Figure 4-10 Disk Filter (Dorr Oliver)

C . Drum Filters

There are two types of rotary drum filter, vacuum and pressurized. The former is more common and offers a wide variety of solid-liquid separation requirements. A vacuum drum filter is used at Lucky Friday mine in Idaho to effectively dewater tailings prior to paste production. The filter is an old machine and represents the capacity limit for the paste system; as such their compatibility in paste production is proven and should be attractive to a paste system designer.

#### 4.2.1.3. Storage and Repulping of Tailing

Tailing is generally prepared prior to backfilling in double the quantity required for backfilling. It is stored either in a silo constructed on surface or underground (Figure 4-11) or simply in raises.

Most storage silos are made of reinforced concrete or steel and have a hemispherical or conical shaped bottom. A favorable ratio of height to diameter for tailing discharge from a silo is 2:1[46]. Alternatively a disposal pond can be used for storage of tailings.

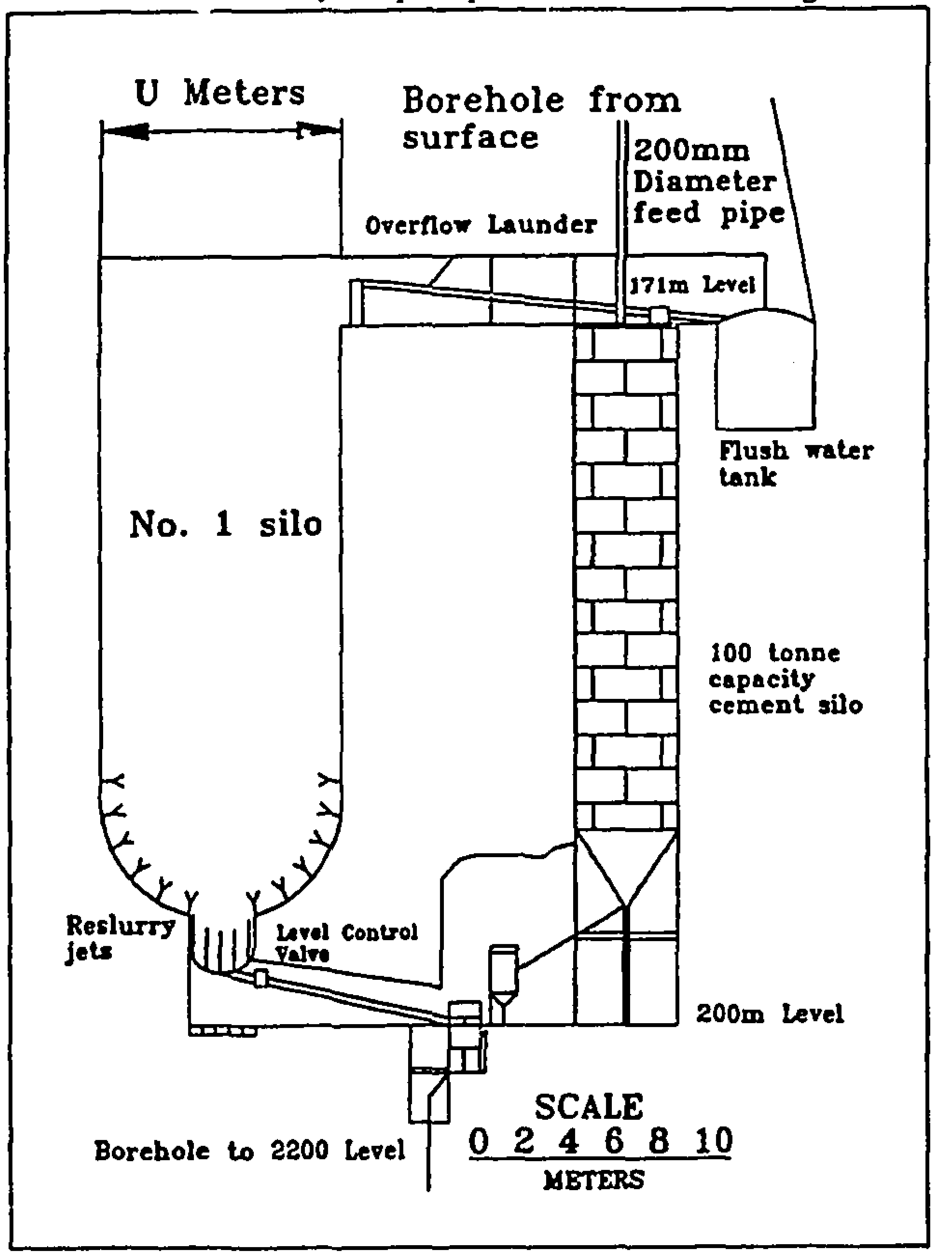



Figure 4-11 Schematic of an underground fill station in the Mufulira Mine, Zambia[47]

The tailings discharged from cyclones settle down in the silo and are compressed by its own weight. To make the tailings flowable and dischargeable under the action of gravity, a fluidization device such as a Marconaflo water jet is generally fixed inside the silo. The water jet disperses the thickened tailings, suspending particles in a slurry which is then discharged to mixing facilities. Such pulp has a density of above 70% solids by weight. The density of the discharged pulp is controlled by adjusting water flow via the discharge valve. The volume of pulp discharged can reach hundreds of cubic metres per hour, depending on the discharge hole size, water injected and silo geometry.

#### 4.2.1.4. Mixing cement with tailing

Repulped tailings flow down to a mix tank, where it is mixed with cement (binding agent or additives) from the cement silo. Silo storage of cement is generally temporary and in small quantities ranging from 50 to 200 tonnes, it is delivered by a screw feeder or conveyer to the mixing tank. The cement supply is measured by a weigher and controlled mechanically or automatically by changing speed of discharging device installed at the bottom of the silo.

To produce the uniform slurry required for backfill, a specific quantity of water is also added to the mixer. When additives are applied, a special container should be installed at the fill station which consistently supplies additives at a pre-set rate. If there are insufficient tailings, a combination of tailings with sand or crushed waste is applied. An example is the fill station at Strathcona Mine (Figure 4-12). It is designed to deliver 150 T/h of deslimed mill tailings and alluvial sand, with a ratio by weight of sand:cement at 30:1.

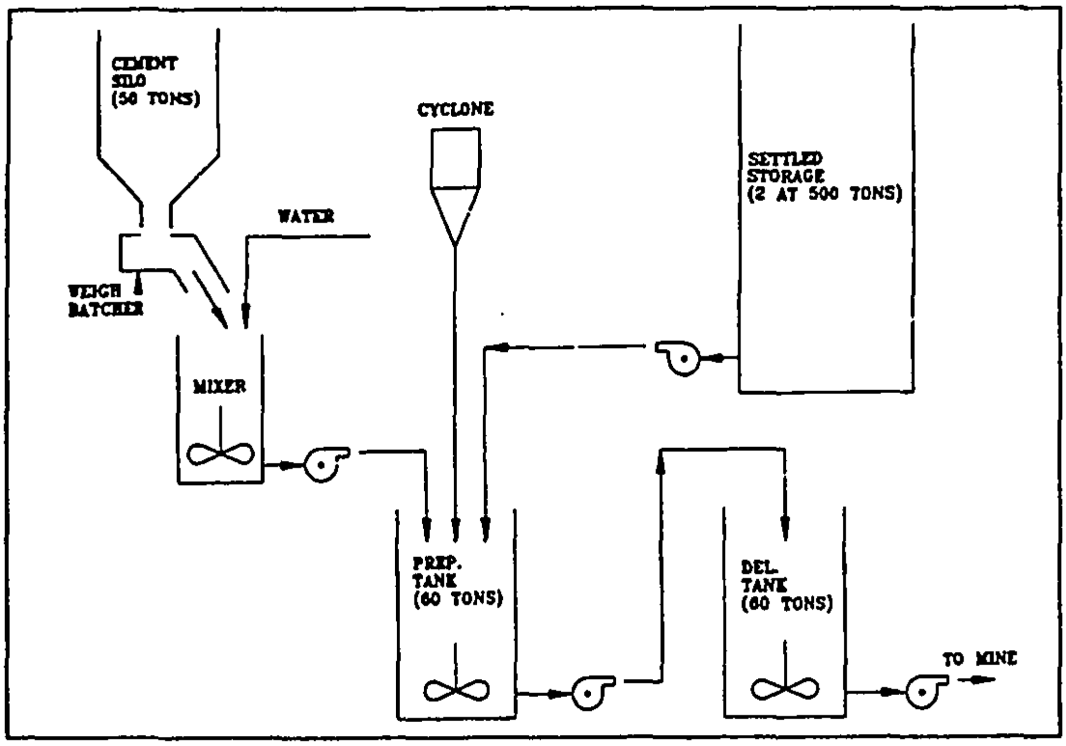



Figure 4-12 Backfill Preparation Plant Flowsheet[48]

### 4.2.2. Preparation of sandfill

The size of the sand used in backfill ranges from 0.05 to 4.76 mm, after screening. It is taken from an alluvial basin, transported by truck, railway or belt conveyor and stored temporarily in a bunker. The sand is then screened and passed through stationary or vibrating screens, before being directed toward a mixing chamber, where it is mixed with cement (if required). It is then transported by pipeline or truck to stopes. In practice, sandfill preparation can be carried out in different ways and the preparation station may be constructed on surface or underground or partly underground.

#### 4.2.2.1. Preparation of uncemented sandfill

The preparation station can be of various types. Depending on the operating requirements it may consists of the following parts:

- Bunker for sand
- Unloading equipment
- Equipment for pulping
- Water tank

Figure 4-13 shows a simple surface preparation station. Sand slurry is produced by a jet of water under pressure (>200 kPa). The nozzle diameter ranges from 15 to 30mm. Sand slurry prepared with a volume ratio of water to solids ranging from 2:1 to 1:1 is generally placed by 150 to 185 mm diameter pipeline.

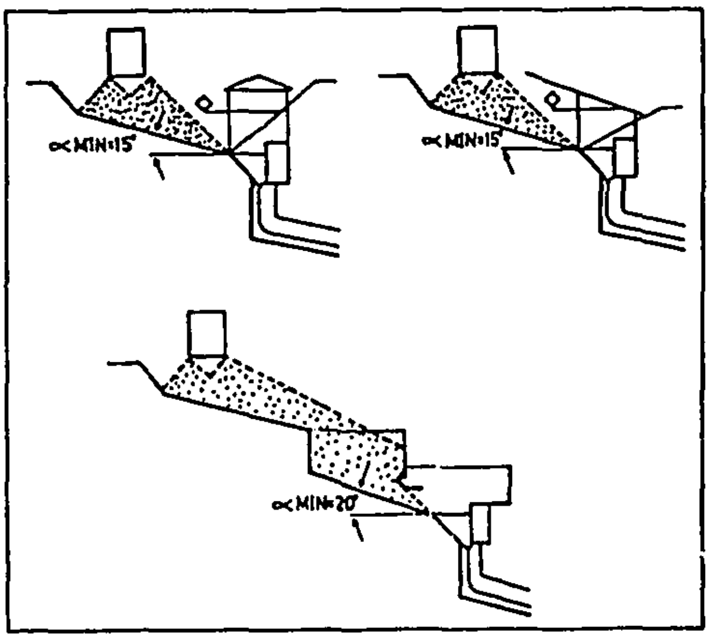



Figure 4-13 Simple surface sandfill preparation station[49]

#### 4.2.2.2. Preparation of cemented sandfill

In this case, cement is added at either a surface fill preparation station or an underground mixing station, or even near the stope to be filled. Cement may be added as dry solid or slurry. At Kidd Creek Mine sand from a 150-tonne bin (screened to -3 cm) is fed into a 500-tonne mixing silo. A Marconajet mixing system slurries the alluvial sand in the bottom of the tank at a rate of 200 tph at a 65% pulp density. The sand is sent underground via boreholes and 15 cm diameter A.B.S. plastic pipe. The pipe network follows the drifts and raises, in a manner similar to the cement slurry line. Near the opening to be filled, the sand line and the 10-cm diameter pipe containing the cement slurry pump join, and the mixed hydraulic fill is sent into the stope through a 200-mm borehole. Figure 4-14 features a mechanized surface station.

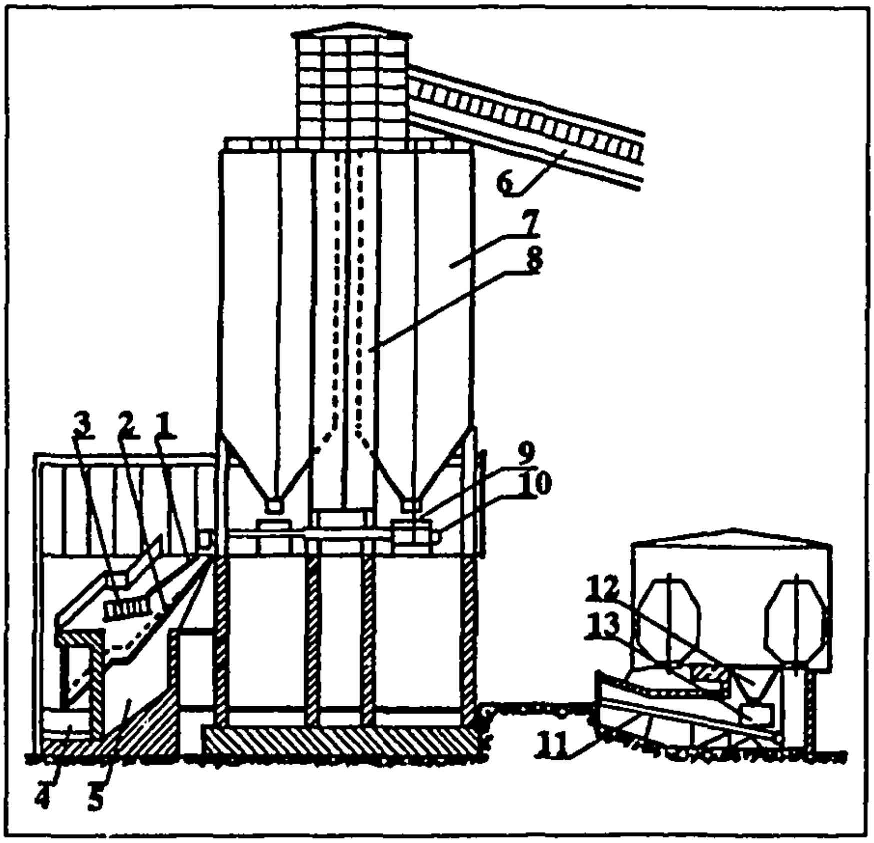



Figure 4-14 Mechanized Surface Sandfill Preparation Station [47]

Key: 1. screen 2. chute
3. vibration screen 4. pipe
5. hopper 6. conveyer delivery system
7. sand storage silo 8. water tank
9. weigher 10. feeder
11. belt-conveyer 12. discharge hopper
13. feeder

The volume of bunker for sand (V) is defined as follows:

$$V = a.\ Q_d \qquad \text{4-2}$$

where: $a$ = proportionality coefficient.
$Q_d$ = maximum daily sand requirement, ($m^3$).

The coefficient $a$ depends on maximum daily sand requirement ($Q_d$), transport distance (L) and average rate of backfill system. It is given in Table 4-2.

Table 4-2 Determination of the proportionality constant (a) to be utilized for calculation of the bunker volume [Adamek, 1980].

| $Q_a$ $m^3/h$ | $b$ | $Q_d$, $m^3$ | | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | 600 | 1000 | 2000 | 3000 | 4000 | 5000 | 7500 |
| | | a | | | | | | |
| 100 | 1,0 | 1,0 | 0,70 | 0,45 | 0,20 | 0,20 | 0,16 | 0,107 |
| 330 | 1,0 | 1,0 | 1,0 | 0,80 | 0,67 | 0,55 | 0,55 | 0,44 |
| 500 | 1,0 | 1,0 | 1,0 | 1,0 | 0,73 | 0,69 | 0,65 | 0,55 |

Key: $Q_a$ = Average rate of backfilling, $m^3/hr$.
b = Coefficient dependent upon the fill delivery distance between the preparation station and the stope (L).
For L<10, km. : b = 1.0
For 10<L<20, km. : b = 1.15
For L>20, km. : b = 1.25

### 4.2.3. Preparation of rockfill

Waste rock from underground development is usually dumped into empty stopes and, as such, constitutes the cheapest backfill method. In most cases, rockfill consists of a conglomerate of various particles with wide size ranges. The porosity of rockfill is largely dependent on the size distribution which affects drainage and support effectiveness.

To improve support capability, and meet delivery requirements, waste rock must be crushed to an optimized grain composition. As a general rule, the major portion of a rockfill consists of medium grains and only minor parts are left for large and small grain sizes. If the waste rock from underground operations is not adequate for the required rockfill, the rest must be taken from the nearby surface mines or quarries. The rockfill preparation alternatives can be numerous depending on local rock sources and ore body geology.

At Mt. Isa Mines, tests showed that Kennedy Siltstone was suitable for cemented rockfill. An outcrop was located to the north of the mine and a quarry was developed for the production of backfill. Figure 4-15 shows how the quarry was integrated into the mine.

After choked draw through the rockfill pass, the siltstone is degraded to 20 per cent. passing 25mm. The rockfill and cemented hydraulic fill are usually introduced into the same pass and placed concurrently at ratios ranging from 1:2 to 3.5:1 rockfill to cemented fill; depending on pass location, shape of stope and position of adjacent pillar mining. Pours of up to 48 hours duration have been experienced, but 18 to 24 hours are more typical. The system permits change of fill placement location at short notice.

At Kidd Creek Mines, approximately 80% of the fill placed has been consolidated material, consisting of cementing agent and aggregate crushed to a maximum size of 15 cm. Alluvial sand has been added in selected fills to increase the fine fraction, thereby acting as a void filler. The remaining 20% of the placed fill was unconsolidated rockfill or sandfill. Waste rock from the open-pit phase is the primary source of fill material.

Two methods of filling a stope with rockfill are: 1) to fill the stope with unconsolidated fill materials and then consolidate the outer edges with cement sand slurry for future pillar recovery, and 2) to pour the cement slurry on the rockfill material in a mixing culvert as it leaves the conveyor belt before entering the stope. The first method is suitable when a high degree of segregation has occurred, and where by consolidating the coarse particles at the walls of the stope, ore dilution can be minimized. The second method results in a much more competent and uniformly distributed backfill.

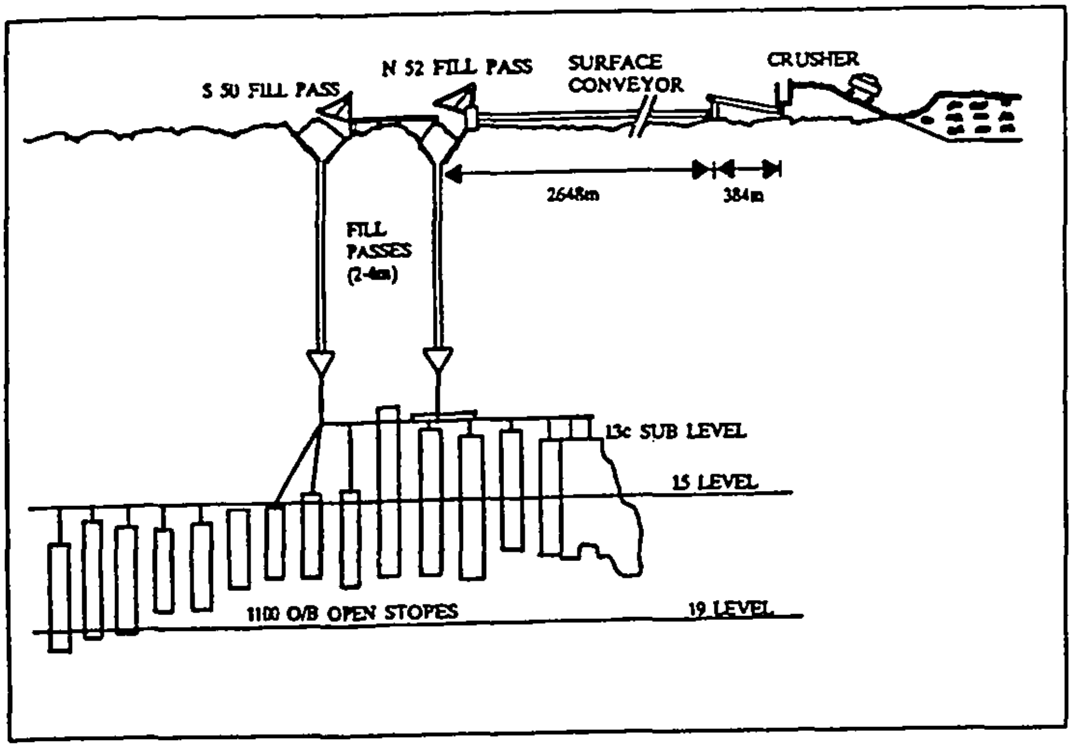



Figure 4-15 Rockfill system in Mount Isa Mine [Neidorf, 1983].

The key for producing a competent consolidated fill is to thoroughly coat all the aggregate with the supplied amount of cement slurry. If the material is not coated with slurry during the mixing process, it may never be properly coated since: 1) it is impossible to control the flow of slurry in the stope, 2) the slurry does not flow uniformly over the entire backfill cone, 3) the percolation rate of the slurry is variable due to differential settling and 4) slurry which is not actually used to coat the aggregate acts as a void filler. Since there is insufficient cement to fill all voids, some portion of the fill may remain unconsolidated.

## 4.3. ECONOMICS OF BACKFILL PREPARATION

There is a large variation in the design of fill preparation station. In general, the total cost for fill preparation station, constructed on the surface of an underground mine, consists of capital and operating costs. The salvage value is negligible. For rockfill the material is cheap but preparation costs and surface transport costs can be high. There is a large capital investment required to install a distribution system but the operating costs can be reduced by using conveyors underground rather than trucks. Hydraulic fill using classified tailings results in less preparation costs (if sandfill is used it would depend on the distance from source to preparation plant). High operating costs are incurred from cement usage and bulkhead construction. The distribution system is not expensive. Paste fill incurs the least operating costs with less cement consumption and no bulkheads required. Capital costs, however, can be high due to efficient dewatering and pumping.

### 4.3.1. Capital costs

This accounts for 75% (on average) of the total fill preparation cost over the life of the mine. Itemized capital costs breakdown and their estimation is given in Table 4-3.

Table 4-3 Estimation of Capital Cost per Item used in Backfill Operation

| ITEM | Purchase (f.o.b) (a) | Installation | | | |
|---|---|---|---|---|---|
| | | Direct | | Indirect | Contingency |
| | | Material (b) | Labour (c) | (d) | (e) |
| | | {0.15 x (a)} | {0.15 x (a)} | {0.3x(b+c)} | {0.1x(b+c} |
| Hydrocyclone system | 75,000 | 11,250 | 11,250 | 6750 | 2,250 |
| Screw Conveyor | 2,500 | 375 | 375 | 225 | 75 |
| Mixer Conveyor | 40,000 | 6,000 | 6,000 | 3,600 | 1,200 |
| Mixer | 9,000 | 1,350 | 1,350 | 810 | 270 |
| Water tank | 9,000 | 1,350 | 1,350 | 810 | 270 |
| Pipes, boreholes | 100-200,000 | 22,500 | 22,500 | 13,500 | 4,500 |
| Cement/back-fill silos | 150-200,000 | 26,250 | 26,250 | 15,750 | 5,250 |
| Water pump/drive | 8-15,000 | 1,725 | 1,725 | 1,035 | 345 |
| Elec. Inst, Eqp & Monitoring | 40-150,000 | 14,250 | 14,250 | 8,550 | 2,850 |
| Backfill Plant | 350-900,000 | 93,750 | 93,750 | 56,250 | 18,750 |

### 4.3.2. Operation costs

This accounts for approximately 25% (on average) of the total fill preparation cost over the life of the mine. The direct operating expenses for fill preparation comprise the following items: electric power, services, maintenance, parts (or material) and labor.

Electric power

The electric power cost ($C_p$) in terms of \$/tonne of dry fill prepared in the fill station is expressed as follows:

$$\sum_{i=1}^{n} N_i \cdot T_i \cdot \frac{C_e}{Q_f} \qquad \text{4-3}$$

where: $N_i$ = motor power consumed at the operation section i, W.

$T_i$ = work time in operating section i per year h/y

$C_e$ = cost of electricity

$Q_f$ = dry fill processed per year t/y

n = number of operating sections.

Material costs for service, maintenance, repairs and spare parts:

This is generally estimated as between 2 and 3 % of the capital cost p.a.

Labour cost = direct labor cost + indirect labor cost.

(A) Direct labor cost for fill preparation ($C_{DL}$) in \$/tonne of dry fill can be estimated by the following equation:

$$C_{DL} = \frac{n \cdot C_l \cdot t_f \cdot D_w}{Q_f} \qquad \text{4-4}$$

where: n = number of the fill preparation attendants.

$C_l$ = hourly salary paid to each fill preparation attendant, \$/hr.

$t_f$ = duration of fill preparation operation, hr/day.

$D_w$ = number of working days in a year, day/year.

$Q_f$ = dry fill prepared in a year, tonne/year.

(B) Indirect labor cost for fill preparation. ($C_{IL}$) = labor fringe benefits + bonus + incidentals = 30 to 50% of the direct labor cost.

# CHAPTER 5

# TRANSPORTATION SYSTEM FOR HYDRAULIC FILL

## 5.0. INTRODUCTION

As one of the most important procedure in backfill operating system, the transportation approaches commonly employed in the practical mining operation are listed as below:

1. Hydraulic transportation system
2. Pneumatic transportation system
3. Conveyor transportation system
4. Truck transportation system

So far, a large amount of researches dealing with the hydraulic transportation system has been conducted and the related results have been published. It is obvious that the rules established for hydraulic transportation system have made it possible that the design procedure can be computerized for feasibility studies and economical evaluation. Unfortunately, the researches and publications dealing with other transportation systems are not likely to enable the computerization to any substantial level for feasibility studies and the economical evaluation. So they are just principally discussed in this section, and the main efforts are given to the hydraulic transportation system. Hydraulic transportation has been widely applied to backfill distribution in underground mines since the early sixties. Its main advantages over other delivery systems in backfill are low delivery cost and energy consumption; relatively high transport capacity and continuity in operation; high packing density after placement, which is in favor of ground control. In most mines utilizing backfill, the fill is distributed as an aqueous slurry through pipeline incorporating both horizontal and vertical components. The flow is maintained either under the action of gravity or with the aid of pumps. Shown in figure 5-1 is the level-1 data flow diagram of process 3.2.

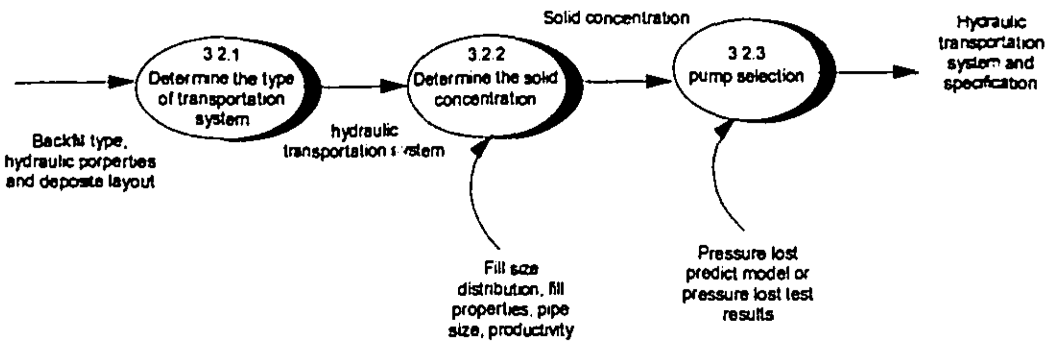



Figure 5-1 The level-1 data flow diagram of process 3.2 of figure 4a

The process requests the information of the backfill type and target properties of fill material specified in process 2 (see figure 2-8) to determine the type of transportation system which is shown in figure 5-1 as the process 3.2.1. In this analysis, level-1 data flow diagram of process 3.2 present only the information related to hydraulic transportation system. The process 3.2.2 requests the information related to fill size distribution, mechanical properties, pipe diameter and productivity requirement of the system to determine the solid concentration of the system. The process 3.2.3 requests the pressure loss prediction model or loop test results of the system to calculate the energy balance and determine the pump, if needed, for hydraulic transportation system. The whole process provide the overall specification of the hydraulic transportation system.

**1. Three Basic Configurations**

There are three possible configurations for moving the fill material from a point on the surface to the stope underground as shown in Figure 5-2 [2] .

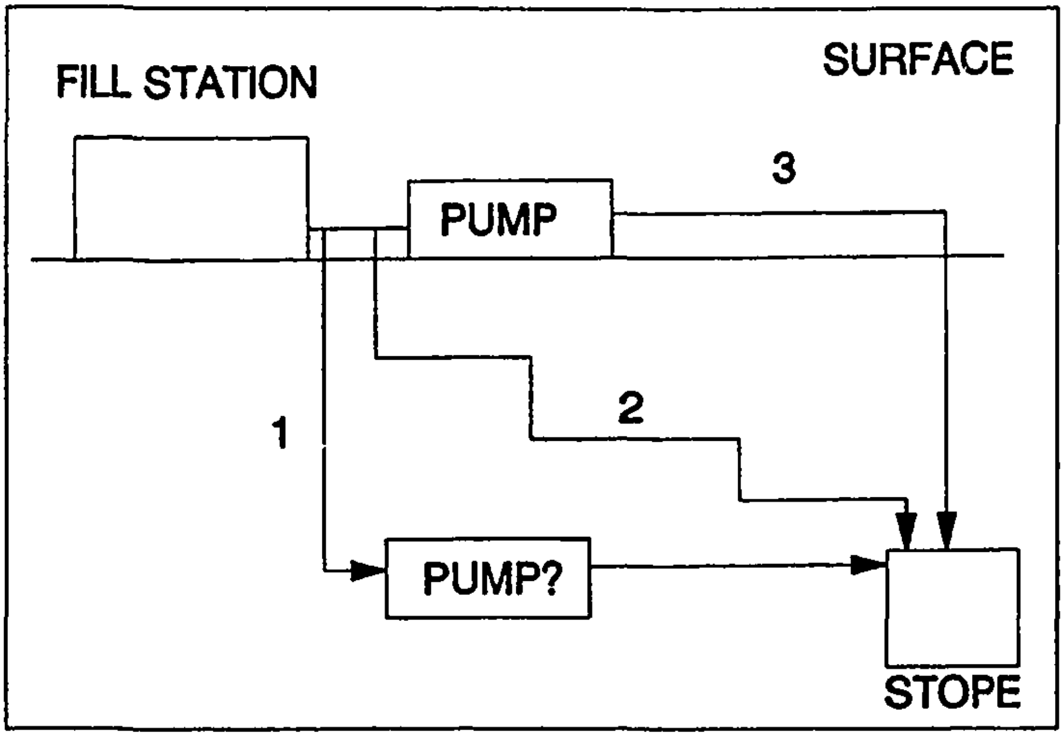



Figure 5-2 Basic Configurations for Fill Placement Systems

The first configuration has the advantage of being totally contained underground, thus causing no disruption to surface activities. Furthermore, the ratio of the vertical to horizontal distance is usually so favorable that little or no pumping energy is required.

The disadvantages of such a circuit become apparent when the ratio of the vertical to horizontal distance is relatively large or small. The first case is encountered in deep mines where the stope to be filled is close to the vertical drop section of the pipeline. This results in very high pressure at take-off point, and a burst line may disrupt the shaft level or main level operations. In the second case, a pump may be needed to convey the fill in the horizontal section of the pipeline with the incurred additional energy and maintenance costs.

The second configuration has the advantage of making the conversion from vertical head to horizontal pressure progressive, thus shorter and lighter pipes can be used. The pressure at take-off points are moderate and line failures, if any, do not disrupt the main shaft or main level of operation. The circuit can be developed progressively as the mine expands. The disadvantages of this configuration are in terms of increased maintenance costs resulting from the stepwise pipeline paths.

The third configuration has the advantage of easy installation, inspection and maintenance, with no special underground level and no disruption of the main shaft. However, such a system makes the filling operation dependent upon a pumping operation and requires a long borehole to place fill underground which results in high pressure take-off point. Furthermore some disruption to surface activities is possible, and in very cold weather, freezing may be a problem.

**2. Energy Equation**

The Bernoulli equation for the conservation of energy as applied to continuous incompressible fill flow in pipeline may be stated as follows:

$$\frac{P_1}{\rho g} + \frac{V_1^2}{2g} + Z_1 + H_A - H_L = \frac{P_2}{\rho g} + \frac{V_2^2}{2g} + Z_2 \qquad \text{5-1}$$

Where: $\frac{P}{\rho g}$ = static head

$\frac{V^2}{2g}$ = velocity head

$Z_1$ = elevation head

$H_A$ = head added by pump

$H_L$ = head lost to friction

The assumption of the incompressibility of the fluid is usually valid; however, the continuity condition may not always be satisfied, especially in the free fall sections of the vertical pipes or boreholes. Free fall should be avoided as it can cause inlet static pressure below atmospheric and hence draw air into the fill line. Furthermore, 'pipe hammer' resulting from impact, as the flow joins the full flow section of the pipe, may cause accelerated wear of the pipeline.

## 5.1. BASIC DESIGN PRINCIPLES

When designing an hydraulic fill system, it is recommended to use as high solids concentration as possible in order to minimize the volume of water for fill transportation. The operating flow velocity should be as low as possible to minimize pipe wear, but high enough to keep coarser particles in suspension.

Critical velocity is usually introduced as a lower bound for the operating velocity below which deposition of solid particles forms a stationary bed indicating imminent plugging of the pipeline. As the solids concentration increases, critical velocity becomes less relevant due to the hindered settling tendency of the particles.

In order to reduce the effect of free fall on the flow behavior of the slurry, pipe diameter in the free fall region may be reduced or some method of restricting the flow should be devised. Reducing pipe diameter may offer a suitable and economical solution.

### 5.1.1. Flow Regimes in Horizontal Hydraulic Transport

Commonly, four flow regimes are recognized in the transport of solid-liquid mixtures. They are:

- Homogeneous (or pseudo-homogeneous) flow
- Heterogeneous flow
- Saltation
- Moving bed

In practice, a state of mixed regime flow is usually established where two or more flow regimes occur simultaneously at different zones of the pipe cross-section.

The prevalent flow regime depends mainly on the solids concentration, the operating flow velocity, and the particle size distribution. The pressure gradient along the pipeline is significantly affected by these parameters as illustrated in Figure 5-3 for a given pipe diameter and particle size of the solids.

Empirical equations which involve drag coefficient or free settling velocity are not considered reliable enough when used in connection with the high density fill. Instead, when treating the hydraulic fill slurry as a pseudo-homogeneous suspension, a simpler equation based on the friction factor at the pipe wall may be used. Thus, the head loss (in meter of slurry) due to friction is given by the Darcy-Weisbach equation given by:

$$H = \frac{fL}{d}\frac{V^2}{2g} \qquad \textbf{5-2}$$

Where: f = friction factor

L = pipe length (m)

V = flow velocity (m/sec)

d = pipe diameter (m)

g = acceleration of gravity (m/s²)

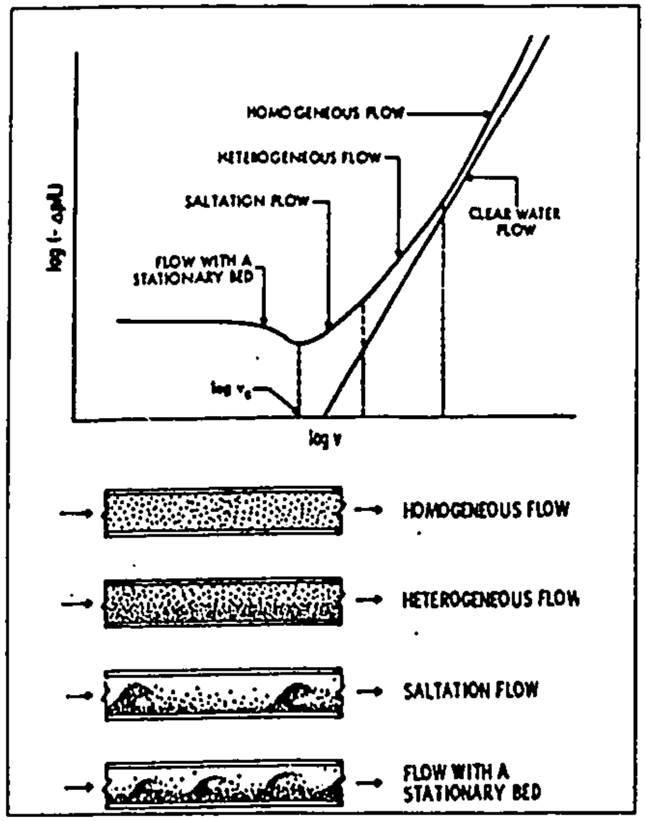



Figure 5-3 Pressure Loss vs. Mixture Velocity[52]

To determine the friction factor (f) from the Moody diagram given in Figure 5-5, the apparent viscosity ($\mu$) of suspension has to be determined since it enters in computing the Reynolds' number at which the friction factor is determined. Apparent viscosity can be determined from a plot of the pressure loss term, $\frac{PD}{4L}$, versus the nominal rate of shear term, $\frac{8V}{D}$. In laminar flow, apparent viscosity may be taken as the slope of the above-mentioned curve in accordance with the expression:

$$\frac{PD}{4L} = \mu \frac{8V}{D} \qquad \text{5-3}$$

Tube or rotational viscometers may be used for such measurements. The viscosity is a very difficult property to measure and is extremely site specific; dependent on the specific gravity, grain size distribution and particle shape to name just a few. Figure 5-4 shows an example of increasing viscosity with increasing concentration, the specific gravity of the tailings used was 2.8.

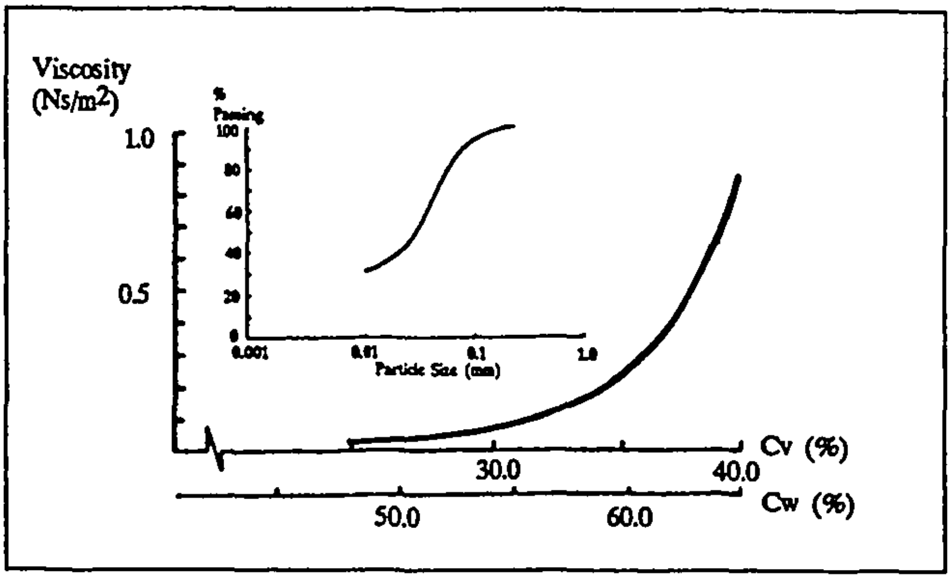



Figure 5-4 Viscosity vs. Concentration[53]

This approach is very approximate given the uncertainties with which the friction factor is determined. Such uncertainties may be traced back to the difficulty in characterizing the rheological behavior of a suspension by a single apparent viscosity, which varies with different flow velocity (shearing strain).

Instead of using the Moody diagram, other similar diagrams have been developed for slurries with well defined rheological behavior such as Bingham plastic or pseudo-plastic. For slurries characterized as Bingham plastics, the shear stress-shear strain equation is given by:

$$\tau = \tau_y + m_{pl}\frac{dU}{dy} \qquad 5\text{-}4$$

Where: $\tau$ = shear stress

$\tau_y$ = yield stress

$m_{pl}$ = plastic viscosity

The corresponding relationship between the pressure gradient and the flow velocity is given by the Buckingham equation expressed by:

$$\frac{PD}{4L} = 8\,\mu_{pl}\,\frac{V}{D}\left(1 - \frac{4}{3}\left(\tau_y\frac{4L}{PD}\right) + \frac{1}{3}\left(\tau_y\frac{4L}{PD}\right)^4\right)^{-1} \qquad 5\text{-}5$$

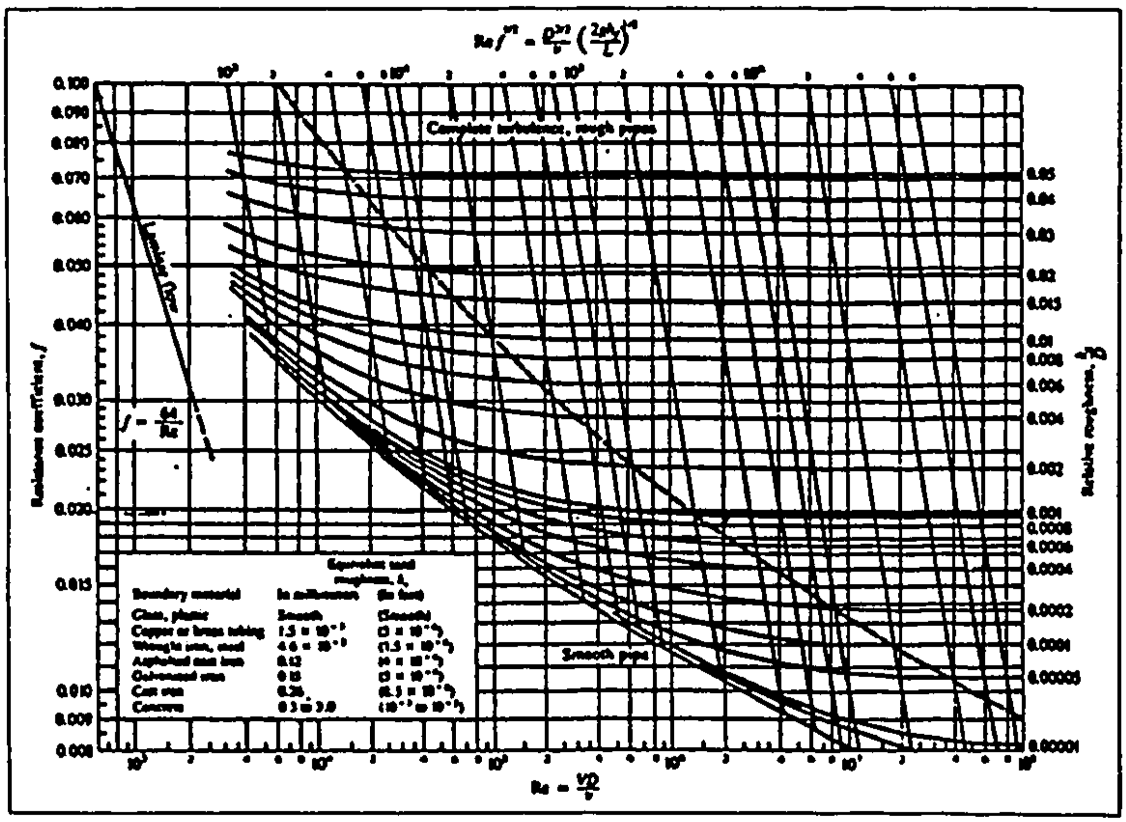



Figure 5-5 Moody Diagram

The friction factor for Bingham plastic suspensions as a function of the Bingham Reynolds' number and the Hedstrom number is given by:

$$\frac{1}{R_{eB}} = \frac{f}{16} - \frac{H_e}{6R_{eB}} + \frac{H_e^4}{3f^3R_{eB}} \qquad 5\text{-}6$$

Where: $R_{eB} = \dfrac{DV\rho}{\mu_{pl}}$

$He = \dfrac{\tau_y\rho}{D^2\mu_{pl}^2}$

The friction factor design chart for laminar flow of Bingham plastic fluids is found from Figure 5-6.

For slurries with a pseudo-plastic behavior (power law fluids), the shear stress versus shearing strain relation is given by:

$$\tau_w = K'\left(\frac{dU}{dy}\right)^{n'} \qquad 5\text{-}7$$

With the corresponding engineering equivalent given by:

$$\frac{PD}{4L} = K\left(\frac{8V}{D}\right)^n \qquad 5\text{-}8$$

Where K is the fluid consistency index and n is the flow index. The corresponding friction factor is a function of the generalized Reynolds' number defined by:

$$Re = \frac{D^{n'}\,V^{2-n'}\,\rho}{\gamma}$$

$$\gamma = K'\,8^{n-1}$$

The friction factor design chart for laminar flow of Power Law fluids is found from Figure 5-7.

Some fill material, at very high solids concentration, may exhibit dilatant behavior which is described by the same equation as the pseudo-plastic fluid except that the flow index (n) is larger than unity. Such flow characteristics would be so energy intensive to maintain that they are usually discounted as unpractical. When proper flocculents are added to such slurries, their flow behavior may be altered without changing their solid concentration, but this benefit has to be balanced against the added cost of such flocculent. Only a thorough economic analysis may determine the viability of such an approach.

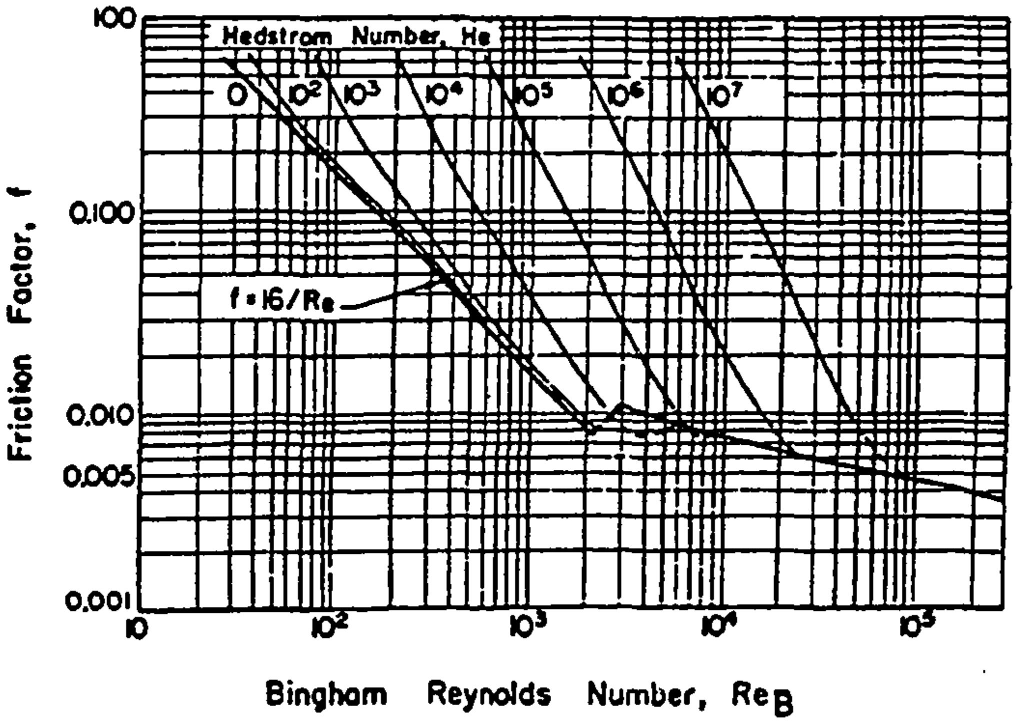



Figure 5-6 Design Chart for Bingham Fluids

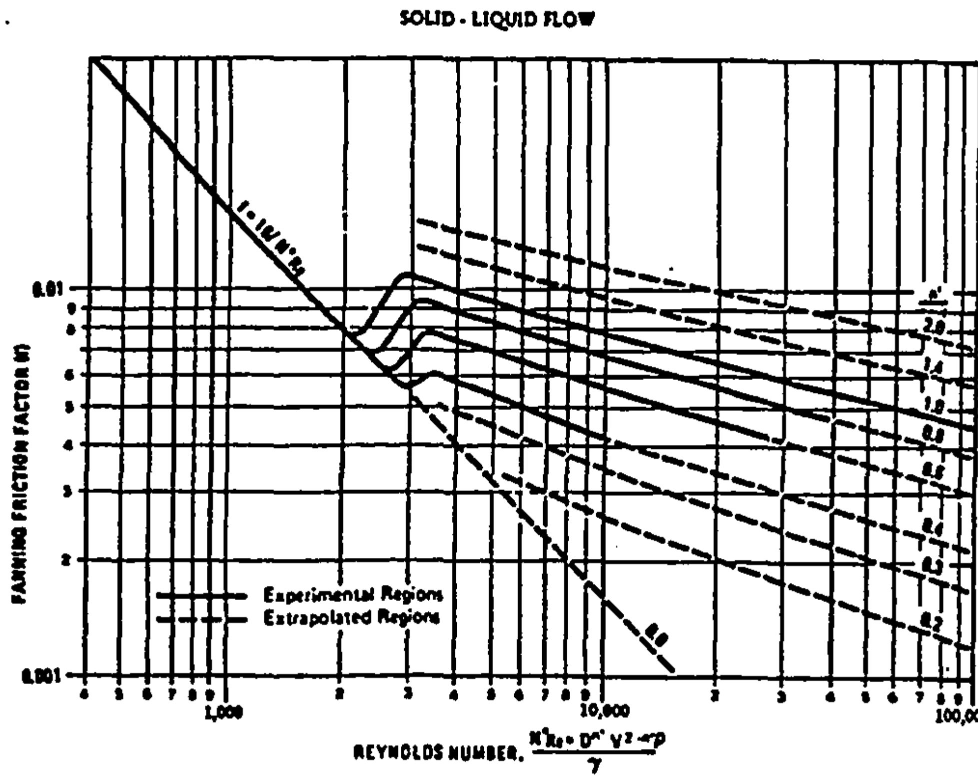



Figure 5-7 Design Chart for Power Law Fluids

### 5.1.2. The Design Approach of de Korompay

The theory introduced earlier is far from practical application in terms of backfill design. An alternative approach to the design of fill placement system is described by de Korompay[54] in his comprehensive review of hydraulic transportation system for mine backfill. Though mostly empirical, this approach is of practical value in the day to day operation of a mine filling system. The following is a critical review of this approach.

In his report, de Korompay analyzed the effect of solids concentration, critical velocity, corrosion and erosion on the transportation system. Formulas to calculate the friction head loss in horizontal and vertical pipes and boreholes were presented. Equations to determine the flow capacity of the vertical pipe and borehole in the free fall and full flow section were also proposed. The flow conditions in the horizontal and vertical pipeline were analyzed separately, then as a connected unit. The results of the study recommend using vertical flow lines with different pipe diameters. Substantial saving in pipe installation cost are claimed when following this approach.

De Korompay also introduced the concept of "specific pipe cost" which is used with the critical velocity to evaluate the economics of the delivery system. The selection of the most economical delivery system was discussed in relation to pipe characteristics (size, strength and thickness).

#### 5.1.2.1. Basic Parameters

The two basic parameters for the design of a fill transportation system are:

- The average daily fill requirement, A (Tons Per Day of dried solids, TPD)
- The fill delivery rate, T (Tons Per Hour, TPH)

The fill delivery rate is given by:

$$T = \frac{A}{t} = \frac{Q\,\rho_m\,C_w}{249} \qquad \textbf{5-8}$$

Where: A = average fill delivery rate, TPH of dry solids

t = operating hours of fill plant, hours per day

Q = Discharge rate ( USGPM)

$\rho_m$ = density of slurry ( $lb/ft^3$)

$C_w$ = solid concentration of slurry by weight (decimal fraction)

d = internal diameter of pipe ( in. )

The flow velocity V (ft/sec) is given by:

$$V = 0.409 \frac{Q}{d^2} \qquad 5\text{-}9$$

### 5.1.2.2. Solid Concentration of the Slurry

Solids concentration is either defined on a volume or weight basis. Most equations include the solids concentration by volume as it is the limiting factor for the maximum solid concentration selection. The following simple nomogram can be used to determine the relevant parameters.

As a general rule, it is suggested that a slurry concentration of 49.5 percent. by volume (approx. 73% by weight for an average tailings) be taken as the optimum for conventional hydraulic transportation of backfill, i.e. a settling slurry. Having defined this upper limit for solid concentration, a new parameter may be introduced which compares the selected solid concentration to the optimum value. This parameter is called the "relative solid concentration" and is given by:

$$B = \frac{\text{Cw (actual)}}{\text{Cw (max)}} = \frac{\text{Cv (actual)}}{0.495} \qquad 5\text{-}10$$

A nomogram for calculating the slurry concentration or specific gravity of the mixture is shown in Figure 5-8.

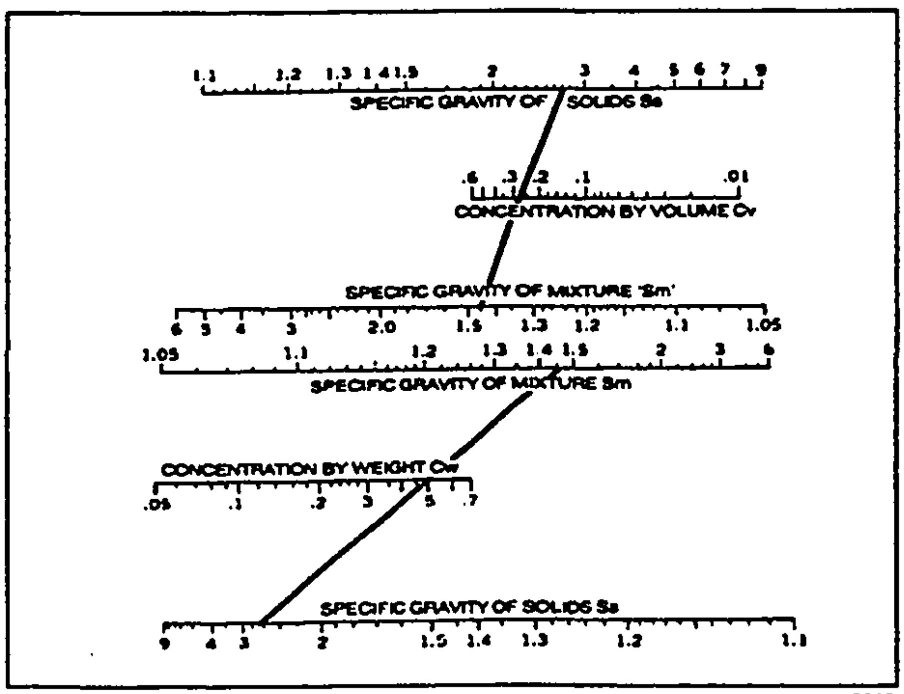



Figure 5-8 Nomogram for Calculating the Slurry Concentration[55]

### 5.1.3. Critical Velocity of the Slurry

Most suspensions exhibit some degree of settling which depends on the relative density of the suspended solid particles with respect to the carrying fluid as well as on their size and shape. Critical velocity may be defined as the velocity at or below which the solids start to form a sliding bed. This velocity usually falls close to the minimum point in the pressure - velocity curve for single sized particles. For multi-sized particles, this minimum point is much less pronounced . In practice, this flow regime is usually detected by visual inspection of a transparent section of the pipeline.

Although no single formula claims to predict the critical velocity for all slurries, the equation developed by Durand[56] is usually used as a first approximation for slurries of low solids concentration. This equations is given by:

$$V_C = F_L \sqrt{2\,g\,D\,(S_s - 1)} \qquad \text{5-11}$$

Where: D = pipe diameter

$S_s$ = specific gravity of solids

$F_L$ = coefficient depending on particle size and volumetric concentration.

For uniformly sized particles the factor $F_L$ may be read from Figure 5-9 for concentrations up to 15% by volume.

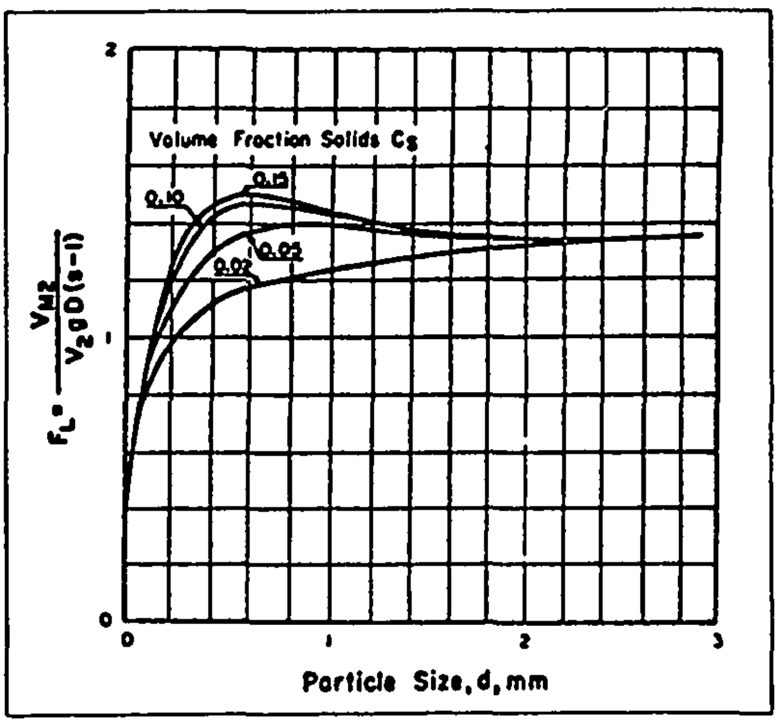



Figure 5-9 Durands correlation for Minimum Transport Velocity

Durand's equation has been modified by Wasp[57] to include the effect of relative particle size with respect to pipe diameter. The modified equation is given by :

$$F'_L = \frac{V_c}{\sqrt{2\,g\,D\,(S_s - 1)}}\left(\frac{d}{D}\right)^{-0.17} \qquad \text{5-12}$$

This equation relates the critical velocity Vc to the pipe diameter with an exponent of 0.33 which is in agreement with experimental results for a variety of slurries. The modification increases the applicable concentration to 30% by volume (54% by weight). For practical purposes the critical velocity calculated this way is relevant for first approximations only. The scale-up formula is given by:

$$V_{c2} = V_{c1}\left(\frac{D_2}{D_1}\right)^n \qquad \text{5-13}$$

Where: $V_{c2}$ = critical velocity of pipe (2)
$V_{c1}$ = critical velocity of pipe (1)
$D_1$ = diameter of pipe (1)
$D_2$ = diameter of pipe (2)
n = 0.5 ......... according to de Korompay
= 0.33 ....... according to Sellgren

Once the critical velocity is determined, a safety margin is usually added to obtain the operating velocity. In practice :

a) If critical velocity, Vcrit < 4 ft/sec $V_{op} = V_{crit} + 1$

b) If critical velocity, Vcrit > 4 ft/sec $V_{op} = 1.25\, V_{crit}$

Where $V_{op}$ is the operating velocity in (ft/sec)

## 5.2. HYDRAULICS OF SLURRY TRANSPORTATION

In order to analyze the connected vertical-horizontal slurry delivery system, it is necessary to have some basic understanding of the mechanism of fluid flow in the vertical and horizontal sections independently. De Korompay suggests the following topics for investigation:

- Determination of the Hazen-Williams friction factor for vertical boreholes.
- Evaluation of the friction head loss and flow capacity of the full-flow section of the vertical borehole or pipe.

- Analysis of the flow conditions in the free-fall section of the vertical pipe.
- Analysis of the friction loss in the horizontal transportation pipe caused by the slurry flow.
- Development of a simple calculation formula and the presentation of a testing procedure to evaluate the friction head loss of horizontal pipe for slurry transportation.

### 5.2.1. Slurry Transportation in Vertical Borehole or Pipes

The Hazen-Williams friction factor for boreholes in hard rocks was determined from flow tests run in Elliot Lake laboratory, which indicated that $C_{hw}$ =mn 130-140. For a different type of rock mass, it is suggested to run in-situ tests with water to obtain the friction coefficient. It is reported that the presence of solid particles in a water-solid mixture does not alter the turbulent friction head loss of water during vertical transportation. This result makes the Hazen-Williams equation for water applicable to the calculation of the friction head losses of vertical boreholes or pipes for slurry transportation. These equations are given by:

$$H_V = \frac{0.2083}{d^{4.8655}} \left(\frac{100\,Q}{C_{hw}}\right)^{1.85} = \frac{5645}{d^{4.8655}} \left(\frac{100\,T}{C_{hw}\,S_m\,C_w}\right) \qquad \textbf{5-14}$$

Where: $H_V$ = friction head loss of vertical pipe or borehole for water or slurry transportation in ft of water head per 100 ft of pipe.

The transportation capacity of vertical borehole or pipe by gravity is given by the following equations ( assuming a Hazen-Williams friction factor of $C_{hw}$= 140 ):

- Discharge rate Q, [USGPM]

$$Q = 39.4\, S_m^{0.54}\, d^{2.63} \qquad \textbf{5-15}$$

- Delivery rate, T [ TPH of dry solids ]

$$T = 0.158\, S_m^{0.54}\, d^{2.63}\, C_w\, \rho_s \qquad \textbf{5-16}$$

- Flow velocity [ ft/sec ]

$$V = 16.1\, S_m^{0.54}\, d^{0.63} \qquad \textbf{5-17}$$

### 5.2.2. Free Fall in Vertical Pipe

It is common in practice for the fill to flow in a free-fall mode through considerable length in the vertical transportation line . The Bernoulli equation is no longer valid for this condition because of the discontinuity in the flow. It was determined that the slurry terminal velocity may reach $V_{ff}$ =45 ft/sec after a vertical drop of about 32 ft. This information leads to the conclusion that it is necessary to restrict the slurry flow in the free-fall sections in order to prevent excessive pipe wear.

The transportation capacity of the vertical pipe or borehole by free-fall can be estimated by the following equation.

$$Q = \mu \left( \frac{V^* d^2}{0.409} \right) \qquad 5\text{-}18$$

Where: $V^*$ = terminal fall velocity of slurry (45 ft/sec)

$\mu$ = pipe cross-section utilization factor ( e.g. 0.7)

### 5.2.3. Slurry Transport in Horizontal Pipeline

The factors known to affect the head loss in the horizontal hydraulic transport are:

- Flow velocity
- Solids concentration of slurry
- Specific gravity of slurry and of suspended solids
- Viscosity of slurry ( for viscosity-controlled suspensions)
- Size and distribution of solid particles
- Characteristics of the carrier fluid

Equations frequently used for predicting head loss in solid-liquid mixtures often take the form given by:

$$\left( \frac{J - J_w}{C_v J_w} \right) = K \left( \frac{V^2 C_d^{0.5}}{g D (S_s - 1)} \right)^{3/2} \qquad 5\text{-}19$$

Where: K = empirical parameters determined from data collected by loop tests;

J = head loss for slurry flow;

$J_w$ = head loss for water flow;

$C_v$ = Solids concentration by volume;

V = mean flow velocity;

$C_d$ = drag coefficient;

D = pipe diameter;

$S_s$ = specific gravity of solids;

This type of equation is suitable for coarse particles of relatively low concentration (up to 30% by volume) slurries flowing in the turbulent mode; although they may be used in other cases by suitably adjusting the empirical parameters. The rheology of the suspensions is not taken explicitly into account in these equations, which makes rheologically-based approaches preferable when dealing with high concentration slurries.

Alternatively, de Korompay [54] proposes a simple empirical equation for the prediction of head loss in the horizontal transport of backfill slurries as given by:

$$R_{h-w} = S_m S_h + C_v ( S_s - 1) \frac{\Omega}{V} \qquad \textbf{5-20}$$

$$\text{Where: } S_h = \frac{0.2083}{d^{4.8655}} \left( \frac{100\, Q}{C_{hw\_h}} \right)^{1.85}$$

$R_{h-w}$ = friction head loss for horizontal pipe for slurry transportation --ft of water head per 100 ft of pipe

$S_h$ = friction head loss of horizontal pipe which would exist if water flowed alone in the same pipe with the same velocity as the slurry -ft of water head per 100 ft of pipe

$S_m$ = specific gravity of slurry

$S_s$ = specific gravity of solid particles

$\Omega$ = settling velocity factor (obtained from Figure 5-10)

$C_{hw\_h}$= Hazen-Williams friction factor for horizontal pipe

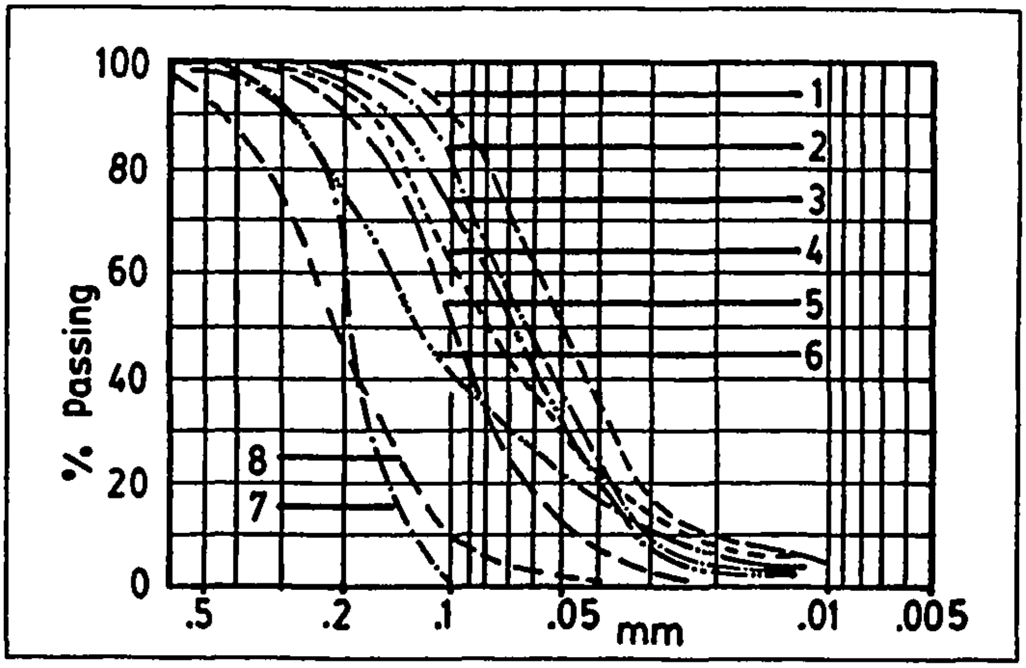



Figure 5-10 Determination of the settling velocity factor Ω

| No. of Curve | $\Omega$ |
|---|---|
| 1 | 13 |
| 2 | 18 |
| 3 | 23 |
| 4 | 30 |
| 5 | 40 |
| 6 | 56 |
| 7 | 80 |
| 8 | 100 |

Compare actual grain size distribution with the master curves and select the closest and read $\Omega$.

### 5.2.4. Slurry Transport in Vertical Section (connected to the horizontal pipe)

Vertical flow in pipe or borehole is characterized by two flow regimes: free fall and full flow. Only the full flow section of the pipe provide the necessary pressure for the horizontal transportation of the slurry. The hydraulic resistance of the vertical pipe depends on the following factors:

- Pipe Diameter
- Length of full-flow section in pipe
- Hazen-William friction factor of pipe
- Flow rate

Two hydrostatic heads are recognized in the vertical pipe connected to the horizontal one:
(1) Hydrostatic head for the horizontal transportation of slurry, $H_{s\text{-}w}$
(2) Hydrostatic head which is necessary to deliver slurry through the full-flow length in the vertical pipe, $h_w$
The sum of (1) and (2) yields the total hydrostatic head $H_{t\text{-}w}$, as given by:

$$H_{t\text{-}w} = H_{s\text{-}w} + h_w \qquad \textbf{2-21}$$

Where: $H_{s\text{-}w} = R_{s\text{-}w} = S_m S_h + C_v\,(S_s - 1)\dfrac{\Omega}{V}$

$$S_h = \frac{0.2083}{d^{4.8655}}\left(\frac{100\,Q}{C_{hw\_h}}\right)^{1.85}$$

$$h_w = S_v \frac{R_{t-w}}{10^2} + S_v^2 \frac{R_{t-w}}{10^4} + S_v^3 \frac{R_{t-w}}{10^6}$$

$$S_V = \frac{0.2083}{d^{4.8655}} \left( \frac{100\ Q}{C_{hw\_v}} \right)^{1.85}$$

$C_{hw\_v}$ = Hazen-Williams friction factor for vertical pipe

The total hydrostatic head in the vertical pipe, $H_{t-s}$ in feet of slurry, corresponds to the length of the full-flow section in the pipe, and is given by:

$$H_{t-s} = \frac{R_{t-w}}{S_m} + S_v \frac{R_{t-w}}{S_m^2\ 10^2} + S_v^2 \frac{R_{t-w}}{S_m^3\ 10^4} + S_v^3 \frac{R_{t-w}}{S_m^4 10^6} \qquad \text{5-22}$$

The relation between the total vertical height for gravity feed $H_g$, the full-flow length $H_{t-s}$, and the free-fall length $H_{f-f}$ is given by:

$$H_g = H_{t-s} + H_{f-f} \qquad \text{5-23}$$

## 5.2.5. Utilization of the Vertical Height for Gravity Feed

The 'vertical height utilization' factor is defined as the ratio of the slurry height in the pipe to the available height for gravity feed as given by:

$$t_S = \frac{H_{t-s}}{H_g} \quad \text{or} \quad t_W = \frac{H_{t-w}}{H_g} \qquad \text{5-24}$$

Where $t_S$ = vertical height utilization factor for slurry -decimal fraction

$t_W$ = vertical height utilization factor for water -decimal fraction

$H_g$ = total vertical height for gravity feed

$H_{t-s}$ = total hydrostatic head in the vertical pipe -ft of water head

$H_{t-w}$ = total hydrostatic head in the vertical pipe -ft of slurry head

It is suggested that a maximum height utilization factor of $t_W$ = 0.65 for slurry transportation is recommended to compensate for any error in the determination of the friction head loss of the horizontal pipe. It is also noted that a vertical to horizontal length ratio in the range of 1:3 to 1:19 is common in practice.

### 5.2.6. Flow Pressure in the H-V System

The maximum flow line pressure can be calculated as:

$$P_m = 0.433\ R_{s\text{-}w\text{-}m} \tag{5-25}$$

Where: $P_m$ = maximum flow line pressure, psi

$R_{s\text{-}w\text{-}m}$= hydraulic resistance of the horizontal pipe at maximum flow rate of the slurry

In practice, the maximum flow line pressure can be measured directly with a gauge installed at the junction point of the H-V system.

### 5.2.7. Selection of Transportation Pipes

#### 5.2.7.1. Case of Horizontal Pipes

Pipe diameter depends on the volume of slurry to be handled and the critical velocity of the slurry. The optimum pipe diameter corresponds to the maximum service life of the pipe which is related to the optimum operating velocity. Hence:

$$d_{op} = \sqrt{\frac{0.409\ Q}{V_{op}}} \tag{5-26}$$

$V_{op} = V_{crit} + 1$ if $V_{crit} < 4$ ft

$V_{op} = 1.25\ V_{crit}$ if $V_{crit} > 4$ ft

#### 5.2.7.2. Case of Vertical Pipe

Since there is no critical velocity for vertical transport, both discharge rate and flow velocity can be selected, hence pipe diameter can be calculated from:

$$d = \sqrt{\frac{0.409\ Q}{V}} \tag{5-27}$$

The technical feasibility of the installation can be determined by checking that the required hydrostatic head $H_{t\text{-}s}$ in the pipe is smaller than the available height for gravity feed.

#### 5.2.7.3. Variable Pipe Diameter

The incentive for using variable pipe diameter in the vertical section of the fill transport system is that pipe cost is proportional to pipe diameter, therefore the smaller the diameter, the lower the cost. Furthermore, a higher utilization of the cross section area of the pipe, which is associated with a smaller pipe cross section, reduces the free-fall velocity in the vertical pipe due to wall friction.

A "pipe cross section utilization" factor defined as $m = \frac{A_s}{A}$ may be introduced as a design parameter,

where: $A_s$ = cross section area of pipe occupied by the slurry

$A$ = cross section area of pipe

For practical applications $\mu$=0.8 or less is recommended. The diameter of the free fall section is obtained from:

$$d_{ff} = \sqrt{\frac{0.409\,Q}{V_{ff}}} \qquad \text{5-28}$$

where: $d_{ff}$ = pipe diameter for the free-fall section, in.

$V_{ff}$ = terminal fall-velocity of the slurry, ft/sec

## 5.2.8. Selection of the Size of the Vertical Borehole

The smaller the borehole diameter, the lower the corresponding cost. The optimum borehole size is the smallest one which satisfies the required operating conditions. It is given by:

$$d_{op} = 1.62 \left(\frac{R_{s\text{-}w\text{-}q} + h_{w\text{-}a}}{h_{w\text{-}a}}\right)^{.21} \left(\frac{Q}{C_{hw}}\right)^{0.38} \qquad \text{5-29}$$

Where: $d_{op}$ = optimum diameter of the vertical borehole, in

$R_{s\text{-}w\text{-}q}$ = hydraulic resistance of the horizontal pipe at Q flow rate, ft of water head

$h_{w\text{-}a}$ = allowable friction head loss in the borehole at Q flow rate, ft of water head

## 5.2.9. Selection of Wall Thickness of Pipeline

The minimum wall thickness required must be selected to resist the effect of line pressure and that of corrosion and erosion, as given by:

$$W_i = W_p + W_{e\text{-}c} \qquad \text{5-30}$$

Where: $W_i$ = initial wall thickness

$W_p$ = wall thickness to control the line pressure, in

$W_{e-c}$ = wall thickness to account for the wear by erosion and corrosion, in

The wall thickness to control line pressure may be expressed by:

$$W_p = K \frac{P_m \, dx}{2\,E} \qquad \text{5-31}$$

where: dx = Outside diameter of pipe, in

E = Ultimate tensile strength of material, psi

K = safety factor

**<u>5.2.10. General Observations</u>**

There has been a great deal of work on the hydraulic transportation of slurries and numerous empirical equations exist. It is suggested that, especially for high concentration slurries (Cv > 40%) and definitely for 'pastes' (Cv >50%), loop tests be carried out to verify pressure loss predictions and optimize the backfill system design.

The design rationale, assuming that the proportions of the backfill ingredients are known to provide adequate flow properties, can be summarized as:

1) The geometry of the backfill placement system must be available.
2) The static head available can be calculated from the total vertical drop and the specific gravity of the slurry.
3) The pressure losses must be determined and must not exceed the static head available, including allowances for the vertical drop and bends.
4) If the static head is not sufficient then a pump will be required. Such information regarding pump suitability can be found in any pump manual.
5) From the velocity required (from Vc calculations) the capacity of the system can be found.
6) Care must be taken at the bottom of large vertical drops where the pressure can be high. This can lead to stress and possible pipe bursts. Introducing 'steps' to the vertical drop reduce the pressure build up. Such measures must be accounted for in the static head calculation.

## 5.3. TRANSPORTATION OF PASTE BACKFILL

The transportation of highly viscous substances has become routine in the food processing and sewage industries, but the transportation of paste backfill owes more to the concrete industry for its technology. Concrete has a similar broad grain size distribution as some paste fills and the plug flow associated with concrete transportation has become the most likely model for describing paste fill.

When attempting to analyze the flow of paste by comparison with concrete a few distinctions must be drawn.

1) The abrasive characteristics of the paste fill particles are likely to be far higher than with concrete.
2) Far less cement is added. The majority of the fines in paste fill is made up of tailings, this results in higher cohesion hence higher resistance to flow.
3) Paste fill is often transported over greater distances.
4) Preparation is far more complex.

These factors result in higher pressure losses and larger ranges in estimates. It is not prudent to design a paste backfill system without loop test results to aid in pressure loss prediction. There are large capital costs involved but the cost of a plugged line is potentially crippling for a backfill operation.

### 5.3.1. Quality Control

The quality control of paste backfill is of paramount importance. If the concentration varies by one or two percent. the pressure losses can double, (at 10 $m^3/h$ the pressure losses increase from 14 to 38 kPa/m when the concentration of gold slimes slurry increases from 76 to 77 percent by weight[59]). To ensure good Q. C, accurate determination of original paste ingredients is required.

Depending on the dewatering method used one will have to add either cement (and aggregates, if available) or more water to reach the desired paste concentration.

| | |
|---|---|
| 1) Centrifugal dewatering | 25 - 40% moisture |
| 2) Filter dewatering | 12 - 30% moisture |

Other quality control method include having a small test loop on surface where the flow of the paste can be checked before being sent into the backfill system. Although there is added cost involved in such methods, their value is immeasurable. Paste backfill can only benefit from increased awareness and development of quality control methods.

The design rationale of a paste backfill transportation system is not yet completely understood but must include the following salient points:

1) The pressure loss range must be well known.
2) Long vertical drops will cause high pressure and must be avoided or minimized.
3) At the end of a fill pour, flushing is required to ensure the material is cleared from the line; this is especially important for cemented paste. High pressure water and/or air is often used. Lucky Friday utilizes slugs of water propelled by air pressure.

## 5.4. PUMPS

The transportation of mine waste in slurry form often requires pumps. The type of pump required will depend on the concentration, weight, and chemical properties of the slurry in question. There are two types of pumps used in slurry transportation; centrifugal and positive displacement pumps.

### 5.4.1. Centrifugal Pumps

Concentration by volume of 50 per cent is the upper limit for a standard heavy duty centrifugal pump, (approx. 73% concentration by weight for s.g = 2.7g/cm$^3$). Centrifugal pumps can handle a very large variety of materials with maximum particle sizes up to 9" in diameter. For hydraulic fill placed at 65-70% concentration by weight, centrifugal pumps are sufficient. Care must always be taken to ensure that any pumping system can lose a pump for maintenance or repair without losing capacity. Shown in Figure 5-11 is the pumping station used to transport sandfill at Inco's Thompson Mine in Manitoba. This setup allows a pump to be bypassed via the liquid end for simple maintenance and the rubber hoses enable the whole pump to be lifted from the line for full servicing or replacement.

### 5.4.2. Positive Displacement Pumps

With the recent increased interest in paste backfill, a few companies have emerged as experienced suppliers of high pressure, high capacity positive displacement pumps. Putzmeister has developed their successful concrete pumps and now supplies pumps actively transporting high density fill in Germany, USA and South Africa. GEHO is a company from Holland with a long history of powerful pumps, some of their pumps can transport highly viscous abrasive materials through pressures of up to 165 Bar. Schwing is among many companies who are able to apply their experience in pumping large volumes of heavy sludge to the problems associated with paste backfill.

The cost of positive displacement pumps are many times greater than those of centrifugal, up to $CDN 250,000. As with all pumps, maintenance is a very important factor in choosing a pump.

The costs involved in repair and replacement parts can be prohibitive, and a warranty protecting purchases is recommended.

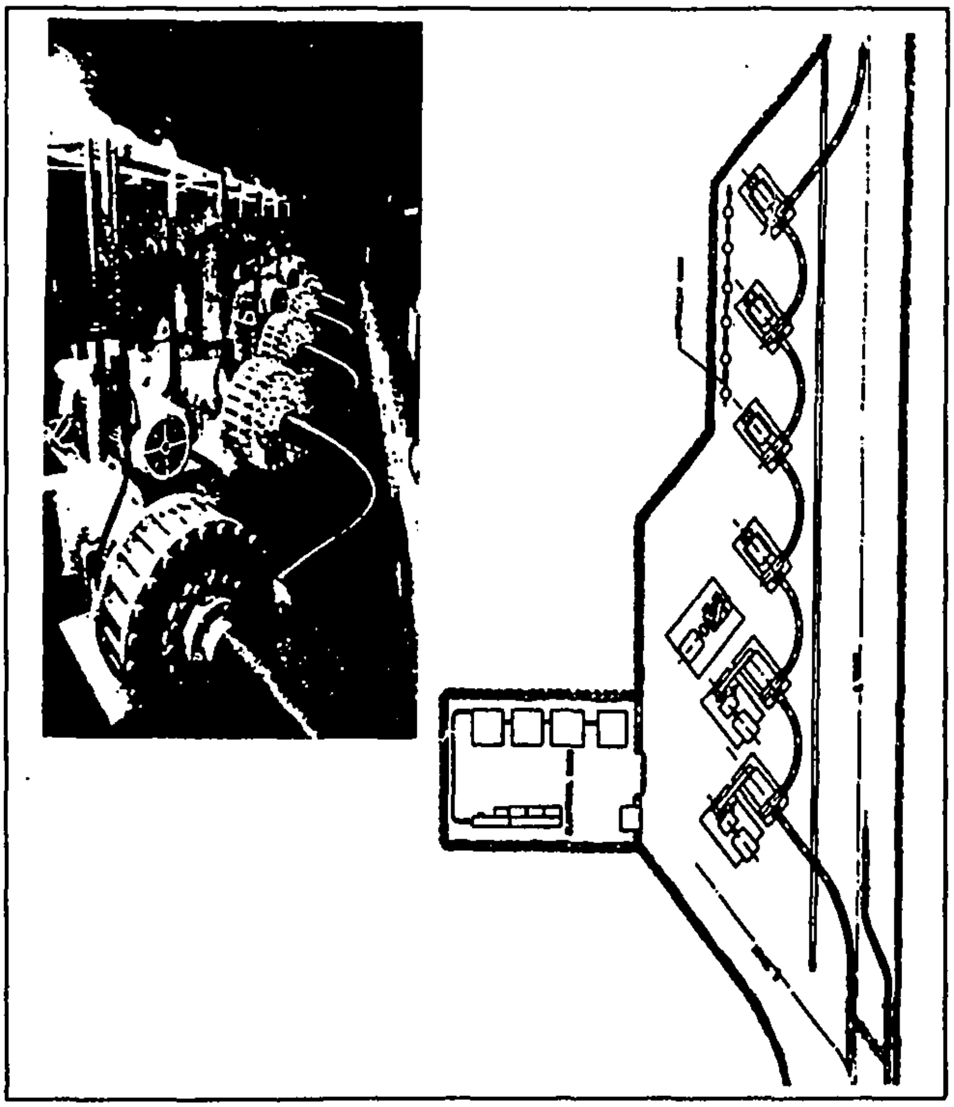


Figure 5-11 The Pumping station at the T1 mine in Thompson, Manitoba

### 5.4.3. Pump Selection

After the initial selection between centrifugal or positive displacement pumps, the hydraulic profiles over the proposed transportation route must be calculated. This requires knowledge of the pressure loss through any pipeline, including corners, bends, inclines, etc.. Once the pressure (or head) and capacity required are determined, the relevant pump size can be found. The material from which the pump is manufactured is dependent on the corrosiveness and abrasivity of the slurry being pumped. So, a simple rationale for pump selection could be:

Concentration $0 < C_v <= 50\%$ = Centrifugal pump
$100 > C_v => 50\%$ = Positive Displacement pump
Abrasivity High, >7 (on the original Mohs scale) - chrome plated steel
Medium, 5 - 7 (Mohs) - rubber lined mild steel
Low, <5 (Mohs) - unlined mild steel

A high pressure positive displacement pump will require superior pipeline connections and overall higher quality, including more maintenance and supervision. Post sales service is very important when choosing a positive displacement pump since spare parts and repairs can be costly. Such machines are precisely engineered, and when working in a harsh environment, such engineering can be expensive to maintain.

Figure 5-12 shows a GEHO positive displacement pump.

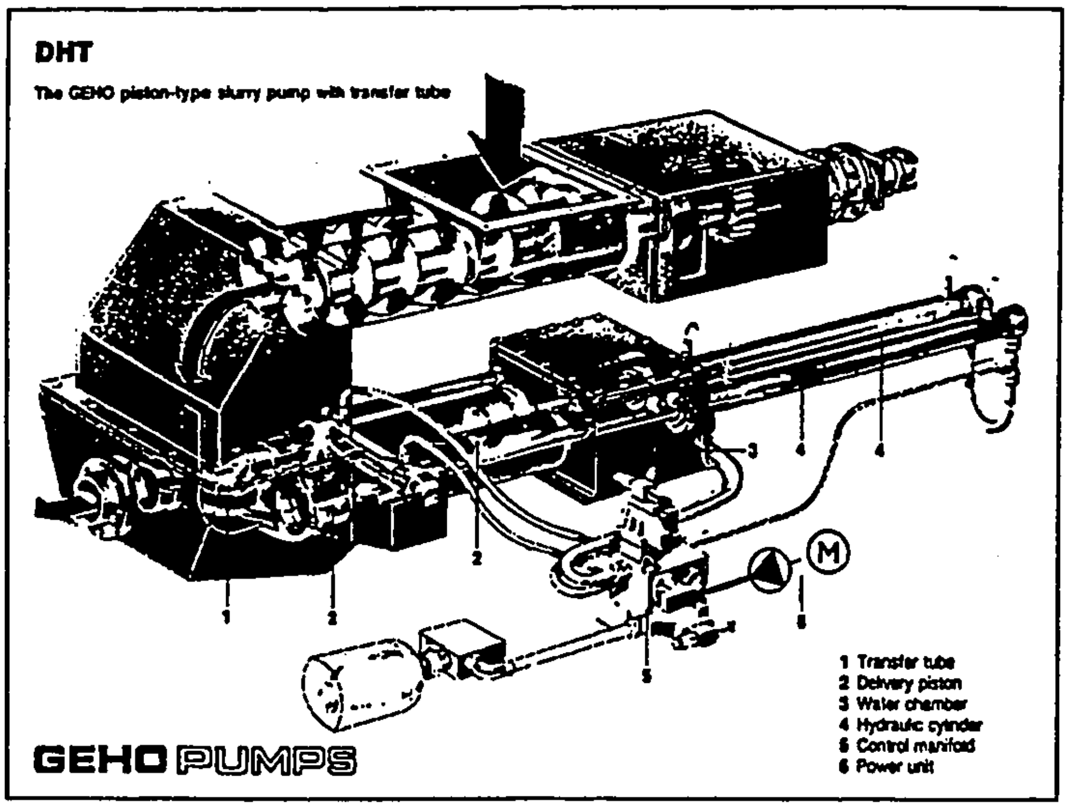



Figure 5-12 GEHO paste piston pump

# CHAPTER 6

# BACKFILL PLACEMENT OPERATION

## 6.0. INTRODUCTION

The preparation of a stope prior to backfilling varies with the type of stopes, which in turn is affected by mining methods. This can be broadly classified as follows:

i) Preparation in cyclic stope production prior to backfilling, which requires an initial or basic preparation followed by cyclic adjustments and maintenance.
ii) Preparation in non-cyclic or open stopes prior to backfilling, in which the preparation activity is delayed until the stope depletion. It is then commenced as one continuous operation to completely backfill the stope.

In general, stope preparation involves the following:

a) Installation of stope and backfill dewatering systems
b) Installation of bulkhead

Illustrated in figure 6-1 is the level-1 data flow diagram of process 3.3.

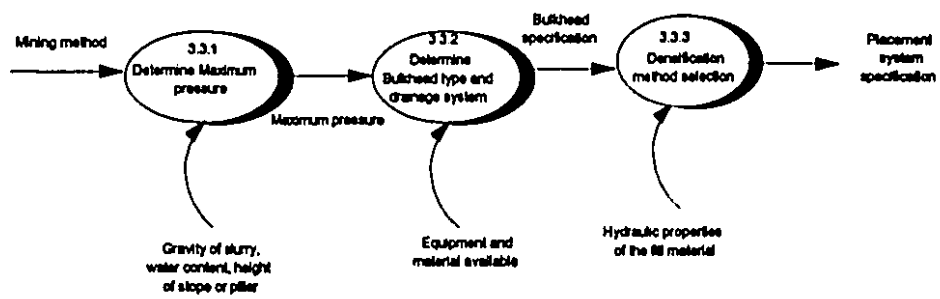



Figure 6-1 The level-1 data flow diagram of process 3.3.

The process 3.3.1 request the mining method specified in process 1 and the information related to the specific gravity of slurry, water content and height of the stope to determine the maximum pressure imposed on the bulkhead. The process 3.3.2 requests the information about equipment and construct available to match with the maximum pressure derived from process 3.3.1 to determine the type of the bulkhead and drainage system. Finally, the process 3.3.3 requests the information about hydraulic properties and fill material to determine the densificaton process. The overall process provides the complete specification of the placement operation of the backfill system. The following section discuss in detail the application of various techniques related to placement operation which can be used as the quick reference for placement design.

## 6.1. STOPE DEWATERING SYSTEM

The following is a typical installation of stope dewatering system from Ashby, I.R & Hunter, G.W[60], which covers most of specifications related to dewatering system.

Once a stope has been mined, slotted 75-150 mm diameter galvanized or polythene agricultural pipe is laid along the footwall of the stope. Hessian or a coarse cotton sleeve is then wrapped around either type of pipe.

Drain towers are built on brick and concrete foundations at either end of the excavation, or at the filling limit for each section being filled in larger excavations. The tower consists of tightly rolled lengths of 150 mm square wire mesh and are wrapped with 4 layers of hessian to act as a filter.

Four towers approximately 3.2 m in height are built close to each drain raise which is located at strategic points along each orebody (see Figure 6-2).

Drain raises are kept open by bolting pre-cut and drilled crib timbers together. The raise is then wrapped with chicken wire, hessian and polythene. Fill water can then only escape through the drain raises via the drainage towers and connecting 100 mm diameter drain pipes. The water is directed to sumps located on lower levels.

Prior to filling all ore passes, access and crosscuts are closed off with 0.4 - 0.9 m thick brick bulkheads constructed on concrete pads and secured at the walls 2.5 - 3.5 m from the stope edge. For ore passes two 0.4 - 0.5m thick brick bulkheads are built approximately 1.5m apart. Wire mesh is placed in the void between the bulkheads and concrete is then blown in. This is done to prevent fill from collapsing into the ore pass during future lifts.

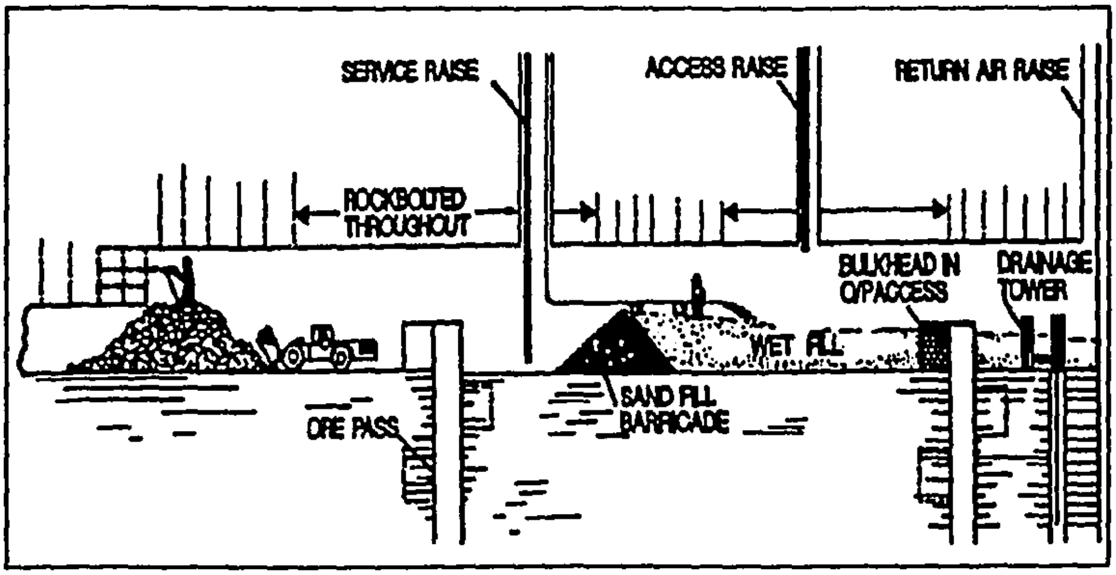



Figure 6-2 Hydraulic Fill Placement

In the longer orebodies sandfill, ramps and barricades are built to separate cyclic production activities, and plastic sheeting is laid down the inner slope to prevent excessive seepage. Hydraulic fill placement commences at this barricade with 6m long x 100 mm diameter pipe being suspended from existing rockbolts and supported by wire rope. Tee pieces are installed in sections of the line to allow for backfilling as the pipe is extended along strike and the fill builds up to a height of 2.6 to 2.8 m from the back.

Two percolation pipes are installed through each bulkhead. They consist of slotted 90 to 110 mm galvanized or polythene agricultural pipe wrapped in hessian and fixed to valves outside the bulkhead, Figure 6-3. Percolation water is either pumped into the sublevel drainage system or allowed to gravitate via drain holes to the main drainage level.

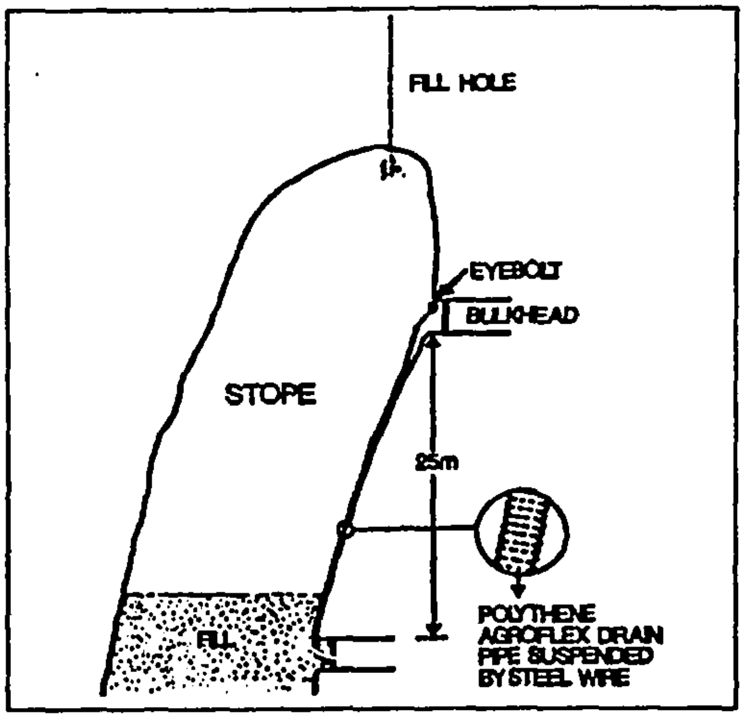



Figure 6-3 Percolation Pipes in Open Stope

## 6.2. BULKHEAD

There are many types of standard bulkhead. The most common one will be described briefly below. Costs are very site specific and can vary tremendously depending on the mining and backfill method.

A bulkhead is defined as a structure which contains fill materials in a stope. Bulkheads are installed in the drawpoints of the stopes or on remnants of sub-levels. Bulkheads to retain dry fill, do not require any special consideration unless water flows into or onto the fill. Rockfill containing appreciably large amounts of fines, when saturated, could generate mud flow into adjacent working areas, hence a bulkhead is required. Bulkhead to retain hydraulic fill, requires special considerations such as drainage system, solids retaining devices, etc.

### 6.2.1. Main Types of Bulkhead Materials

#### 6.2.1.1. Pressure treated wood

These are used most commonly in the drawpoint. This type of bulkhead can resist fill pressure of up to 207 kPa (30 psi). Wooden bulkhead is relatively easier and faster to install, easily available in most mining operations, has a higher ratio of deflection to load than does concrete or steel, and is less costly than the other types. However, wood is combustible in high-heat areas and deteriorates fast. For bulkheads, timber is sometimes used in various forms such as: post, crib, tie, cap piece, wedge, crossbar and plank of various sizes, given in Table 6-1.

Table 6-1 Typical bulkhead timber form and sizes[61]

| Dimensions Form | Smaller Diameter (D) or thickness (cm) | Width (cm) | Length (L) (cm) |
|---|---|---|---|
| **Post** | 10 - 30 | - | varies but usually L/D < 15 |
| **Crib Block** | 10 - 15 | 10 - 20 | 75 - 400 |
| **Tie** | 10 - 15 | 15 - 20 | 150 - 200 |
| **Cap Piece** | 3 - 10 | 10 - 20 | 30 - 50 |
| **Wedge** | 0.2 - 1 | 10 - 15 | 25 - 45 |
| **Crossbar** | 7 - 20 | 10 - 20 | 200 - 480 |
| **Plank** | 3 - 6 | 15 - 25 | 45 - 400 |

The type of timber can be hardwood such as maple, oak and aspen, or softwood such as fir, and spruce. The mechanical properties of which are given in Table 6-2. The bulkheads are generally made from hardwoods.

Table 6-2 Mechanical properties of commonly used bulkhead timber[62]

| Properties<br>Type | Specific gravity ( $\alpha$,kN/m$^3$) | Modulus of rupture (E, MPa) | Compressive strength (||el*. to the grain, MPa) | Shear strength (||el*. to the grain, MPa) |
|---|---|---|---|---|
| Maple, green wood | 4.3-4.5 | 39.5-40.5 | 17.0-17.5 | 7.3-7.7 |
| " , 12% moisture | 6.0-6.5 | 108.0-109.5 | 53.5-54.5 | 15.7-16.0 |
| Red oak, green wood | 5.0-5.4 | 50.5-51.5 | 21.2-21.5 | 6.3 - 6.5 |
| " " ,12% moisture | 6.6-7.1 | 105.1-106.4 | 59.5-60.2 | 14.1-14.4 |
| White oak, green wood | 5.5-6.0 | 49.3-50.0 | 22.6-22.9 | 8.2-8.5 |
| " " , 12% moisture | 7.0-7.4 | 126.5-127.0 | 61.3-61.5 | 18.1-18.4 |
| Aspen, green wood | 3.5-3.8 | 35.1-35.3 | 14.4-14.6 | 4.0-4.2 |
| " , 12% moisture | 3.6-4.0 | 62.5-62.9 | 36.4-36.7 | 7.3-7.5 |
| Fir, green wood | 3.2-3.5 | 33.7-34.0 | 16.3-16.6 | 4.1-4.3 |
| ", 12% moisture | 4.2-4.5 | 73.6-73.9 | 44.7-44.9 | 7.4-7.6 |
| Pine, green wood | 3.3-3.5 | 32.3-32.6 | 16.7-16.9 | 4.6-4.9 |
| ", 12% moisture | 5.7-6.2 | 99.5-101.0 | 51.852.1 | 10.3-10.5 |
| Spruce, green Wood | 3.1-3.5 | 32.2-32.6 | 14.9-15.1 | 4.2-4.5 |
| " , 12% moisture | 3.8-4.2 | 70.1-70.5 | 40.5-40.8 | 8.1-8.4 |

(*llel = parallel)

Preservative treatment can increase the service life of the wooden bulkhead. The wood is heated slightly to decrease its moisture content to 19% or less, prior to treatment. This is to provide good penetration of the preservative solution into the wood. It is then soaked in a mixture of: - 12 to 16 pcf creosote, and - 0.6 to 1.5 pcf chromated copper arsenate. This mixture preserves the wood for 20 to 50 years, dependent on the site condition.

#### 6.2.1.2. Concrete block, or brick and cement

Concrete block, or brick and cement bulkhead can be used to resist backfill pressure of up to 2758 kPa (400 psi). They are more expensive than the wooden bulkheads and have 28 days curing period before withstanding the fill pressure.

#### 6.2.1.3. Reinforced concrete, or steel frame and plate

Reinforced concrete, or steel frame and plate are used to resist higher fill pressures. They are the most expensive bulkheads. A combination of the above noted types provides advantages of each. Such combinations are also more expensive than the wooden bulkheads.

### 6.2.2. Estimation of Maximum Pressure on Bulkhead

Considering a vertical bulkhead, installed some distance from a stope face, as shown in Figure 6-4.

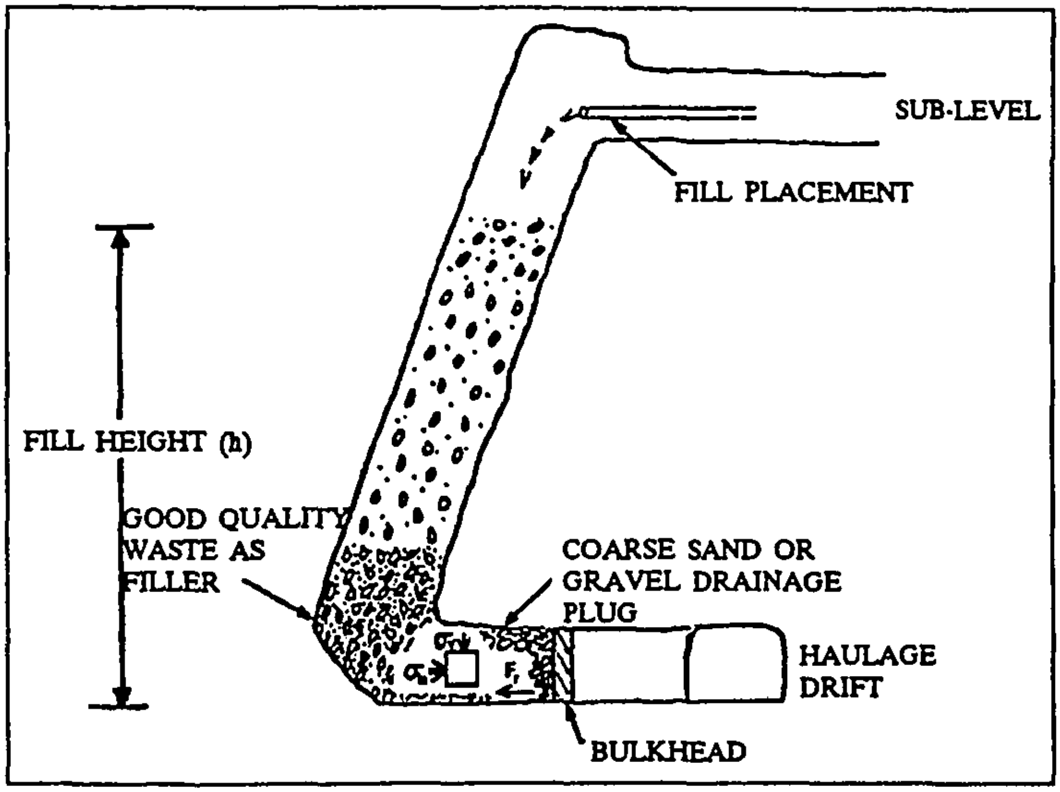



Figure 6-4 Schematic of a stope being filled.

The general equation is:

$$P = \sum \sigma_h + \sigma_v - F_r = U + P_a - F_r \qquad \textbf{6-1}$$

where: P = Total max. pressure on bulkhead, kPa.

$\sigma_h$ = horizontal fill pressure, kPa.

$\sigma_v$ = vertical pressure due to the fill and water, kPa. $\sigma_v = \gamma h$

$\gamma$ = average weight per unit volume of the fill, $kN/m^3$.

h = fill height, m.

$F_r$ = frictional resistance of the fill material, kPa; $F_r$= 0.1 P to 0.4 P.

U = hydrostatic water pressure in the fill = $\gamma_w$.h, kPa

$P_a$ = active fill pressure acting horizontally on bulkhead, kPa.

$$P_a = \sigma_v \cdot \tan^2\left(45 - \frac{\theta}{2}\right) = \gamma \cdot h \tan^2\left(45 - \frac{\theta}{2}\right) \qquad 6\text{-}2$$

$\theta$ = internal friction angle of the fill, degree.

### 6.2.2.1. Stope filled with water

The total pressure on the bulkhead in this case is: (P) = hydrostatic pressure exerted by a column of water h metres high (U) + active fill pressure ($P_a$) - Fractional resistance ($F_r$). So,

$$P = \gamma_W . h + 0 - 0 \qquad 6\text{-}3$$

where: h = fill or water height

$\gamma_w$ = weight per unit volume of water = 9.8, kN/m³

For h= 30m then P= 294 kPa

### 6.2.2.2. Stope with saturated fill and no drainage

This situation occurs when the drainage system (mousetraps, pipes, etc.) becomes inoperative.

The total pressure on the bulkhead:

$$P = U + P_a - F_r = \gamma_w \cdot h + \gamma_{sf} \cdot h \tan^2\left(45 - \frac{\theta}{2}\right) - F_r \qquad 6\text{-}4$$

### 6.2.2.3. Stope with 50% saturated fill and good drainage

This situation prevails during good filling practice. Ideally this condition exists when the fill grain size is 2 to 200 mm, with zero to 2 mm sieve sizes removed. This involves considerable expense to prepare. Poor quality waste such as clay, chlorite, mica, or sericite schist (which would produce abundant fines, or readily decompose) should be avoided.

The total pressure on the bulkhead (P) = U + P - $F_r$

The hydrostatic water pressure (U) at a free draining bulkhead is zero. However, it exists at the plug side of the bulkhead to force water to be drained. This pressure acts to dislodge the bulkhead. The magnitude of this pressure can be reduced by providing a clean pervious drift plug coarse sand, gravel or inside drainage boxes. Hydrostatic water pressure can be estimated as follows:

$$u = Q \cdot l_d \cdot \frac{\gamma_w}{R_w} \cdot A \qquad 6\text{-}5$$

$$Q = \left(\frac{W_{ds}}{t}\right) \cdot \left[\frac{(1-\gamma_p)}{\gamma_p}\right] - \left[\frac{n(1-s_w)}{\rho_s(1-n)}\right] \qquad 6\text{-}6$$

where: $Q$ = maximum quantity of drainage water, tonnes/hr or $m^3$/hr.

$l_d$ = length of drift plug

$\gamma_w$ = weight per unit volume of water

$R_p$ = percolation rate of drift plug material

$A$ = cross sectional area of drainage drift

$W_{ds}$ = weight of dry solids in the fill per days

$t$ = fill placement times

$\gamma_p$ = pulp density as a decimal

$S_w$ = water saturation as a decimal

$n$ = porosity of settled fill

$\rho_s$ = specific gravity of fill solids relative to water in the fill

$\gamma_{bsf}$ = weight per unit volume of 50% saturated fill (kN/$m^3$)

### 6.2.2.4. Stope with 15% water by weight saturated fill and fully drained

This condition exists after completion of filling a stope.

The total pressure on the bulkhead (P) = $U + P_a - F_r$; U = O, for fully drained stope

$$P_a = \gamma_f \cdot h \cdot \tan^2\left[45 - \frac{\theta_p}{2}\right] \qquad 6\text{-}7$$

where: $\gamma_f$ = weight per unit volume of 15% by weight water saturated fill (kN/$m^3$)

$\theta_p$ = internal friction angle of the 15% saturated fill (degrees)

### 6.2.2.5. Stope with liquefied fill

Liquefaction involves a reduction to zero of the effective horizontal stress within the mass of granular fill. This situation exists if energy is imparted to the mass by occurrences such as rockburst, blasting, earthquakes etc. Under such conditions, the fill behaves as a liquid with a density nearly double that of water. The result is a sudden pressure increase on the bulkhead equivalent to the hydrostatic head of a fill slurry. Therefore:

The total pressure on the bulkhead (P) = U + $P_a$ - $F_r$

$u = \sigma_V = \gamma_{lf} . h$ , and $P_a = 0$

Therefore:

$P = \gamma_{lf} . h + 0 - 0$ (kPa, Psi)

where: $\gamma_{lf}$ = weight per unit volume of the liquefied fill material

### 6.2.3. Bulkhead Design

Design of bulkheads for mine backfill requires more experience than theoretical calculation. There are three main designs for bulkheads under mining conditions. These are discussed below.

#### 6.2.3.1. Design of free draining timber bulkheads

This consists of vertical timbers spaced about 1.3 cm. apart for drainage and bearing on laminated beams which are bolted to the drift floor and back (Figure 6-5). Burlap or synthetic filter fabric is usually fastened to the stope side of the timbers and sealed to the perimeter with quick setting cement.

Timber members in bulkheads will generally behave as uniformly loaded beams. Some members could behave as concentrate loaded beams at mid-span. Some braces will behave as a short column. Size of the timbers is determined generally by the allowable working stresses in shear, bending or deflection. For bulkhead construction, deflection should not exceed 1/240 of the timber length (or span). Therefore, for a 3m. span the permissible maximum deflection is 1.25 cm[63]. In short, heavily loaded beams, the horizontal shear is likely to control the beam size. In long beams, deflection may control the size determination.

Design specifications for timber bulkheads modified from Smith and Mitchell[64] are as follows:

$A_b$ = End bearing area of the timber bulkhead, cm²

$$= FS . P . W_w . \frac{h_d}{2} \ \sigma_c$$

where: FS = factor of safety for the bulkhead = 3 to 5

P = maximum pressure on the bulkhead, kPa.

$W_w$ = load width of the timber, cm = timber width, ($t_w$) + spacing between timbers, (s).

$h_d$ = height of the excavation where the bulkhead is to be erected, cm.

$\sigma_c$ = compressive strength of the timber perpendicular to its grain, kPa.

$L_b$ = bearing length of the timber (cm) = $\frac{A_b}{t_w}$

The timbers should be bolted spaced at either:

d = 40 cm. centres using > 19 mm. dia. rockbolts, or

d = 30 cm. centres using > 19 mm. dia. rockbolts.

Pins can be placed behind these timbers to increase the support capacity; but pinning without rockbolting is not recommended because of lack of resistance to timber rotation. The bolts should be tightened sufficiently to develop friction between the timber and rock surface to prevent rotation.

$\tau_a$ = max. shear stress across laminated timber beams between bolts, kPa.

$= P.h_d / 2\,\tau_l\,.\,L^3$

where: L = timber length, cm.

$\tau_l$ = max. longitudinal shear stress along the timber, kPa.

$$= 1.5\,P\,.\,W_w \frac{(h - E_i)}{2\,\tau_w}\,.\,L$$

$E_i$ = end bearing influence of the timber = 65 to 85, cm.

At drawpoint or drift openings higher than about 4.3m, the max. fibre stress for timbers would be high. This can occur where bulkheads must be built close to the brow of a drawpoint. Under such conditions, the length of the drift plug is reduced or eliminated. Therefore, to reduce bending of the timber under the fill pressure, external bracing is required. Steel beams are better than timber for this purpose. The beams should be notched into the drift walls at mid - height of the opening with sufficient bearing area to carry 50% of the design loading (P). The depth of the bearing notches depends on the rock quality, and rock bolts may also be necessary if the rock is fractured or contains bedding planes.

This part of the design is site specific and each location must be examined by an engineer with ground control experience. When the setback of the bulkhead from the drawpoint is zero, the design pressure (P) = 0.5 $\gamma$.h. In cases where strong arching is provided by good quality rock fill in drawpoint cones, above the drift opening, the actual P-value may be less than 0.5 $\gamma$.h.

A central bracing system can often be considered temporarily for the purpose of resisting seepage water pressures in the fill during pouring operations. The bracing can be removed for use elsewhere after a fill pour has drained. Recharge of a drained fill can be avoided by pouring an impervious layer or a thin cemented tailings layer on top of each fill pour. After completion of the fill drainage, i.e. monitored fill water pressure behind the bulkhead approaches zero, the central bracing system can be dismantled for use elsewhere.

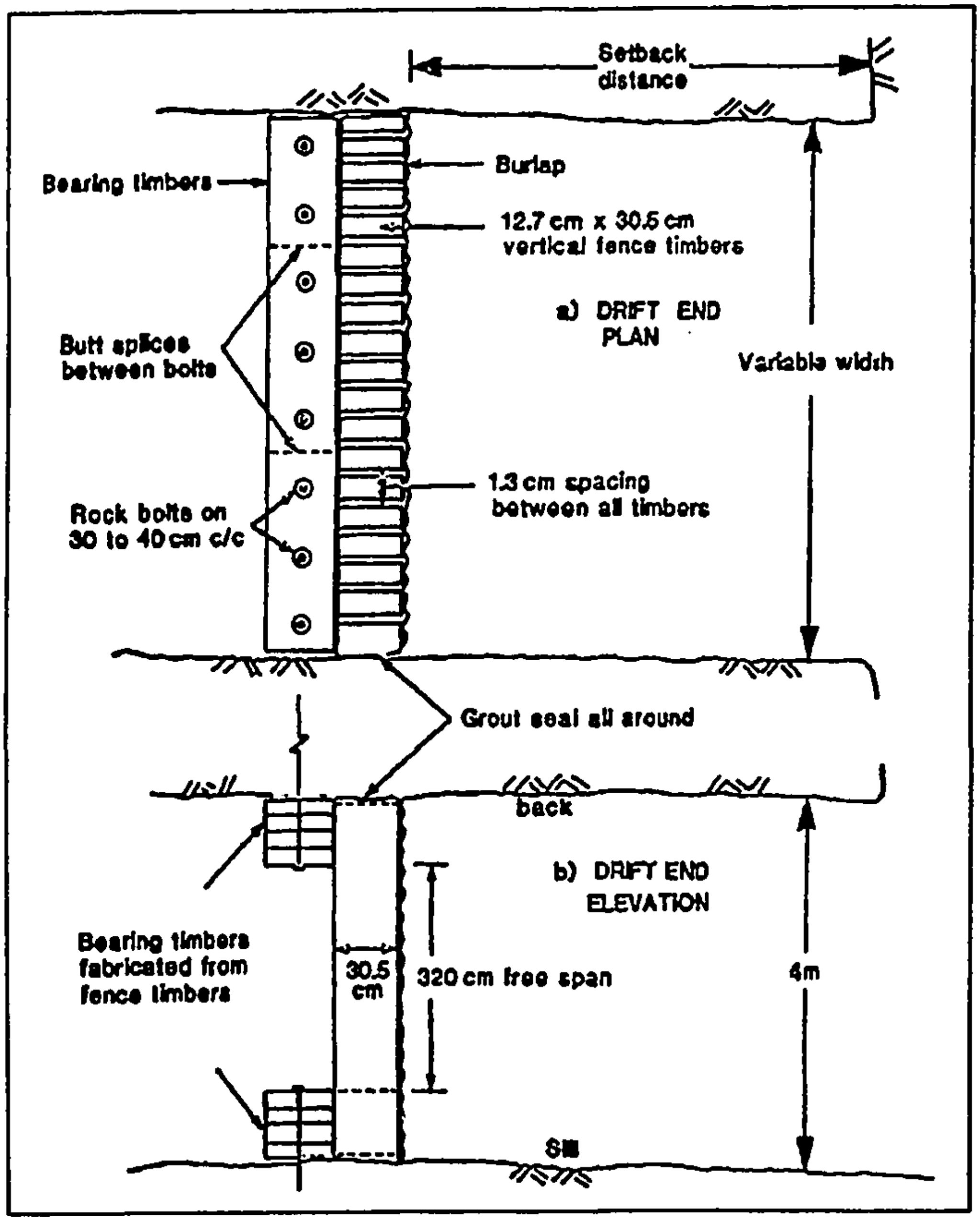



Figure 6-5 Timber bulkhead design

Removing and reusing timber bracing is time-consuming and results in damage to timbers. However, temporary metal braces could be designed for easy removal and reinstallation, where required. The metal braces should be light enough for handling. It comprises adequate numbers of 1 to 1.5 m length telescopic steel pipes. A horizontal beam on the timber bulkhead serves as a connector between the telescopic braces and the timber bulkhead. It is screwed and bolted as required. Variations in the types of bulkhead bracing and steel bearing plate systems is shown in Figures 6-6 to 6-9.

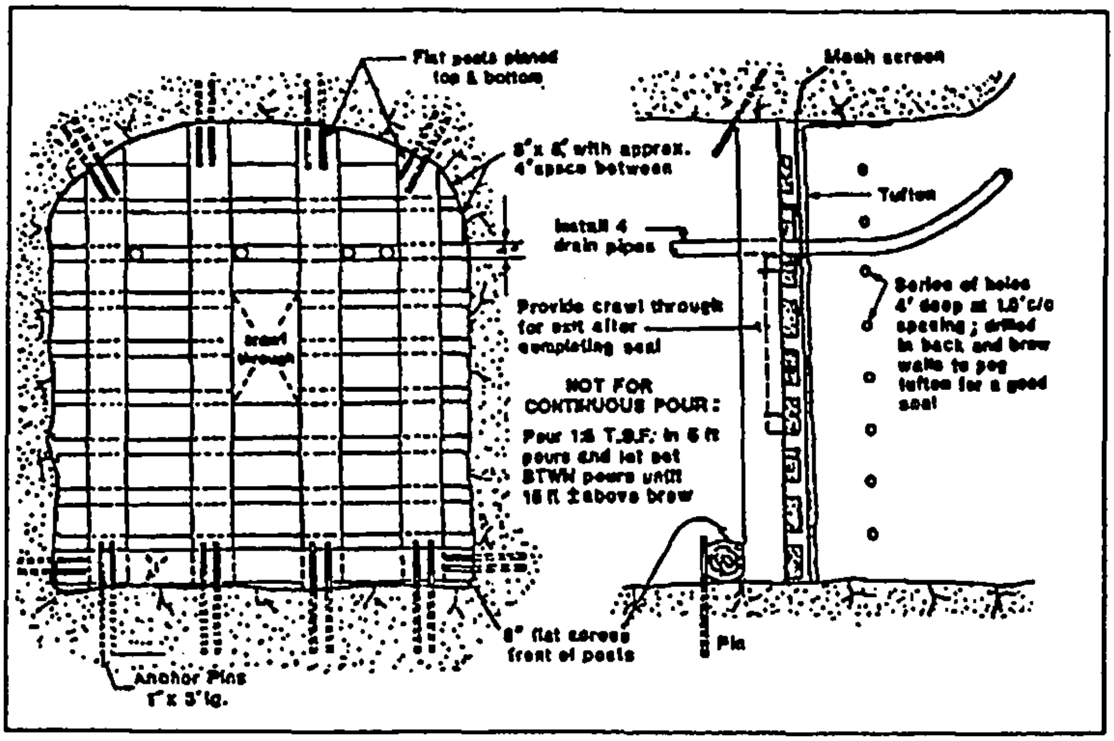



Figure 6-6 Fill bulkhead for blasthole stope in Falconbridge mines

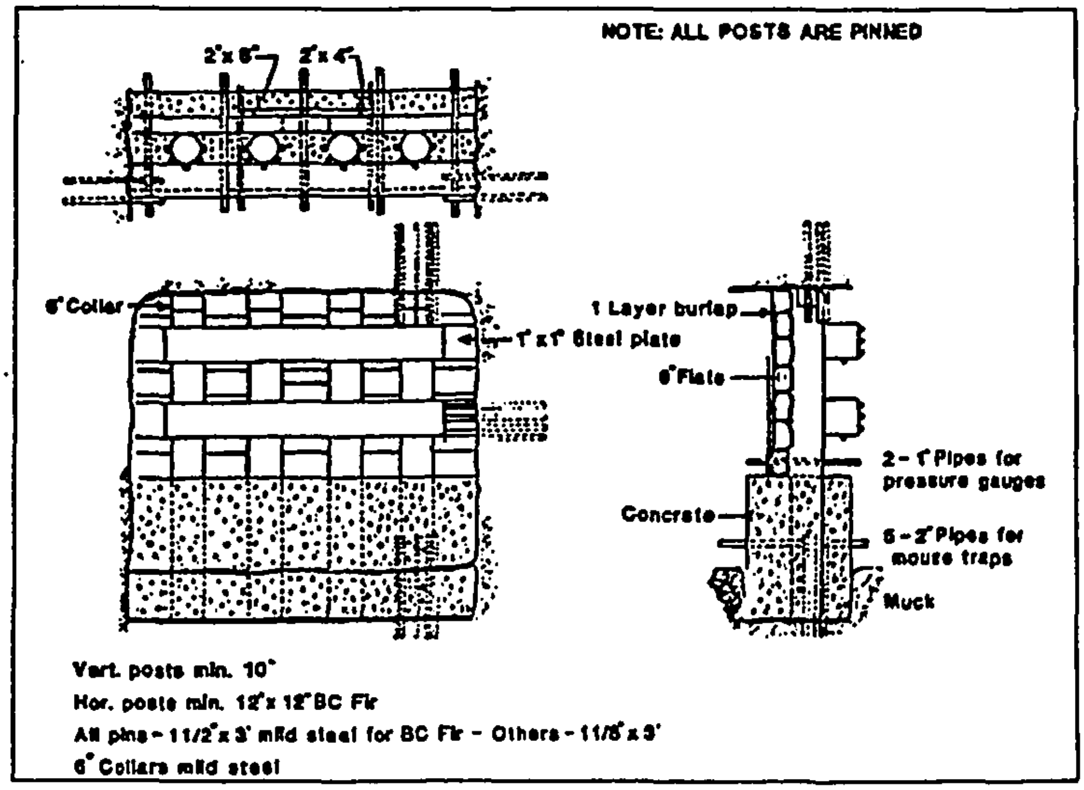



Figure 6-7 Bulkhead steel bearing plates in Dome mines

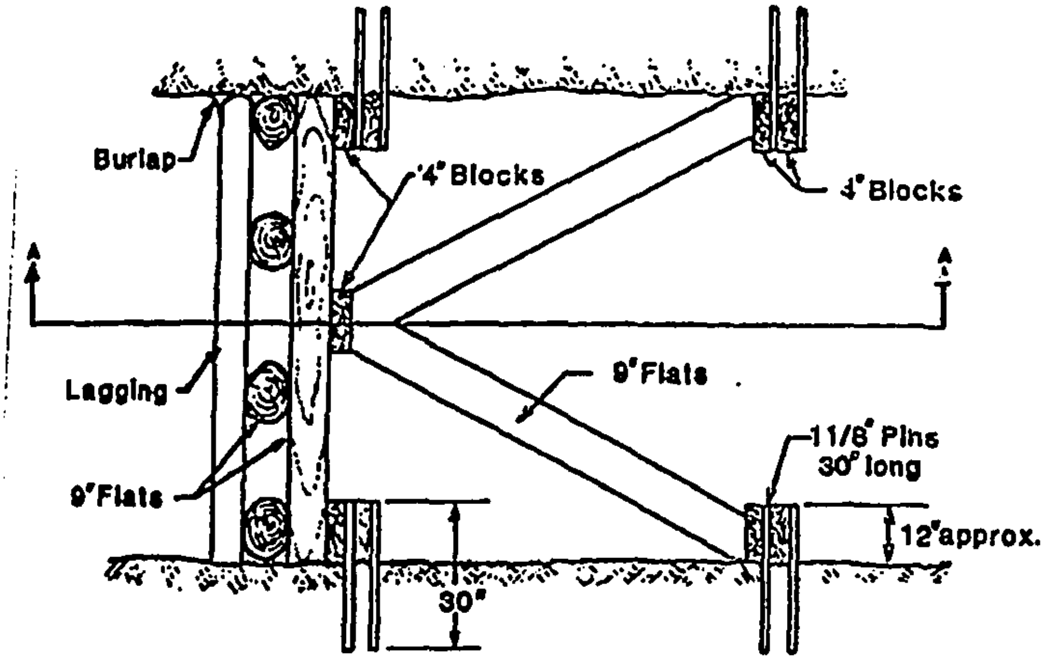



Figure 6-8 Bulkhead bracing system in Falconbridge nickel mines.

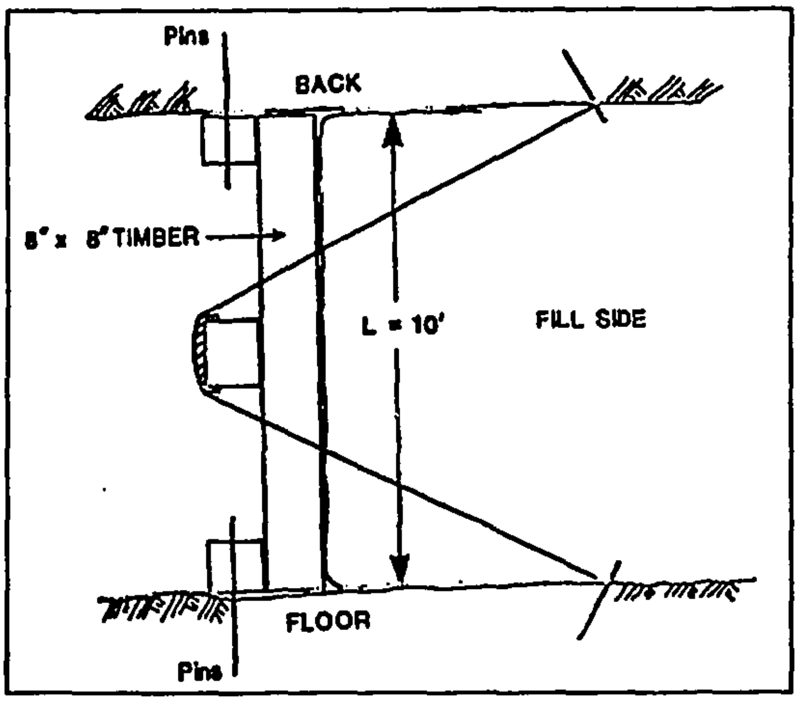



Figure 6-9 Use of steel cables as bracing for wooden bulkheads in the Horne mine.

### 6.2.3.2. Design of concrete and reinforced concrete bulkheads

A conservative alternative to free draining timber bulkheads is a massive concrete bulkhead structure capable of withstanding the fully liquefied hydrostatic fill pressure. The following conditions may occur:

(a) If mousetrap drains behind a concrete bulkhead perform adequately, the bulkhead will experience very little pressure.

(b) If the drains do not perform adequately, or become blocked, the hydrostatic pressure can build up to the full fill height because the drainage water has no alternative escape route.

(c) If the fill is water saturated, it is subject to blast liquefaction. This worst condition must be considered in the design of impervious concrete bulkheads. In this case, monitoring (e.g. piezometer installations) is to be carried out to establish a control on the hydrostatic pressure, and to limit (or discontinue) the water saturated fill pouring rate according to the piezometric levels.

Considering the worst condition (case c) and Figure 6-10.

$P$ = the design pressure on the bulkhead, kPa.

$= FS \, . \, \gamma \, . \, h$

where: $FS$ = factor of safety for the bulkhead = 3 to 5.

$\gamma$ = weight per unit volume of saturated fill = 20 to 30, $kN/m^3$.

$h$ = height of the fill, m.

Following is examples of possible design parameters:

1) Anchorage distance of T/2 is based on anchorage in sound rock with an allowable bearing pressure of 3.8 MPa. If rock is fractured or allowable bearing is less than 3.8 MPa, then appropriate adjustment to anchorage depth is required.
2) Concrete cover = 75 mm, ± 5 mm.
3) For rectangular bulkheads, place reinforcement parallel to shortest dimension on outside.
4) Bulkheads which may be loaded from either side, place reinforcement, as indicated in tables, on both sides.

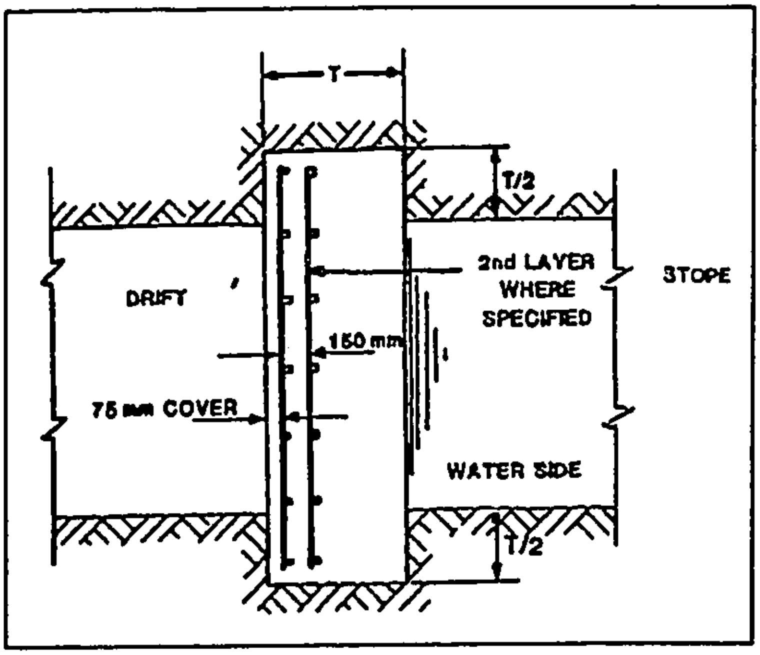



Figure 6-10 Cross section of a typical concrete bulkhead

The concrete bulkheads should be provided with internal mousetrap drains and outlet pipes to provide pore drainage. If the drains work satisfactorily, only a small proportion (about 2 to 10%) of the design pressure will be attained. If they become plugged, the design pressure may be attained but the fill will not drain adequately and transport water will pond on the fill surface.

### 6.2.3.3. Design of Arched concrete Block Bulkheads

This type of bulkhead has been used successfully in Australia. Construction is fast, and materials are readily available. Usually a pad is poured at the bulkhead location, and sometimes hitches are made in the drift walls. Since concrete block construction is weak in shear and tension, an arch design is required for a mortared block bulkhead. Dimensions of the concrete blocks are 20 x 20 x 40 $cm^3$ with 3 holes. Two holes blocks are suitable, but would not provide as much concrete area. These blocks would be mortared in place with the holes horizontally oriented to provide drainage ports, as shown in Figure 6-10. A 20 cm arch depth is sufficient for drifts up to 4.3 m. wide. For openings wider than 4.3 m., a double row of blocks or buttress walls could be constructed. Blocks would be mortared in place to form the arch. Filter screens are to be placed on the stope side and sealed with mortar on all edges. The screens are wire mesh covered with burlap.

T = the required thickness of arched concrete bulkhead, m= $R \cdot \frac{P}{\sigma_c}$

where: R = radius of the bulkhead arch, m.

P = pressure on the bulkhead, kPa.

$\sigma_c$= compressive strength of concrete block, $= 2 \times 10^3 - 3 \times 10^3$, kPa.

This type of bulkhead construction allows placement of the bulkhead filter screen, drift plug, sand and gravel as the height of the arch are raised. Tight filling between the bulkhead and the waste rock drainage blanket is essential to the stability of bulkhead.

## 6.2.4. Bulkhead Pressure Measurement

Total pressure cells are commercially available but would not generally be used for underground bulkhead pressure measurement. This is due to their high capital cost, the fragility of the cells and highly technical nature of the measurement techniques. A simple, but meaningful, direct bulkhead pressure measurement can be made by careful measurement of bulkhead defections, using a dial gauge located at the mid span of the bulkhead beam.

The measuring device consists of a pole with fixed location points and a dial gauge with ± 0.025 mm. accuracy. The measuring device can be moved on to various bulkheads.

P = pressure on the bulkhead, kPa.

$$= \frac{384\, E \cdot I \cdot \Delta}{5 \times 14\, L^4}$$

where: E = average deflection modulus, $= 7 \times 10^6$ to $9 \times 10^6$, kPa for spruce timber.

I = moment of inertia of the beam, $m^4$.

L = free span or length of the beam, m.

$\Delta$ = mid - length timber deflection, m.

At a design load of P = 96.5 kPa (14 psi ) on the spruce timber bulkhead, the timber deflection ($\Delta$) would be 7 to 8 mm. If the difference between the zero dial gauge reading (after the bulkhead construction) and the current reading at any location exceeds 8 mm, the timber bulkhead pressure has reached its maximum design value of 96.5 kPa. An alternative measurement technique would replace the movable measurement pole and dial gauge with a fixed taunt wire stretching between the upper and lower lag screw locations. On initial installation, the allowable deflection (8 mm) can be set as a spacing between the wire and the central reference point.

Visual inspection and measurement of the distance between the wire and the central reference point on a regular basis, will establish if that reference bulkhead pressure would be considered to have attained the allowable limit. In addition to these measurements, bulkheads should be inspected by a supervising engineer during and after construction to ensure that specifications have been met. Inspection should also be continued during the stope backfilling, noting whether the bulkheads appear to be stressed and ascertaining that there is no adverse deformation such as cracking of bulkhead, tilting or rotation.

### 6.2.5. Records of Bulkhead System

The mine operators should keep adequate records of all phases of the bulkhead system in a log book. The following information is suggested to be recorded:

- Bulkhead identification, type and location.
- Date, dimensions and details of construction.
- Dimensions of the stope and bulkhead drift.
- Degree of saturation of fill in each stope.
- Height of fill in each stope.
- Rate of fill placement and its density.
- Water input and output by stope.
- Any unusual or abnormal behavior of the bulkhead system.

### 6.2.6. Costs of Bulkheads

Bulkhead construction can be expensive and account for 45% of the total operating costs in a cut and fill operation. For long hole mining with hydraulic and waste rock fill the costs are 2 - 7% of the total operating costs. Bulkhead construction is very labor intensive and can take many shifts to complete. Table 6-3 shows results from the Quebec Mines Survey related to bulkhead costs.

Table 6-3 Cost data on Bulkheads used in Quebec Mines

| MINE | TOTAL ORE BKF (mt) | MINING METHOD | B.FILL MAT. (mt) | TOTAL FILL (mt) | NO OF BLKHDS | FILL PER BLKHD mt/BLK | BLKHD COSTS ($) | BLKHD COSTS $/mt FILL | BLKHD COSTS $/mt ORE |
|---|---|---|---|---|---|---|---|---|---|
| 11 | 169000 | C & F | T | 95100 | 333 | 286 | 2862 | $10.02 | $5.64 |
| 1 | 195000 | C & F | T | 117000 | 144 | 813 | 2375 | $2.92 | $1.75 |
| 12 | 49500 | C & F | T | 34320 | 98 | 350 | 3065 | $8.75 | $6.07 |
| 3 | 397000 | C & F | T | 231570 | 315 | 735 | 1870 | $2.54 | $1.48 |
| 10 | 116000 | C & F | S | 68000 | N/A | N/A | N/A | N/A | N/A |
| 16 | 700000 | L.H. | T | 337600 | 0 | — | 0 | $0.00 | $0.00 |
| 15 | 363533 | L.H. | T + R | 242355 | 0 | — | 0 | $0.00 | $0.00 |
| 13 | 450000 | L.H. | R | 225000 | 0 | — | 0 | $0.00 | $0.00 |
| 2 | 334310 | L.H. | R + S | 217325 | 44 | 4939 | 1254 | $0.25 | $0.17 |
| 14 | 272357 | L.H. | R | 248202 | 18 | 13789 | 7500 | $0.54 | $0.50 |
| 6 | 426000 | L.H. | T | 203857 | 14 | 14561 | 3396 | $0.23 | $0.11 |
| 9 | 385500 | L.H. | R | 281415 | 19 | 14811 | 2366 | $0.16 | $0.12 |
| 7 | 255000 | L.H. | T + R | 134110 | 8 | 16764 | 7975 | $0.48 | $0.25 |
| 17 | 700000 | L.H. | S | 250500 | 12 | 20875 | 5000 | $0.24 | $0.09 |
| 18 | 700000 | L.H. | R | 304000 | 12 | 25333 | 5000 | $0.20 | $0.09 |
| 8 | 900000 | L.H. | T + R | 900000 | 18 | 50000 | 6000 | $0.12 | $0.12 |

(D) - DELAYED BACKFILL    T - TAILINGS
(C) - CYCLING BACKFILL    R - ROCK
(D)* - PROJECTED    S - SAND

For reference purposes the average cost of bulkheads per metric tonne of fill for rockfill, hydraulic fill and cut & fill mining are $0.30, $0.24 and $6.06 respectively.

# CHAPTER 7

# THE ECONOMICAL EVALUATION

## 7.0. INTRODUCTION

Cost-related aspects of backfill design have been examined by different authors in recent years. It is generally agreed that variations in cost analysis are due to different site selection criteria, as well as physical and chemical characteristics of fill material[65]. Moreover, general agreement on the backfill-related costs identification can hardly be reached in the practical backfill operation. It is even impossible to predict the backfill related cost by sum of all the cost items because cost items recognized by one mine may not be recognized by other mines. The site specific nature of each operation prevents any formulation of a general cost distribution model for the industry. Therefore, the cost model can still be established under statistical base for regional application on the feasibility study level, provided certain definitions of backfill related cost are clarified. The following sections of the chapter present the results of a survey of backfill operation conducted within Quebec mining industry. Based on the survey, backfill-related costs are clearly defined and statistical models for regional cost estimation are presented. The following discussion is not intended to standardize the cost prediction model in any general purpose, but rather as an attempt to reach a simple regional cost model in principle, and use it only for the purpose of feasibility study. Further studies are needed to fully address all aspects of backfill-related cost. As claimed at the very beginning, the purpose of this research is to build a computer-based system for backfill. The specification so far is to cover various aspects of backfill design. Some of the aspects discussed will be implemented in a prototyping system for backfill design, which opens an architecture to integrate different tools to support some decision makings involved.
In this survey, the capital cost and operation cost are based on the assumptions described in the following .

**1. Operation cost**

The operation costs are based on 7 cost items that can be compared from site to site:

1. Material: This item refers to the direct costs involved in the production of the backfill material required for the operation;

2. Haulage(S): This item refers to any kind of direct transport cost involved at surface;
3. Slurry: This item refers to any direct cost related to the surface cement slurry , hydraulic fill, high density fill etc.;
4. Haulage(U): This item refers to any direct cost related to underground transportation of backfill material to the appropriate stope. It means placement cost in the case of hydraulic fill or high density fill;
5. Bulkheads: This item refers to any direct cost related to the bulkheads construction;
6. Binding: This item refers to the cost of any binding agent (Portland cement, Fly ash, Accelerator, etc.);
7. Other: This item refers to all others direct costs related to the backfill operation. Those included monitoring, dewatering, and cleaning cost.

**2. Capitalization costs**

The capital costs are based on 8 cost items that can be compared from site to site:

1. Fill plant: This item refers to all the surface installations needed for the cement slurry, the hydraulic fill or the high density fill preparation. This does not include cyanide treatment when required;
2. Boreholes: This item refers to any frill holes needed for the cement slurry, hydraulic fill or high density fill placement;
3. Piping: This item refers to any piping material needed for the cement slurry, hydraulic fill or high density fill placement;
4. Material pass: This item refers to raise construction;
5. Loading station: This item refers to all construction and equipment needed underground to prepare fill material for transportation;
6. Transportation unit (U/G): This item refers to the total costs of scooptrams and trucks needed for underground transportation. This does not apply for rail transportation;
7. Transport unit(S): This item refers to the total costs of truck loader needed for surface transportation of the fill material;
8. Other: This refers to all other unspecified material needed for backfill operation.

Environmental assessment and impact studies represent an additional cost to any mining activity. These are lengthy procedures, which might have to continue well after the actual mining operation has started[66]. The cost of an environmental study is directly related to the complexity and variations associated with the type of investigation that is necessary. As far as this survey is concerned, the cost of environmental assessment is not included in either capital or operation cost, even though it is indeed an important aspect to be taken into account.

## 7.1. QUEBEC MINES SURVEY CASE DESCRIPTION

The aim of this survey is to monitor the practical mining industry and build tools needed to evaluate the economic and technical feasibility of backfilling projects. One of the essential steps in the process was to gather and collate data relating to the experience and practice of backfilling in Quebec.

At the beginning of this project, a questionnaire was sent to 26 underground operations. The responses from mining fields showed that 18 mine operations have used, are using, or project to use backfill. Sixteen mines responded, and the documented information is presented in the report 'Economical and technical feasibility evaluation for backfill design in underground mines. Based on this report, a Quebec regional cost model of backfill operation is integrated as part of the framework of computer assistant backfill design and evaluation system. The reported mining operations include:

- 5 cases of Rockfill;
- 4 cases of cut & fill;
- 3 cases of hydraulic fill;
- 3 cases of rock and hydraulic fill;
- 1 case of high density fill;

From the previous mentioned data, the total backfill production cost and the total backfill capitalization were divided into 7 and 8 cost elements respectively, that can be compared from site to site. Backfill production and capitalization cost models have been elaborated for: 1. Rockfill, 2. Tailings and Rockfill, 3. Tailings or Sandfill for cut and fill operation. The cost models presented here should be considered as conceptual models where the user will find an estimation of the total backfill operation cost at +/- 20% and capitalization cost at +/- 30 in $ of 1990. This level of precision represents the first cost evaluation stage of any engineering project. So, these cost models should be used as an estimation for comparison of alternatives. The information gathered is served to establish the design procedures and practical criteria for the evaluation of the technical and economic feasibility of backfilling projects.

## 7.2. SURVEY ANALYSIS

### 7.2.1. Mining methods

It has been observed that :

- 74% of total ore is excavated by using the long holes method;
- 21% of total ore production by using the Cut and Fill method;
- 5% of total production by using other mining method;

### 7.2.2. The preparation of cement slurry and hydraulic fill

The survey results indicated that the main material used in Quebec is waste rock, tailings and sand. Shown in figure 7-1 is the distribution of material used for backfill. The hydraulic fill is prepared directly in the mill and the cement slurry is most often prepared in a specific backfill plant. Illustrated in table 7-1 and 7-2 are the statistical data concerned.

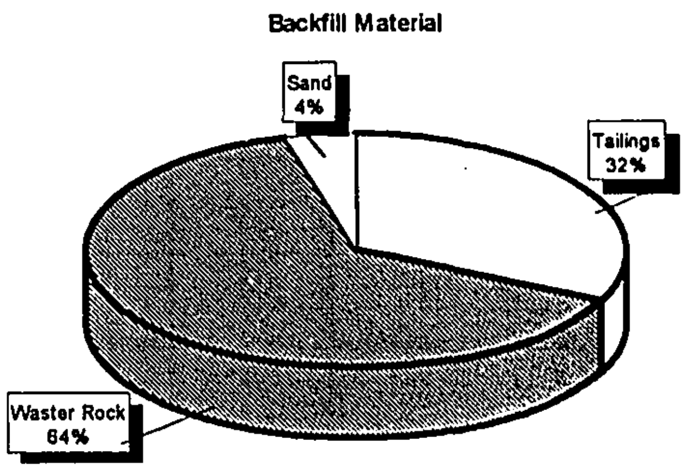



Figure 7-1 The Distribution of material used for backfill

Table 7-1 The summary of material:

| Hydraulic fill | | | |
|---|---|---|---|
| Unit | mt | % solid | Density |
| Average | 295518 | 63 | 1.70 |
| **Cement Slurry** | | | |
| Unit | mt | % solid | Density |
| Average | 12399 | 60 | 1.66 |

Table 7-2 The summary of the material preparation costs in terms of hydraulic fill:

| Description | Min $/mt | Average $/mt | Max $/mt | Std $/mt |
|---|---|---|---|---|
| Hydraulic fill | $0.46 | $0.87 | $1.09 | $0.22 |
| Cement slurry | $0.14 | $0.3 | $0.44 | $0.11 |

### 7.2.3. Fill transportation

The hydraulic backfill transportation system is usually implemented through pipes of 10 to 15 cm in diameter by gravity effect or with pump. As a general rule, the horizontal distance should be no less than 10 times of the vertical height. Pipe length varies widely. The transportation cost, in most cases is equal to pipe maintenance cost. The rockfill transportation is carried out by trucks and rail cars. Presented in table 7-3 is the data related to transportation:

Table 7-3 The summary of rockfill transportation observed:

| Transport observation | | | |
|---|---|---|---|
| | D(m) | $/mt | mt/hr/unit |
| Mine13 | 110 | 2.80 | 63 |
| Mine02 | 250 | 2.39 | 36 |
| Mine07 | 215 | 2.41 | |
| Mine02B | 250 | | 75 |
| Mine15 | 300 | 2.54 | |
| Mine06 | 330 | 2.50 | |
| Mine09 | 400 | 3.38 | 35 |
| Mine14 | 2000 | 5.00 | 18 |
| Mine11 | 2400 | 4.75 | 11 |

### 7.2.4. Bulkheads

According to the survey, bulkheads used are sufficiently safe, and the present design is adequate, despite of variations in terms of materials used and construction costs. Bulkheads are made of wood, or cement blocks, or steel rods covered with impermeable material. Presented in table 7-4 and 7-5 are the basic results with regard to the cost of bulkhead design.

Table 7-4: The summary of bulkheads cost for cut and fill stopes

| | Mine #01 | Mine #11 | Mine #03 | Mine #12 | |
|---|---|---|---|---|---|
| Bulkhead($/mt) | 2.92 | 10.11 | 2.54 | 8.75 | |
| Total bkf(mt) | 117000 | 95100 | 231570 | 34320 | |
| ToT. blk | 150 | 358 | 348 | 97 | |
| mT/BLK | 778 | 266 | 666 | 354 | 516 |
| Bulkhead($) | 2272 | 2686 | 1693 | 3096 | 2436 |

Table 7-5: The summary of bulkhead cost for long holes stopes

| | Mine #02 | Mine #06 | Mine #07 | Mine #09 | |
|---|---|---|---|---|---|
| Bulkhead($/mt) | 0.25 | 0.18 | 0.48 | 0.16 | 0.27 |
| Total bkf(mt) | 217325 | 203857 | 134110 | 281000 | |
| ToT. blk | 44 | 14 | 8 | 19 | |
| mT/BLK | 4939 | 14561 | 16764 | 14789 | 12763 |
| Bulkhead($) | 12355 | 2621 | 8047 | 2366 | 3567 |

### 7.2.5. Dewatering

The survey conducted indicates that pumping the backfill water represented only a very small fraction of total pumping and consequently, of total pumping cost. Very few special dispositions are taken for evacuating backfill water. It is estimated that backfill water represents less than 8% of total water pumped on surface. No problems of liquefaction were noted.

## 7.3. COST ANALYSIS

Illustrated in table 7-6, 7-7, 7-8, 7-9, 7-10 and the corresponding figures are average production cost distribution for various backfill type according to the survey in terms of percentage of total backfill production cost.

Table 7-6: The average operation cost distribution for rockfill:

| # | Description | Min $/mt | Average $/mt | Max $/mt | Std $/mt | (n) |
|---|---|---|---|---|---|---|
| 1 | Material | 4.00 | 5.03 | 6.87 | 1.10 | 4 |
| 2 | Haulage (s) | 0.08 | 0.49 | 1.04 | 0.37 | 7 |
| 3 | Slurry | 0.14 | 0.30 | 0.44 | 0.11 | 5 |
| 4 | Haulage(u) | 0.99 | 2.82 | 3.38 | 1.20 | 10 |
| 5 | Other | 0.22 | 0.35 | 0.64 | 0.18 | 5 |
| 6 | Bulkhead | 0.16 | 0.28 | 0.54 | 0.15 | 6 |
| 7 | Binding | 0.81 | 2.11 | 3.73 | 1.01 | 8 |
| 8 | Total cost (MT) | 4.66 | 7.44 | 10.03 | 2.06 | 5 |
| 9 | Total cost $M^3$) | 9.69 | 14.87 | 18.93 | 4.14 | 5 |

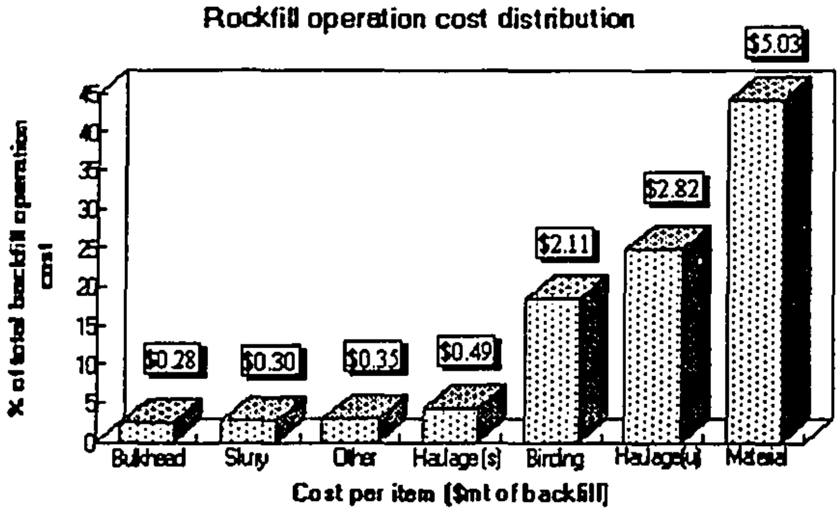



Figure 7-2 The operation cost distribution of rockfill

Table 7-7: The average operation cost distribution for tailings and rockfill:

| # | Description | Min $/mt | Average $/mt | Max $/mt | Std $/mt | (n) |
|---|---|---|---|---|---|---|
| 1 | Material | 4.00 | 5.03 | 6.87 | 1.10 | 4 |
| 2 | Haulage (s) | 0.08 | 0.49 | 1.04 | 0.37 | 7 |
| 3 | Slurry | 0.46 | 0.83 | 1.09 | 0.25 | 5 |
| 4 | Haulage(u) | 0.99 | 2.82 | 3.38 | 1.20 | 10 |
| 5 | Other | 0.22 | 0.30 | 0.64 | 0.19 | 6 |
| 6 | Bulkhead | 0.16 | 0.31 | 0.54 | 0.15 | 6 |
| 7 | Binding | 0.81 | 2.11 | 3.73 | 1.01 | 8 |
| 8 | Total cost (MT) | 5.31 | 7.02 | 9.24 | 1.64 | 3 |
| 9 | Total cost $M^3$) | 12.63 | 14.44 | 19.27 | 3.45 | 3 |

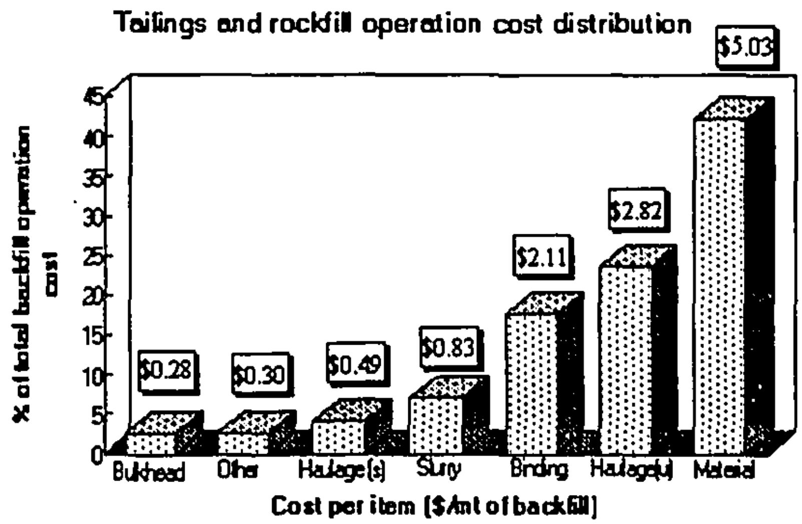



Figure 7-3 Tailings and rockfill operation cost

Table 7-8: The average operation cost distribution for tailings or sandfill operation:

| # | Description | Min $/mt | Average $/mt | Max $/mt | Std $/mt | (n) |
|---|---|---|---|---|---|---|
| 1 | Material | 3.00 | 3.00 | 3.00 | 0.00 | 2 |
| 2 | Haulage (s) | 1.15 | 1.15 | 1.15 | 1.15 | 2 |
| 3 | Slurry | 0.05 | 0.83 | 1.09 | 0.25 | 5 |
| 4 | Haulage(u) | 0.08 | 0.29 | 0.50 | 0.20 | 3 |
| 5 | Other | 0.10 | 0.10 | 0.10 | 0.00 | 1 |
| 6 | Bulkhead | 0.16 | 0.31 | 0.54 | 0.15 | 6 |
| 7 | Binding | 3.27 | 4.34 | 5.40 | 1.07 | 2 |
| 8 | Total cost (MT) | 4.67 | 7.82 | 10.97 | 3.15 | 2 |
| 9 | Total cost M3) | 8.11 | 13.10 | 15.40 | 4.99 | 2 |

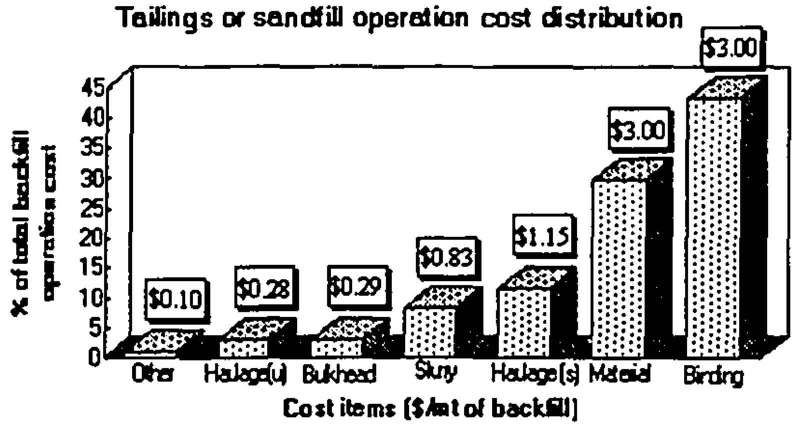



Figure 7-4 Tailings or sandfill operation cost distribution

Table 7-9: The average operation cost distribution for cut and fill:

| # | Description | Min $/mt | Average $/mt | Max $/mt | Std $/mt | (n) |
|---|---|---|---|---|---|---|
| 1 | Material | 0.00 | 0.00 | 0.00 | 0.00 | 4 |
| 2 | Haulage (s) | 0.00 | 0.00 | 0.00 | 0.00 | 4 |
| 3 | Slurry | 0.72 | 0.93 | 1.04 | 0.15 | 3 |
| 4 | Haulage(u) | 1.20 | 5.90 | 8.89 | 3.80 | 4 |
| 5 | Other | 0.55 | 0.55 | 0.55 | 0.00 | 8 |
| 6 | Bulkhead | 2.54 | 6.08 | 11.32 | 3.39 | 4 |
| 7 | Binding | 0.57 | 1.68 | 4.43 | 1.61 | 4 |
| 8 | Total cost (MT) | 7.66 | 14.89 | 21.47 | 6.67 | 4 |
| 9 | Total cost M3) | 13.12 | 26.87 | 40.11 | 12.62 | 4 |

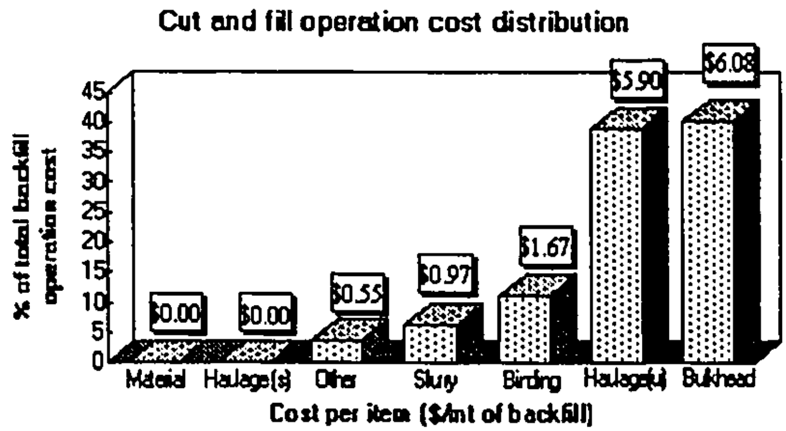



Figure 7-5 Cut and Fill cost operation cost distribution

Table 7-10: The average operation cost distribution for high density fill:

| No | Description | $/mt |
|---|---|---|
| 1 | Material | 0.00 |
| 2 | Haulage(s) | 0.00 |
| 3 | Slurry | 2.27 |
| 4 | Placement | 1.26 |
| 5 | Bulkhead | 0.00 |
| 6 | Other | 0.00 |
| 7 | Binding | 2.70 |
| 8 | Total cost(mt) | 6.23 |
| 9 | Total Cost($M^3$) | 13.85 |

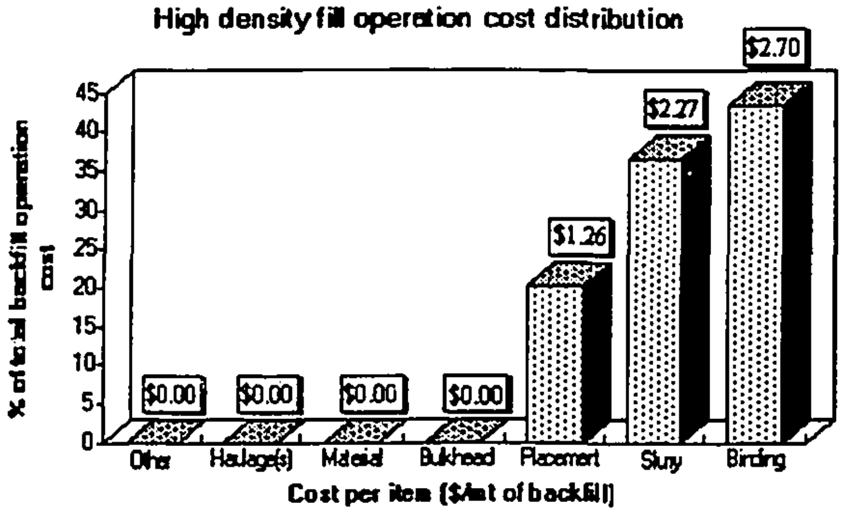



Figure 7-6 High density fill operation cost distribution

Presented in table 7-11 is the typical capitalization cost distribution of overall backfill operation in Quebec regional area.

Table 7-11: The average backfill capitalization cost distribution:

| No | | Min($) | Average($) | Max($) | Std($) |
|---|---|---|---|---|---|
| 1 | Fill plant(R) | 25000 | 437625 | 701500 | 167959 |
| 1 | Fill plant(H) | 317000 | 919250 | 1360000 | 384059 |
| 2 | Boreholes | 0 | 99537 | 168000 | 64054 |
| 3 | Piping | 7300 | 44935 | 109250 | 37600 |
| 4 | Rock pass | 225000 | 493595 | 934375 | 327953 |
| 5 | Loading station | 109250 | 225862 | 375000 | 145985 |
| 6 | U/G transport | 292000 | 577250 | 1050000 | 254498 |
| 7 | surf transport | 188678 | 234400 | 280000 | 46 00 |
| 8 | other | 39750 | 51583 | 115000 | 47688 |
| 9 | total capital(R) | 1900000 | 2214818 | 2544375 | 263271 |
| 9 | Total Capital (H) | 1295000 | 1559750 | 1824500 | 264750 |
| 9 | Total Capital (H+R) | 1295000 | 1888159 | 2544375 | 405869 |

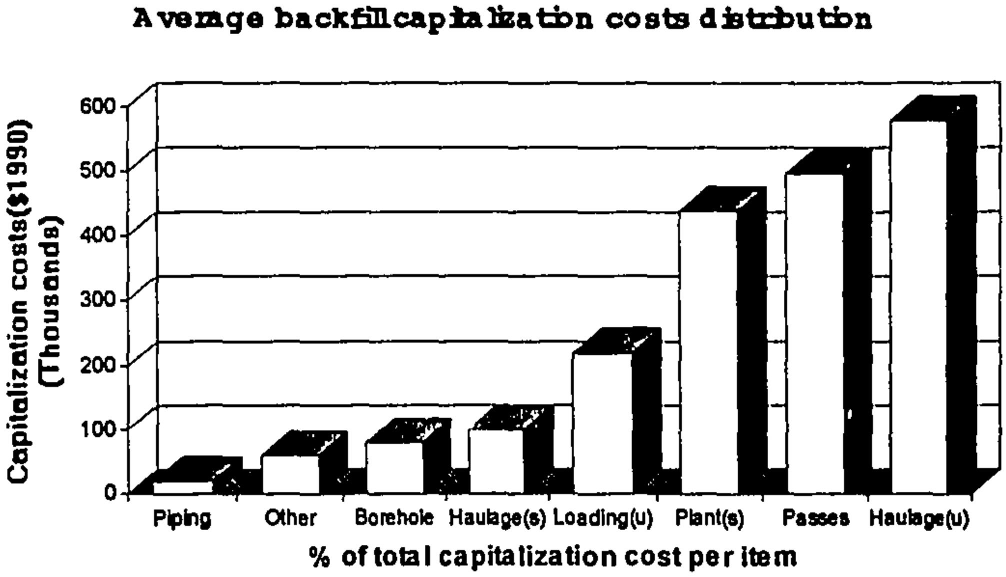



Figure 7-7 Average backfill capitalization cost distribution

Illustrated on the figure 7-8 is a typical capitalization cost distribution for a rockfill, figure 7-9 is the average capitalization cost distribution for tailings or sandfill.

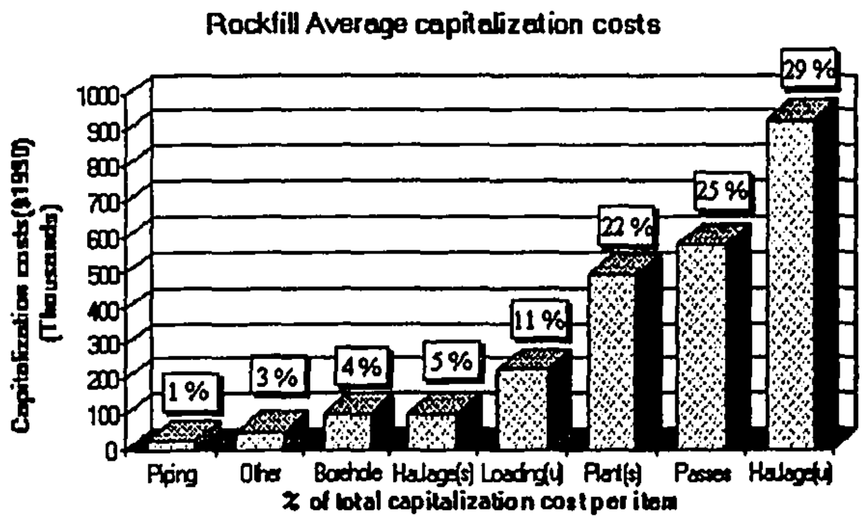



Figure 7-8 Average Rockfill capitalization cost distribution

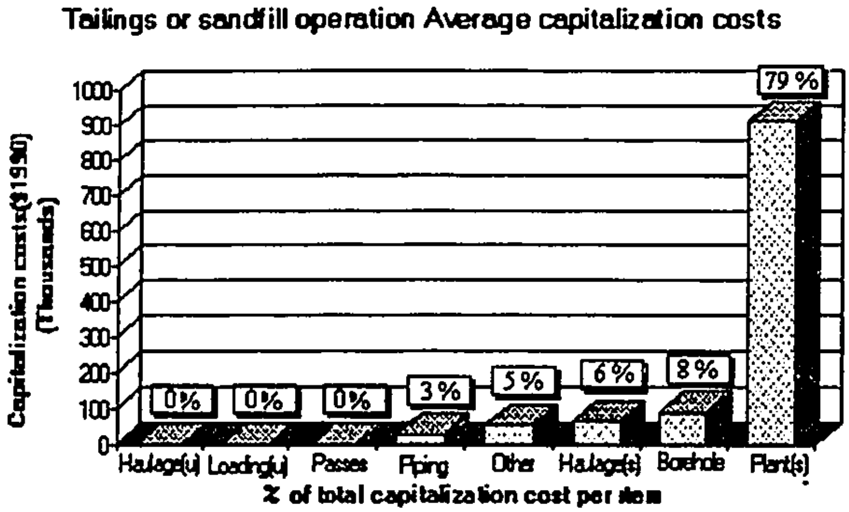



Figure 7-9 Average tailings or sandfill capitalization cost distribution

## 7.4. COST ESTIMATION MODEL

As stated earlier, the total backfill costs are divided to operation cost and capitalization cost. Because the operations required differ greatly according to fill type and mining method, cost models should be considered in line with the backfill type. In this section, four cost models are discussed; they are:

1). Rockfill operation costs model;
2). Tailings fill operation costs model;
3). sandfill operation costs model;
4). Cut and fill operation costs model.

### 7.4.1. Operation costs modeling

#### 7.4.1.1. Backfill material cost:

The backfill material cost is the function of source. From the survey conducted, the material cost for different rock sources are:

- Surface pit waster rock = \$3.5/MT (Drilling, blasting and mucking)
- U/G waste rock stope =\$7.5/MT
- Development waste rock =\$0.0/MT
- Surface stockpile waste rock =\$0.0/MT

The choice of a specific type of waste rock source is mainly based on:

- The amount of waste rock needed;
- The availability of the waste rock from different sources;
- The life expectancy of the operation.

Tailings material comes directly from the mill. The cost is included in the surface preparation cost.

Sand generally comes from an alluvial deposit some distance from the mill. The average cost of sand material = $2.75 per MT.

#### 7.4.1.2. Surface transportation costs

Based on transportation distance, the following results can be drawn from the survey:

Distance > 1 Km

Surface transportation cost($/MT)=$1.08/MT+[($0.08/MT) x D] where D is in Km;

Distance <1 Km

Surface transportation cost($/MT)=$0.20/MT+[($0.0008/MT) x d] where d is in m;

D and d represent total round trip distance.

#### 7.4.1.3. Surface plant preparation

The average value obtained was $6.21/MT of slurry prepared.

As the average solid concentration in the slurry=60%

Slurry tonnage = Total solid tonnage / 60%

Cost of surface plant preparation of rockfill = [$6.21 x (MT of slurry)]/(MT bkf)

Cost of surface plant preparation of tailing and sand fill = ($/MT of solid material) ={$0.45+[85000/total tailings]}/( total tonnage of backfill)

Cost of surface plant preparation for Cut and Fill operation = $0.93 per MT

(Divided by total solid backfill resulting in the cost of per mt of total solid backfill)

#### 7.4.1.4. Underground transportation

The evaluation of underground transportation is based on an average of the operation costs for different transportation distance regardless of the fill type:

U/G transportation cost($/MT)= $[\ 5.00 + (0.01 \times d)]^{1/2}$

Where: d is the total distance of both ways.

For Cut and Fill operation, the overall underground placement cost is a function of the lift width, lift height, the number of bulkheads per lifts, stope volume, and total volume backfilled. The smaller the lift volume, the shorter the mining cycle, and the higher the operation cost which can be estimated by the following formula:

Total underground backfill cost = A x B x C x D x E

Where: A = 4.00 + { 19/(tope width in m)};

B = 57/(lift length in m);

C = $\{$ (# of fences per lift)/2$\}$ x $\{2.7/$ Cut's height in m$)\}^{0.5}$

D = $\{140000/\text{Total tailings used}\}^{0.4}$

E = $\{12400/(\text{Stope volume in } m^3)^{0.1}$

Hydraulic backfill placement cost = 0.5 x total underground backfill cost.

### 7.4.1.5. Bulkhead costs

Unit bulkhead costs depends mostly on their strength. In this survey it is obtained that the average of 12750 MT of rockfill per bulkhead, at a cost of $3567 per bulkhead. The average bulkhead cost is:

For rockfill:

Bulkhead cost = $0.28 per MT

For tailing and sand fill:

Bulkhead cost = $0.31 per MT

For Cut and fill:

Bulkhead cost = 0.5 x total underground backfill cost.

### 7.4.1.6. Other costs

Other costs which include monitoring, dewatering, cleaning, can be estimated as:

For rock fill:

Other costs = $0.35 per MT

For tailing fill:

Other costs = $ 0.30 per MT

For sand fill:

Other costs = $0.1 pre MT

For Cut and Fill:

Other costs = $0.55 per MT

### 7.4.1.7. Binding and binding cost

The cement cost is a function of the cement to solid ratio used for the fill. As an average of different size of rock fragments, it is estimated that ratio of cement for the desired strength expressed in Mpa, from the following formula:

For rockfill:

% of binding = 1.46 + {1.47 x Strength (Mpa)}

For tailing fill:

% of binding = 3.5 + {4.2 x strength (Mpa)}

For sand fill:

% of binding =4.4 +{6.0 x strength (Mpa)}

For Cut and Fill:

The average binding ratio for cut and fill stope obtained in this survey is 10%. The floor thickness of 15 to 25 cm of fill per lift is cemented. The average cement fill ratio is 7% according to the survey.

The average cost of Portland observed = \$110 per MT

The average cost of Fly Ash observed = \$ 85 per MT

Average ratio of binding agent observed :

Portland = 65%

FlyAsh = 35%

So: Average binding cost = 101.25 per MT,

and cement cost (\$/MT of cemented fill )= \$101.25 x binding %

The figures 7-10 to 7-13 present the comparison between the observed and modeled values for various type of backfill operation costs. The difference observed is less than 20%.

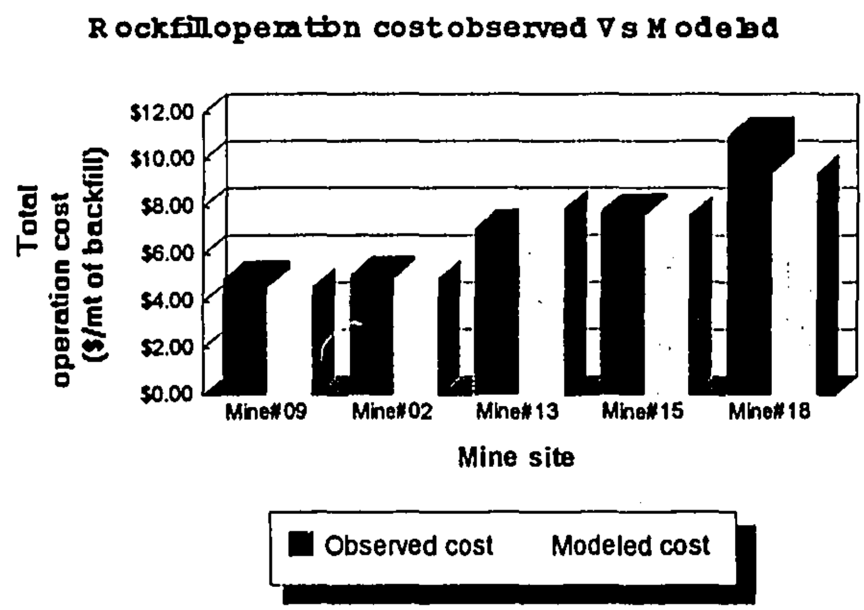



Figure 7-10 Comparison of rockfill operation cost observed and modeled

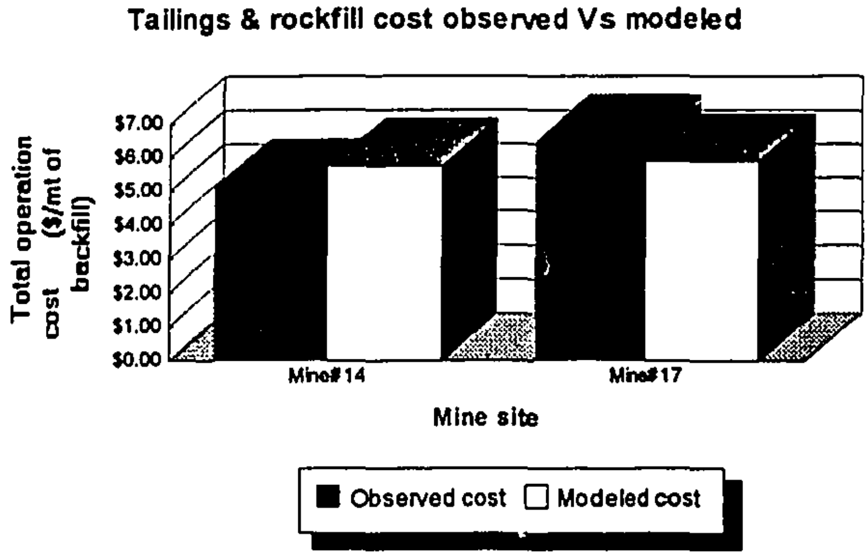



Figure 7-11 Comparison of tailings and rockfill operation cost observed and modeled

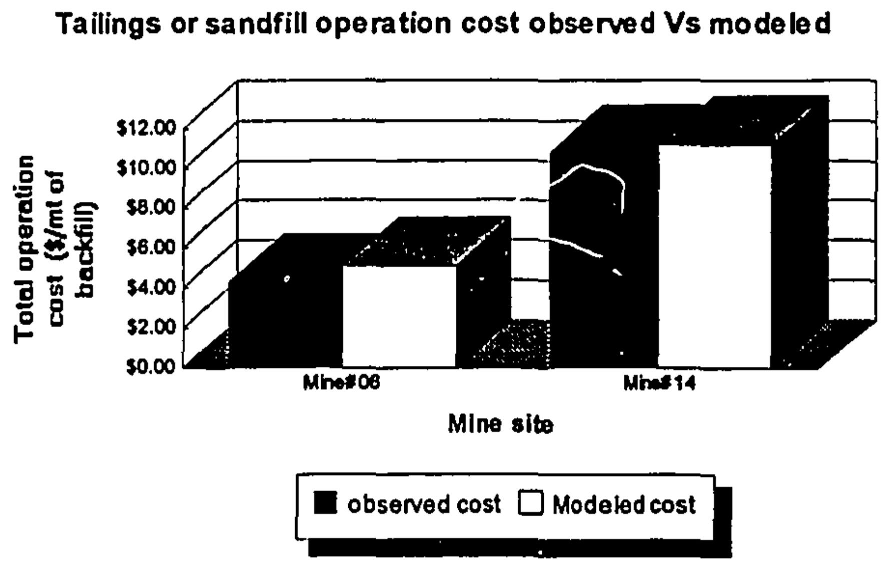



Figure 7-12 Comparison of Tailings or sandfill operation cost observed and modeled

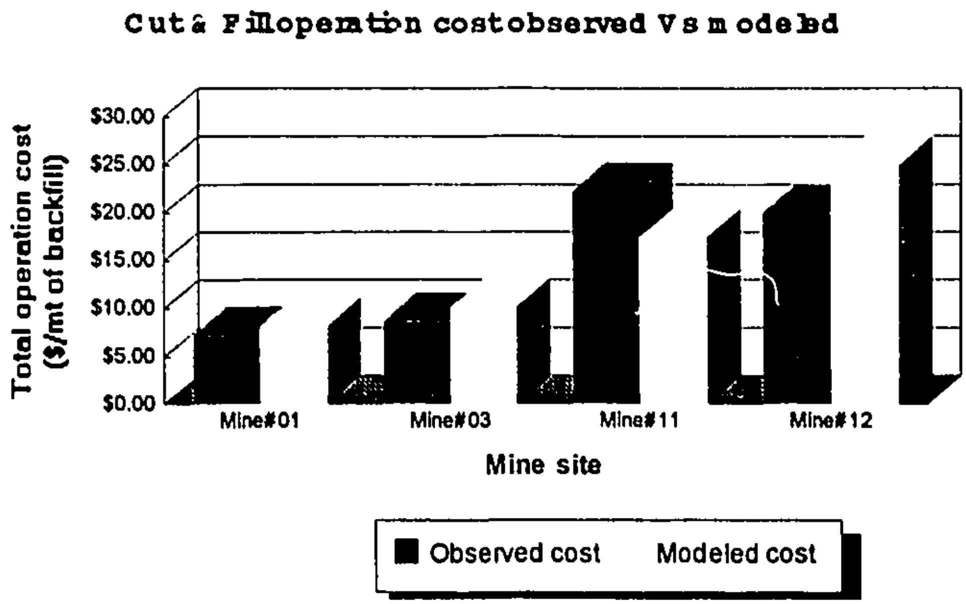



Figure 7-13 Comparison of Cut and Fill operation cost observed and modeled

### 7.4.2. Capitalization costs of modeling

The capitalization cost models presented here should be considered as a conceptual models where the user will find an estimation of total backfill capitalization cost at +\- 30%. As it is for the operation cost models, this level of precision represents the first cost evaluation stage of any engineering projects. So cost models should be used as an estimation for comparison of alternatives.

Figure 7-14 presents the comparison between the observed and modeled capitalization costs. The difference observed is less than 30%.

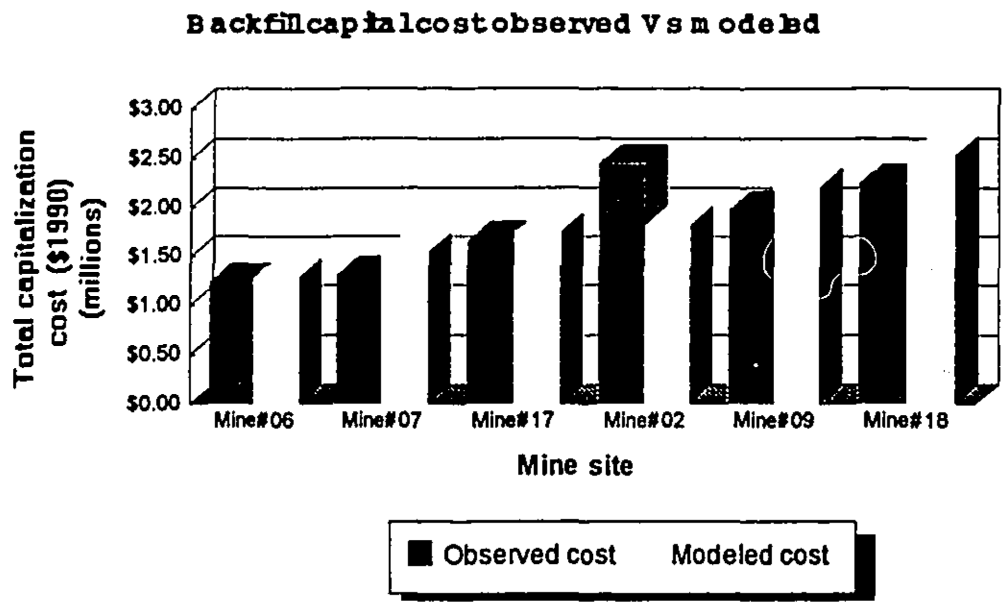



Figure 7-14 Comparison of Backfill capital cost observed and modeled

### 7.4.2.1. Surface plant

Cement slurry plant:

The average capital cost to build a new slurry plant for total backfill placement averaging 400,000 mt per year can be presented by following relationship:

Slurry plant capital cost = $ 1.3 x (total slurry tonnage per year)

Hydraulic fill plant:

The total cost of hydraulic plants follows the relationship:

Capital costs = $\{1500+(-62000/Nc\}^2$

Where Nc = the nominal capacity of the plant in term of mt of solid per hour.

High density fill plant:

The total costs of high density follows the relationship:

Capital costs = $(20480 \times Nc)^2$

Where Nc = the nominal capacity of the plant in term of mt of solid per hour.

### 7.4.2.2. Surface transportation unit

The surface transportation unit cost model is represented as:

Surface transportation unit cost = $0.45 x (average yearly tonnage ) x [D (m)/100].

Where D is the transportation distance.

#### 7.4.2.3. Boreholes

Average borehole cost = $120 per meter.

#### 7.4.2.4. Piping

Average piping costs = 20.00 per meter.

#### 7.4.2.5. Waste rock pass

Average rock passes cost = $925 per meter.

#### 7.4.2.6. Loading station

The loading station cost = 53500 per unit

#### 7.4.2.7. underground transportation units

The total cost of the underground transportation unit is obviously contingent to the distance between the loading and unloading point. The following formula is recommended to evaluate the productivity of the overall transportation unit:
Productivity (mt/hour/unit) = $\{1 / [0.015 + (2\text{x } 10^{-5} \text{ x D (m)}]\}$
The average underground unit cost = $250000 per unit

#### 7.4.2.8. Other cost

Average other cost= $51000

## 7.5. CONCLUSIONS

The survey and the analysis on the backfill operation in Quebec's underground mines demonstrated a clear view of mining practice in this region. The basic cost items have been clearly defined so that the cost analysis of backfill operation can be conducted with reasonable standards. As has been shown in the previous sections, operation and capitalization cost models have been obtained at the approximation of +/20% to +/-30, which satisfy with the basic precision requirement of feasibility studies of mining design. It is worth noting that the data of survey are limited. Further studies are needed to have deeper understanding of practical backfill operation in Quebec, and hence improve the cost models. This research thesis is not intended to address all aspects and features of backfill operation in Quebec. But rather to use the survey results as an example and knowledge base to design and test a computer based backfill supporting system which will be clear in the next chapters.

# CHAPTER 8

# ARCHITECTURE OF INTEGRATED DECISION SUPPORT SYSTEM FOR BACKFILL DESIGN

## 8.0. INTRODUCTION

The previous chapters outlined the main aspects of backfill design and various technologies involved. The context diagrams specified the major requirements, tasks and basic data flow regarding backfill design process. Our purpose of the overall specification of backfill design rationale is to present a clear scenario so that it can be represented and simulated with a computer system. This chapter defines the main models of computer system that can integrate knowledge base management system, expert system and hypermedia system together to support decision makings involved in backfill design and operation. Also in this chapter, a blackboard architecture is presented for different computer technologies to cooperate each other. In addition, the conceptual model of the system is discussed. The following chapters will present various technologies discussed in this chapter, and demonstrate how they help the mining engineers.

## 8.1. GENERAL ARCHITECTURE

The backfill design requires a large amount of information in various formats as shown in the previous chapters. From computer system point of view, the backfill design rationale defined earlier can be viewed as an operation on a computer program which consists of an information system and various tools to support decision making. Users need to access information system, and make decisions based on specific mining environment. The knowledge and previous experience of backfill operation serves as a knowledge source. This kind of knowledge source defined as backfill design database model of the system. This notion of backfill design database was first introduced in 1988[67] by M. Scoble, and the database is presented in the format of a paper-based reference manual. Based on the discussion, the backfill design database provides two major functions:

1) As a knowledge base to provide user with the backfill design knowledge, expertise and information needed to support certain decision making.
2) To store the design parameters of backfill operation for further evaluation.

The paper-based database need enormous manual work to maintain and update the information. Moreover, the process of information to meet with different purposes is the key of efficient use of information, and is unfortunately labor intensive and requires professional domain specific knowledge.

The modern information society has arisen out of the ability to store information and data in electronic database. The applications of database systems in mining engineering have been reported in many aspects such as, the geological exploration and evaluation[68],[69],[70],[71], mine planning[72],[73], equipment maintenance information system[74], slope stability assessment[75], management information system[76],[77],[78], etc. Those early application of database technology into mining engineering had demonstrated the successful power of using computer-based database technology in some areas such as geological data management and mining operation management. But the traditional database systems were developed in the mainframe environments of the early 1960s. They were typically applied to numeric and record-based data (e.g. employee records or parts inventories). More advanced technologies are still needed to meet with the information need of most engineering design problems like backfill.

One serious problem of traditional database technology is that it can not represent different data formats in a single seamless scheme, and different data representation schemes are treated in isolation. Most of conventional database systems represent data as formatted records. As is shown in the early chapters, backfill design deals with various technologies, and the knowledge is represented in different formats such as numeric number, text, graph, spreadsheet, chart figure, etc. These data formats can not be integrated into a unique environment. Users are forced to jump from different systems to manipulate different data of that format. For instance, in a typical record-based database system, users can easily store and retrieve part data as records. But in many cases, the user might want to have a pictorial view of the part. The traditional database does not provide simple way to do this. So users are forced to open another application environment that may be designed to present the picture of different parts. Further more, the processing of data in database is also limited. Data are stored and retrieved as passive objects. The deeper interrelationship of data can not be addressed.

To extend the traditional database system for backfill design, an integrated decision support system is proposed, which evolves as a result of the integration of traditional approaches to databases with more recent fields such as:

- Object-oriented programming
- Expert system
- Hypermedia
- Knowledge base management system

The merge of these technologies is illustrated in Figure 8-1:

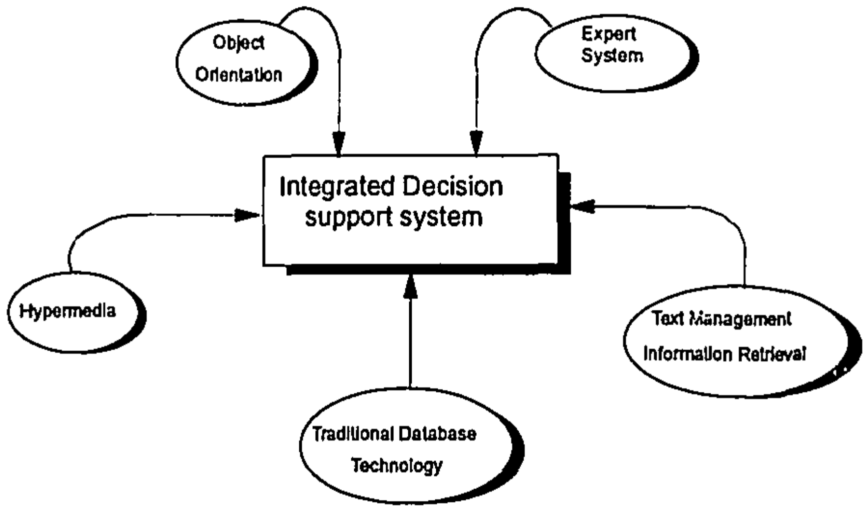



Figure 8-1 The merging technologies into integrated decision support system

The development of integrated decision support system relies on defining three levels of intelligent processes:

1. Intelligence of high-level tools,
2. Intelligence at the user-interface level.
3. Intelligence at the underlying knowledge management system.

These are illustrated in Figure 8-2:

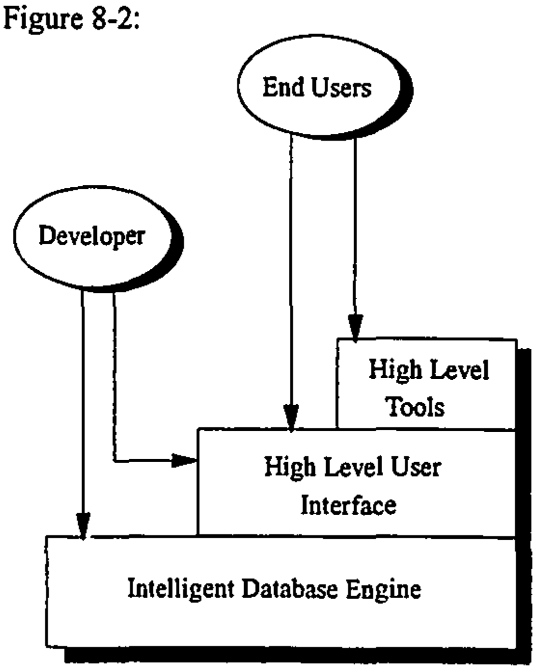



Figure 8-2 Three levels of database intelligence[79]

The first of these levels is the high-level tools. These tools provide the user with a number of facilities such as logical inference, data quality and integrity control, and automated discovery. These high-level tools represent an external library of powerful tools that some users may find useful, but not others. Most of these tools are implemented as task solvers to couple with a single task. For example, in the context diagram representation (see Figure 2-4), mining method selection is a single task encountered during backfill design. The high-level tool library should provide tools to help mining designers to select a suitable mining method to meet with mining condition. They look and work much as their stand-alone equivalents, such as consultation expert system and graphic representation tools.

The second level is the high-level user interface. It is in this level that users directly interact with the decision support system. This level creates the model of the task and database environment. As such, it has to deal as much with how the user wants to think about databases and information management as it has to do with how the database engine actually operates. Associated with this level is a set of representation tools. For instance, the interface can open a hypermedia database environment, and users are allowed to browse through backfill design reference manual in non-linear way. The user interface provides an environment to users and guide the user to perform various tasks. The model consists of the object-oriented representation of information along with a set of integrated tools for creating new object, i.e. mining operation, for example. In addition, there is a set of high-level tools, which enhances the functionality of the system. Object-oriented representation scheme presents information in a natural way. And more important, the integrity of the database can be easily enforced by attached predicate which will be clear when the concrete example is presented later.

The base level of the system is the knowledge base management system engine. The knowledge base management engine incorporates a model that allows for a deductive database representation of information. The engine includes various inference mechanism as well as query optimization process.

The development of the three level database intelligence involves different knowledge representation scheme. Different components of the same level or different levels need to cooperate each other. There is no universal representation scheme available for all kinds of knowledge representation and process. We are faced with the basic problem of designing an adapted architecture that integrates all tools and representation scheme within a simple environment and communicate each other. Several approaches have been proposed. For our purpose, we propose the blackboard approaches which allow the knowledge of multiple sources to be represented in a single representation system.

The blackboard model was developed for HEARSAY-II speech understanding system[80], and the idea behind the blackboard model may be described as follows:
We have a group of human experts, each of whom is highly qualified in a specific field. We are trying to coordinate the knowledge of these experts to solve a difficult problem. As it turns out, the experts will not directly speak to each other, but in order to help solve the problem, will agree to interact with a coordinator or scheduler and to read from and write on a blackboard. We gather the experts in a room with a large blackboard and write the initial statement of the problem on the blackboard. The experts read the problem statement and begin to think. As each expert comes up with an interesting hypothesis or an important idea, he writes it on the blackboard for everyone to see. This helps the other experts in their thinking, and provides them with important clues based on knowledge outside their own domain. Eventually, one of the experts solves the problem and writes the final solution on the blackboard. Under this architecture, the overall system is partitioned into different model. Each model maintains its own knowledge representation scheme. While the global knowledge is presented in the blackboard. Thus, a blackboard architecture is made up of three basic components.

1. A globe database (the blackboard).
2. Independent knowledge sources that have access to the blackboard
3. A scheduler to control knowledge source activity.

The knowledge sources are independent and influence each other by responding to and modifying information on the blackboard. Blackboards are distinguished more as an architecture for distributed problem solving than as a distinct method of knowledge representation. In fact, a number of different knowledge representation methods can be used for each knowledge source without disturbing the overall blackboard structure, providing that any information on the blackboard can be read and used by the knowledge sources that need. The blackboard model has at least the following three advantages:

1. It can be used to organize knowledge in a modular way.
2. It can easily integrate different knowledge representation methods.
3. It may be executed in a distributed computing environment for greater efficiency.

As a special purpose decision support system for backfill design, it is intended to solve a group of problems related to that domain, rather than provide general problem solver. Figure 8-3 shows the basic architecture of the proposed integrated decision supporting system for backfill design.

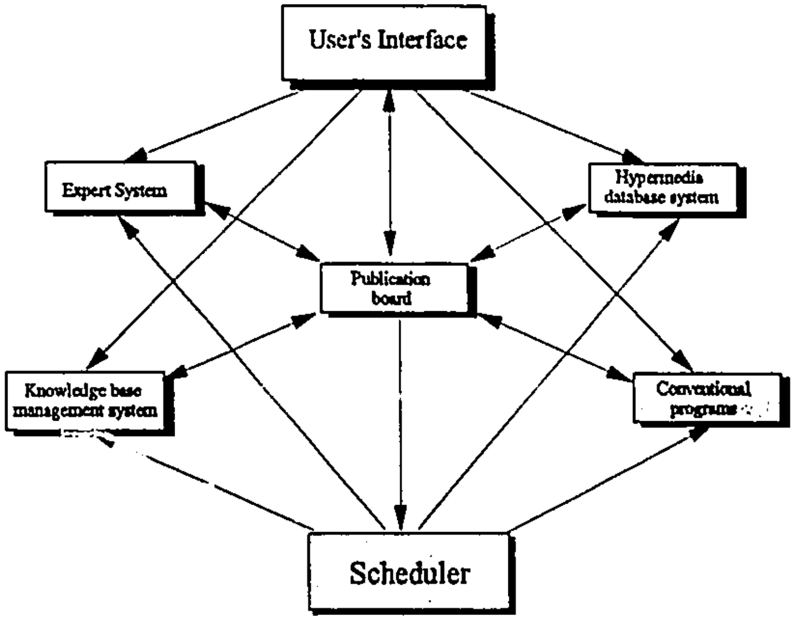



Figure 8-3 Basic architecture of integrated decision support system for backfill design

Arrows point to the direction of data flow and programming control. As indicated, the user interface accesses various integrated components concerned. Each component does not response to the user or other components directly, but rather communicates with publication board, then write and read data from the board. The scheduler reads from publication board and decides how to control the cooperation and communication between each components. In the next section, we present the basic components of integrated decision supporting system for backfill design, namely expert system, knowledge base management system and hypermedia system. The conventional programs are an open- ended library of modular implemented to solve various backfill problems that are suitable to be solved by procedural programming. For instance, the simulation of pressure ingredient of hydraulic transportation system is a well defined problem and suitable to be implemented as a modular. The program is implemented as modular and integrated in the overall system as a subroutine accessible from user interface.

## 8.2. BASIC COMPONENTS OF THE SYSTEM

Within the context of backfill design, there exist two primary areas of information. One is the reference documents such as standard specification described in the backfill design handbook. literature and other government regulation. The other one is the practical operation information and knowledge of experienced personnel. For the first one, the knowledge is represented explicitly by certain format which could be numeric data, text, graphic, heuristic rules and so on. However, the second one needs personal judgment and implicitly exists among the successful designs. Proper process of the knowledge, such as automated reasoning, or expert systems, are needed to make it useful. Accordingly, different techniques are adapted to deal with each problem. Except for the conventional numerical calculation and other procedural methods, the following components are considered essential techniques to be incorporated into the integrated decision making system.

### 8.2.1. Hypermedia database

Hypermedia is a tool for building and using associative structures. A normal document is linear, and one tends to read it from beginning to end. In contrast, reading hypermedia is open-ended and one can jump from idea to idea depending on one's interests. The nearest thing to a hypermedia document that most people are familiar with is a thesaurus. A thesaurus has no single beginning or end. Each time the thesaurus is consulted, it is entered at a different location based on the word used to initiate the search. Hypermedia can be thought of as an enriched thesaurus where, instead of links between words, links between documents and text fragments are available. In essence, the hypermedia system provides the ability to incorporate information in various formats such as numeric data, flat record, text, graphic or even sound and motion picture. In addition, the hypermedia system provides a non-linear access to the information source. These features provide us with sufficient facilities to create a backfill design hypermedia reference manual. The manual contains most of backfill technologies covered through chapter 2 to 7, from basic concepts to the state-of-the-art technology. Further more, the maintenance of the system should be easy to update. On the other hand, the access to the hypermedia manual is non-linear depending on the information need and the users' experience. The experienced backfill designer may only need a quick access to specific portion of the manual and read the information relevant. The novice designer, on the other hand, may need to get acquaint with the basic concepts first and then go to specific domain.

The backfill design rationale presented through chapter 2 to 7 outlines the basic framework of backfill design and covers most of disciplines related to the issue. In chapter 10, we will discuss the basic concepts and techniques of hypermedia system along with the implementation of backfill design manual. This brief introduction to hypermedia system focuses only on the functionality of the hypermedia system in the overall integrated decision support system.

### 8.2.2. Knowledge base management system

The backfill operation information and knowledge of experienced personnel play important role in decision making involved. The solution of most backfill design problems requires, in addition to vast numerical calculations, substantial use of practical judgment and expertise based on experience. The technologies introduced in the previous chapters or any reference manual can only provide principles in general. The mining design is usually case specific. Successfully designed mining operation is used as a source of good examples and certain decisions are made simply by drawing the comparison from case to case. Since the conditions inherited from different deposits are never identical, an apparent drawback of this method is the burden of collecting the related mining operation for comparison. Even though the general information is available, close review of data and case by case analysis manually is also an expensive process. In this integrated backfill decision support system we propose a knowledge base management system for information storage and analysis, therefore provide an automated or semi-automated tool for decision making of backfill design.

The underlying data representation scheme is based on the relational database. In addition to the capacity of database management system, the knowledge base management system provides the declarative language to serve the role played by data manipulation language and host language. It supports efficient data access and manipulation like database management system. Above that, the knowledge base management system provides the expressive power with declarative language based on logic as has been successfully demonstrated in expert systems, production system and logical programming language. Chapter 11 is devoted to illustrate architecture, functionality and other implementation techniques of knowledge base management system along with a prototyping system for hydraulic transportation system design. For now, we only introduce the knowledge base management system as a component of integrated decision support system.

### 8.2.3. Expert systems

The site specific nature of backfill design requires enormous heuristic justification and personal judgment, which falls into the context of expert system technology. An expert system is generally defined as:

A computer program that relies on knowledge and reasoning to perform a difficult task usually performed only by a human expert. A human expert reasons and arrives at conclusions based on personal knowledge. In similar fashion, an expert system reasons and arrives at conclusions based on the knowledge it possesses.

In the context of integrated decision support system, expert system may be generally viewed as advisory system. It can solve a wide range of simple isolated problems. Following this feature, we formulate the backfill design problems into individual tasks to simulate the task driven procedure of backfill design, and use expert systems as tools to solve those tasks one by one. These tools are accessible either from user interface or from within hypermedia database system as intelligent links. The backfill designers are provided with these tools at the interface level. The design process becomes the specifications of various predefined backfill operation and basic configurations as outlined in previous chapters. For example, the first task according to context diagram (Figure 2-4) is to select a proper mining method. Under certain circumstance, the mining method is inherited from the overall developing system. But the backfill designer usually has to select one method with his own knowledge. This kind of task can be formulated to a consultation expert system. The user simply consult the expert system from interface level, and make decision based on the recommendation. The concrete examples and technique details will be presented in chapter 9.

## 8.3. USER INTERFACE

The integrated decision supporting system introduces an open architecture. Different technologies, programs, tools etc. can be easily incorporated into a unique environment. The communication between programs is loosely defined. The loosely defined integration system has its inherited disadvantages and the research to seek a closely integrated system is still under development. We do not hope in the short term future, that this loosely defined integrated system can be replaced by a universal representation scheme.

In this loosely integrated system, the key to the success is to develop a multi-tasking user interface. In essence, each component mentioned early is an individual program with its own interface. It can run as a stand-alone program without concerning other components. The fundamental task of user interface is provide an interactive environment which guide backfill designer to go through all the design procedure. During this process, underlying technology of backfill operation and decision support tools should be easily accessible. Users should not be concerned with information communication between programs. Every specification of backfill operation is stored in knowledge base management system for further reference and evaluation. From engineering point of view, all the specifications have to meet with certain criteria as shown in backfill design rationale. From database point of view, those criteria impose integrity constraints on database. In other words, those criteria are the semantic knowledge of application domain, and the database has to be maintained to be in line with these semantics. The function of imposing integrity constrains is another important function of user interface. In the context of integrated backfill decision support system, the fundamental functionality of user interface is the following:

1. Present the backfill design procedure in a natural and easy to understand way;
2. Provide facilities to access various design tools;
3. Maintain the integrity constrains of database;
4. As a coordinator to schedule information exchange between different programs.

Figure 8-4 shows a typical window of the user interface. The pull down menu "Task" lists the specification tasks related to backfill design. The "Help" menu provides interface to other decision support tools.

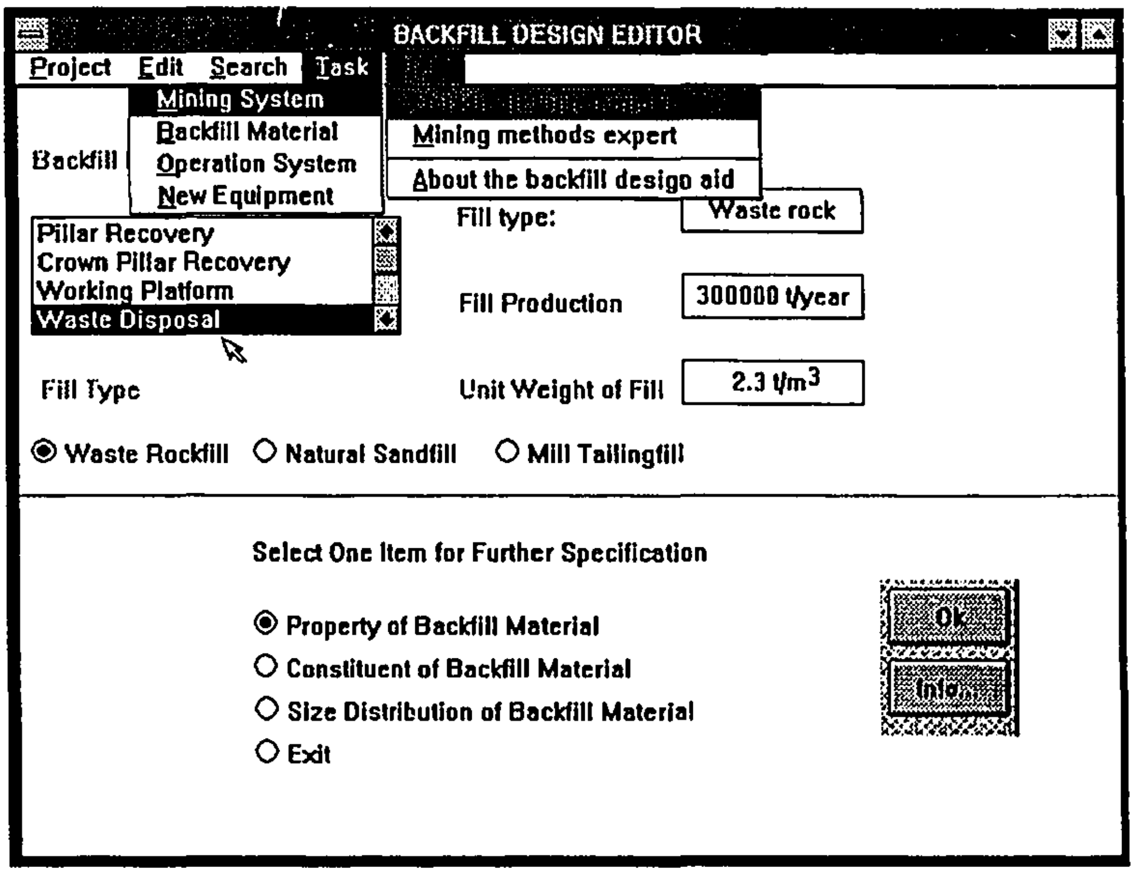



Figure 8-4 Sample screen of user interface

Whenever user comes to the point to make a crucial decision and need additional help, he can call the decision support tools for help. 'Backfill design manual', for example, provides a hypermedia backfill design manual for quick reference. The 'mining method expert' starts the mining method selection expert system session for consultation. Once the solution is reached, users can come back to the original point to continue backfill specification.

We introduce object-oriented paradigm as the user interface programming language. Object-oriented paradigm provides both data modeling scheme and programming language. The intuitive appeal of object orientation is that it provides better concepts and tools to model and represent the real world as closely as possible. The advantages of this direct representation capability in programming and data modeling are that object oriented programming allows a more direct representation of the real world model in the code. The result is that the normal radical transformation from system requirements (defined in user's term) to system specification (defined in computer terms) is greatly reduced[81].

One of the most important features of object orientation is the support of abstract data types, which define sets of similar objects with an associated collection of operators. Each abstract data type defines a class of similar objects, which incorporates the definition of the structure as well as the operations of the class. Thus, a class defines an abstract data type, and elements pertaining to the collection of objects described by a class are called instances of the class. An object is accessed and modified only through the external interface routines and operations defined for its class. Another important concept that characterizes object-oriented languages and systems is inheritance. Through inheritance we can build new classes on top of an existing less specialized hierarchy of classes, instead of redesigning everything from scratch. The new classes can inherit both the behavior (operations, methods, etc.) and the representation from existing classes. This type of data representation scheme is very similar in structure and organization to the frame-based knowledge representation[82] scheme and expert system in AI community. Within this context, the backfill operation system can be modeled as compound object which consists of several classes of objects, and the backfill design process can be viewed as a procedure of creating new interrelated objects predefined. By modeling the application this way, the user interface presents the design procedure in the way closely simulating the activities of real world backfill design.

It is worth to indicate that the concept of modeling based on object-oriented concepts is limited within the user interface level. It is integrated as a programming language rather database model. Other modular or components have their own representation scheme. For instance, The underlying data model of knowledge base management system is relational schema. The objects created in interface level can not be directly integrated into knowledge base management system but rather through blackboard approach.

Another benefit of employing object-oriented approach is to enforce the integrity constrains which are one of functions defined in the user interface level. As stated early, the creation of backfill operation objects has to make engineering sense, and comply with rules of nature. A good design takes full advantage of modern technology and pursues the best possible profit. While a poor design may lead to disaster. The objective of backfill designer is, of course, to reach the best possible backfill operation. Unfortunately, there is no single algorithm that will automatically determine this best design process. Hypermedia backfill design manual provides users with general knowledge of backfill design for personal judgment. The high level design tools defined in the interface level provide certain tools to solve some difficult problems but not all. Other semantic knowledge can be enforced by attached predicate on the attributes, which could be used to restrict access, evaluate missing information or enforce constraints on the attributes, therefore maintain integrity constrains. The object-oriented programming provides a natural way to implement this function. Next section illustrates the basic techniques of conceptual modeling based on object-oriented concepts and demonstrates the approach to implement attached predicates.

## 8.4. CONCEPTUAL MODELING

In order to satisfy the users' needs, software engineering requires the design of a suitable representation of its application's environment. This representation is called a conceptual model. A conceptual model of an application environment is thus an abstract representation of that environment that contains only those abstract properties of the environment relevant for the information requirements of its users[83]. Data models provide the conceptual basis for thinking about data-intensive applications and they provide a formal basis for tools and techniques used in developing and using information systems. Data modeling with respect to the system design can be described as follows. Given the information and processing requirements of a data intensive application, i.e. context diagram Figure 2-8 for example, construct a representation of the application that captures the static and dynamic properties needed to support the desired processes. The result of data modeling is a representation of the static and dynamic properties of application scope with integrity rules over objects and operations[84].

### 8.4.1 Techniques of conceptual modeling

The approaches to the design of a conceptual model of an application environment is based on a collection of abstraction techniques. The most fundamental techniques are classification, aggregation, generalization, and association.

1. **Classification:** classification is a form of abstraction in which a collection of objects is considered as a higher level object class. An object class is a precise characterization of all properties shared by each object in the collection. An object is an instance of an object class if it has the properties defined in the class. Classification represents an instance-of relationship in the class. For example, an object class employee that has properties employee-name, employee-number, and salary may have, as an instance. the object with property values "John Smith," 402, and $50,000.

2. **Aggregation:** aggregation is a form of abstraction in which a relationship between objects is considered as a higher level aggregate object. This is the part-of relationship. For example, an employee may be an aggregate of components employee-name, employee-number, and salary. The graphic representation of the aggregation is as in Figure 8-6.

3. **Generalization:** generalization is a form of abstraction in which a relationship between category objects is considered as higher level generic object. This is the is-a relationship. For example, the generic employee may be a generalization of categories secretary and manager (i.e., secretary is-a employee and manager is-a employee). The graphic representation of the aggregation is as in Figure 8-5.

4. **Association:** association is a form of abstraction in which a relationship between member objects is considered as a higher level set object. This is the member-of relationship. For example, the set trade-union is an association of employee members, and the set management is an association of employee members. The graphic representation of the association is as in Figure 8-5.

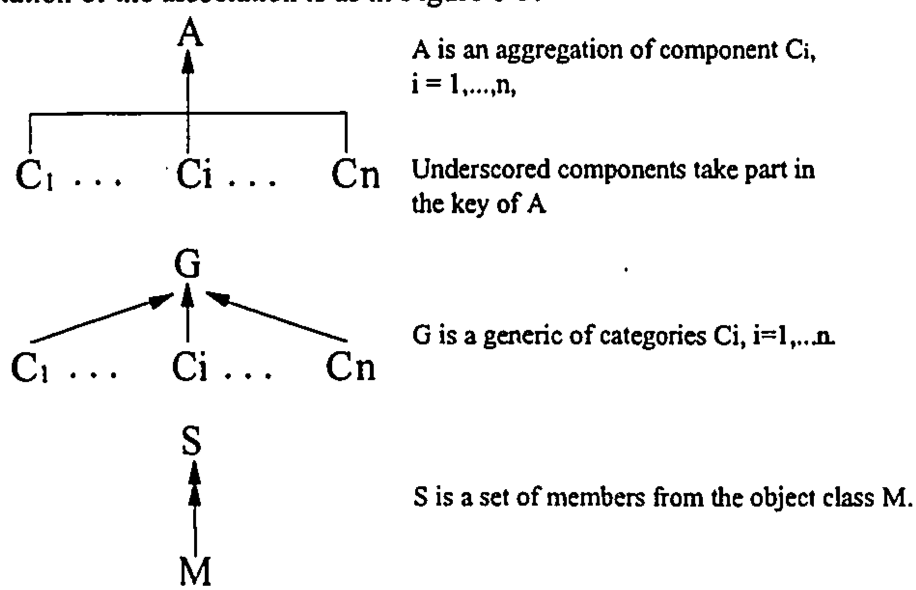



Figure 8-5 Object scheme notational conventions

## 8.4.2 The Object Identification

**1. Definition of Object**

In the object-oriented paradigm, the concept object represent both the data structures and the procedures related to the data[85]. This section deals mainly with the object identification and data abstraction, which include:

1).Object identification;

2).Object hierarchy specification;

In terms of the object programming, object is collectively defined by defining a class (i.e. Classification). A class may be modified to create another class which is either a superclass or subclass of the class. The superclass is a generalization of the class and the subclass is a specialization of the class.

A set of objects is regarded as an object type if all the objects in that set share the same set of relevant properties (attributes). The relevance of a property of an object is, of course, determined by the purpose of the model. In order to apply the classification abstraction, we have to specify precisely the properties shared by a set of objects that belong to the same type. So the identification of objects of the interested scope includes the specification of the name of objects and its attributes. Based on this definition, the following objects have been identified.

**2. Object Identification**

We start the conceptual modeling from the entity-relationship diagram. The objects are usually related with each other one way or another. These relationships represent the way these objects are linked and how they are used as a whole in the conceptual model. Relationships can be captured by the entity-relationship diagram as shown in Figure 8-6. As a convention, the entities are represented by rectangle, while the relationships are represented by diamonds. Those object types capture the main features of a backfill operation, and the relationships illustrate how objects are interrelated. Each relationship is assigned name based on the closest real meaning. For example, the 'mine' object defines the main characteristics of a mine site such as the mine location, mineral product, productivity, capital investment, etc. One mine could possibly maintain several mining projects. So we use 'maintain' relationship to specify this feature. The entity-diagram is served as high level tool to describe the application domain and give further understanding of data organization. It is not invented to capture the implementation details. We present this diagram as global picture rather as a complete specification.

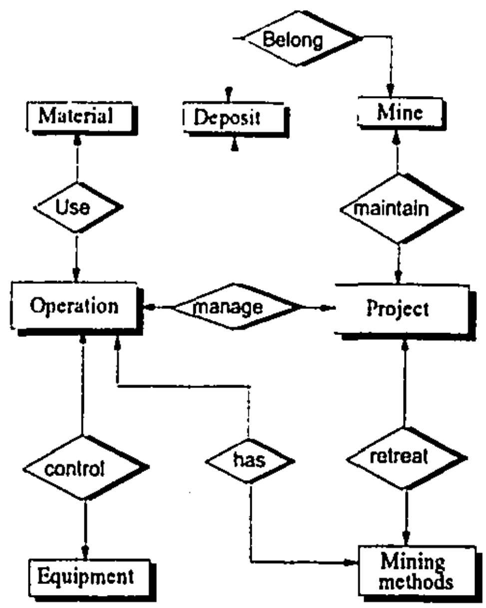



Figure 8-6 Entity-relationship diagram

The identification of objects starts from the data typing. Further specification of objects, related attributes and relationships between objects will be given later. Figure 8-7 is an example of object hierarchy of base object EQUIPMENT. Starting from the object equipment, the basic abstraction of structure tree of objects related are shown sequentially on the figures based on the notation of generalization and specialization. A more complete object identification are shown in the Figure 8-8, 8-9, 8-10 and 8-11 of the Appendix.

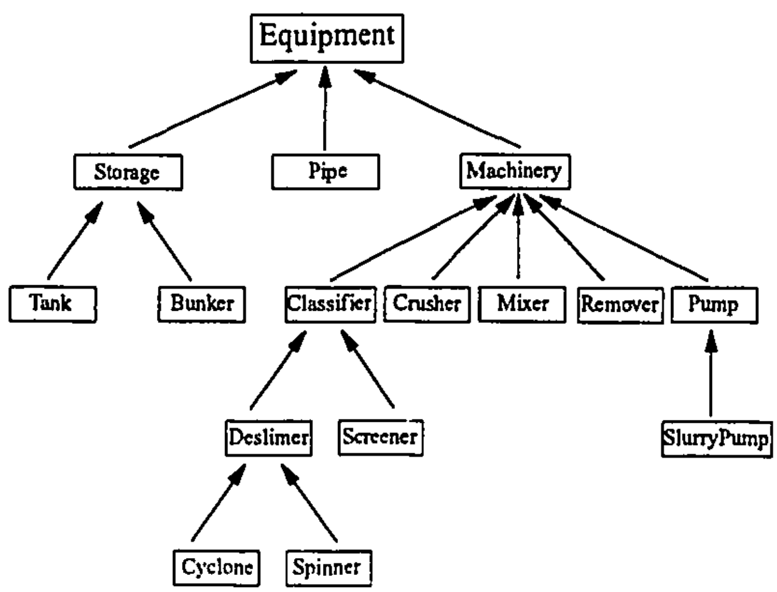



Figure 8-7 The object hierarchy of the base object EQUIPMENT

### 8.4.3. Object Specification

Based on the Figures 8-7 to 8-11 (see Appendix), the detailed specification of objects in terms of structure could be described in terms of combination of simple objects (attributes) and compound objects. Followings are the specification of these objects in the sequence of a tree structure. The objects are basically defined by the record type which is a structured type that has a fixed number of possibly different component types specified in the definition of such a type. Components of a record are called attributes. The definition of a record type specifies for each attribute its type and an identifier that denotes it. Its general form is as following:

TYPE T=RECORD $A_1$: $T_1$;

$A_2$: $T_2$;

.

.

.

$A_N$: $T_N$

END

In the above definition, T is the identifier of the record type; $T_1$, $T_2$, ..., $T_n$, are types of its components (attributes); and $A_1$, $A_2$,...,$A_n$ are their identifiers. The underline set of T is

$$T=T_1 \times T_2 \times \ldots \times T_n.$$

Given x of type T, its components are denoted as

$$x.A_1, x.A_2, \ldots, x.A_n.$$

The followings are an example of the specification of base object EQUIPMENT. The complete object specifications are listed in the Appendix.

```
Type EQUIPMENT=Record Equipment ID:       Number;
                      EquipmentType:   (Storage, Pipe, Machinery);
                      END.
```

Subtype PIPE=Record Equipment ID: Number;
Pipe ID: Number;
PipeType: String;
Model ID: Number;
Length: Number;
Roughness: Number;
UnitPrice: Number;
Type Manufacturer=Record(refer to object manufacturer);
Pipe diameter: Number;
If-added: check diameter diameter (to make sure the pipe diameter is in between 50-200mm),
If-added: Write message: The pipe diameter is normally between 50- 200mm.

END.

This notation defines the complete structure of object pipe and an if-added attached predicate to check pipe diameter. The if-added predicate will be inherited by other subclass. When a pipe object is created to certain operation, designers have to specify the diameter of the pipe to be used. When the pipe diameter is added in the form,

Pipe diameter :=(specified pipe diameter)

The if-added predicate will be activated to check whether the pipe diameter specified falls between 50-200 mm. The if-added predicate might be implemented as the following rules:

Rule 1

check diameter 'X'

If

X>=50 mm

and X<=200

Rule 2

write message: pipe diameter X has to be in between 50-200mm

if

X<50

or X>20

The pipe diameter can be accepted only if the attached if-added predicate is proved as a goal. Otherwise, the rules will display a message to indicate the acceptable range. The attached predicate can be implemented as rules or other procedural modular. In doing so, the integrity constrains are enforced in interface level.

## 8.5. USER INTERFACE DESIGN AND IMPLEMENTATION

The previous sections completed the definition of basic functionality of user interface. In the conceptual level, we describe the basic concepts of object-oriented programming and various modeling techniques. This section presents concrete issues of user interface design and implementation and shows various features by examples. The implementation is not expected to cover all aspects discussed earlier since the conceptual modeling is not to that far yet. But because it is an open-ended architecture, more advanced features and tools can be integrated to cover the whole backfill design disciplines without fundamental changes. So this preliminary implementation really gives a concrete example and lays down the framework for further development as well.

### 8.5.1. Working environment and programming software

The function of user interface requires the ability to integrate various tools for multi-tasking process. We proposed the object-oriented approach as implementation machinery. The user interface is designed to be fully integrated in MS-Windows working environment. The underlying software chosen for programming is Knowledge Garden's 'KnowledgePro'. KnowledgePro is a high level general purpose language designed to implement MS-Windows applications. It supports object-oriented programming, frame-based expert system with backward chaining inference engine and hypermedia language[86]. In the context of KnowledgePro, objects are defined with topics and the operation on the objects are event-driven. When you view a program as communication between the designer and the user, this means that the designer "talks" to the user and then waits for the user to "talk back". When the user "talks back", an event occurs. An event has both a name and a window or screen object with which it is associated. For example, the button OK in Figure 8-4 is an object associated with certain operations. When a user select the button with a mouse device or any other screen operation devices, a selection event occurs and the operation associated is performed. So the interaction between an end user and program progresses along with occurrences of different events. Users have control of choosing different object to perform requested task. In KnowledgePro, it is easy to use any of the objects provided in the Microsoft Windows interface. While creating these objects is simply a matter of calling a topic.

### 8.5.2. User interface design

The user interface is structured by grouping functions into different pull-down menu. At the top level, the user is presented with a backfill design editor. The design editor is organized in a window screen following MS-Windows style. On the top of the window is horizontal menu providing access to different functions. Figure 8-12 is the opening window of a backfill design editor:

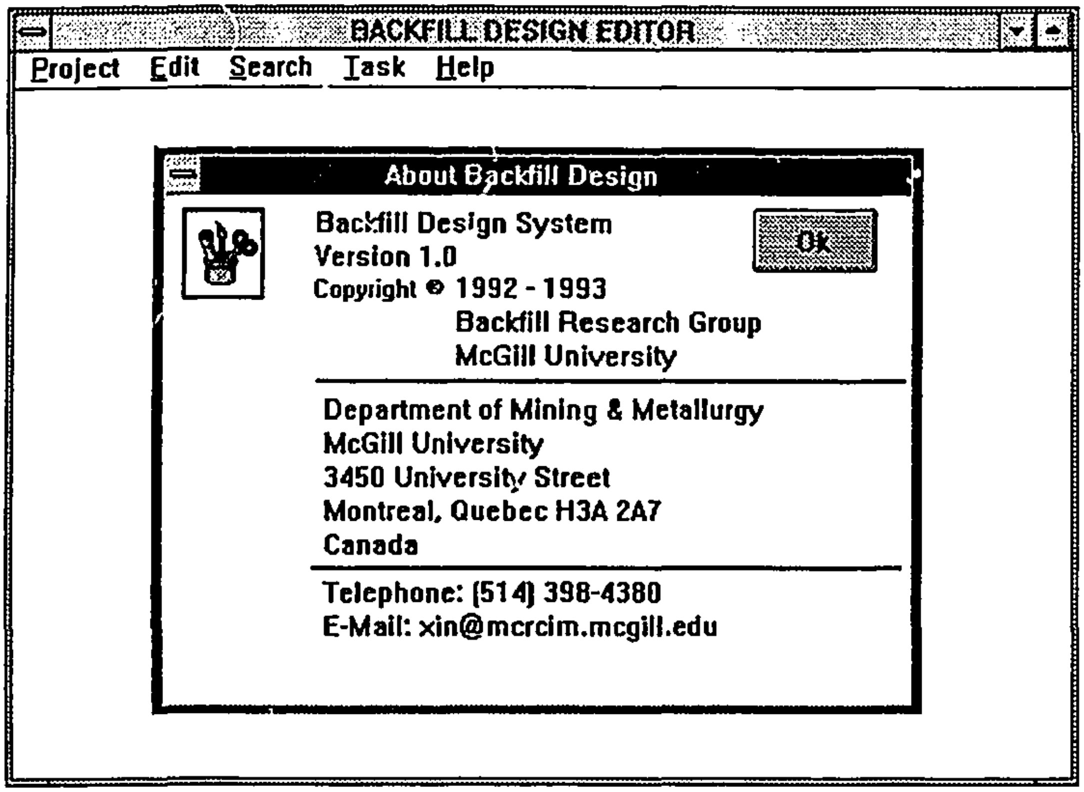



Figure 8-12 The opening window of user interface

The option 'Project' provides a facility to temporarily save design parameters for further modification and global evaluation before committed to knowledge base management system. The option 'Edit' toggles the interface between edit model and input mode. In the edit mode, the user can search object instance concerned for modification. While in input model, users are provided with object input form for specification. The option 'task' presents backfill operation as individual task. Users are supposed to specify these task as described in context diagram (see Figure 2-4). Figure 8-13 shows the major tasks listed within option 'task'. According to Figure 8-4, these tasks are:

1. Mining system: Specification of mining method and corresponding parameters,
2. Backfill material: Specification of target properties of backfill material;
3. Operation system: Specification of backfill operation system;
4. Equipment: Create objects of new equipment.

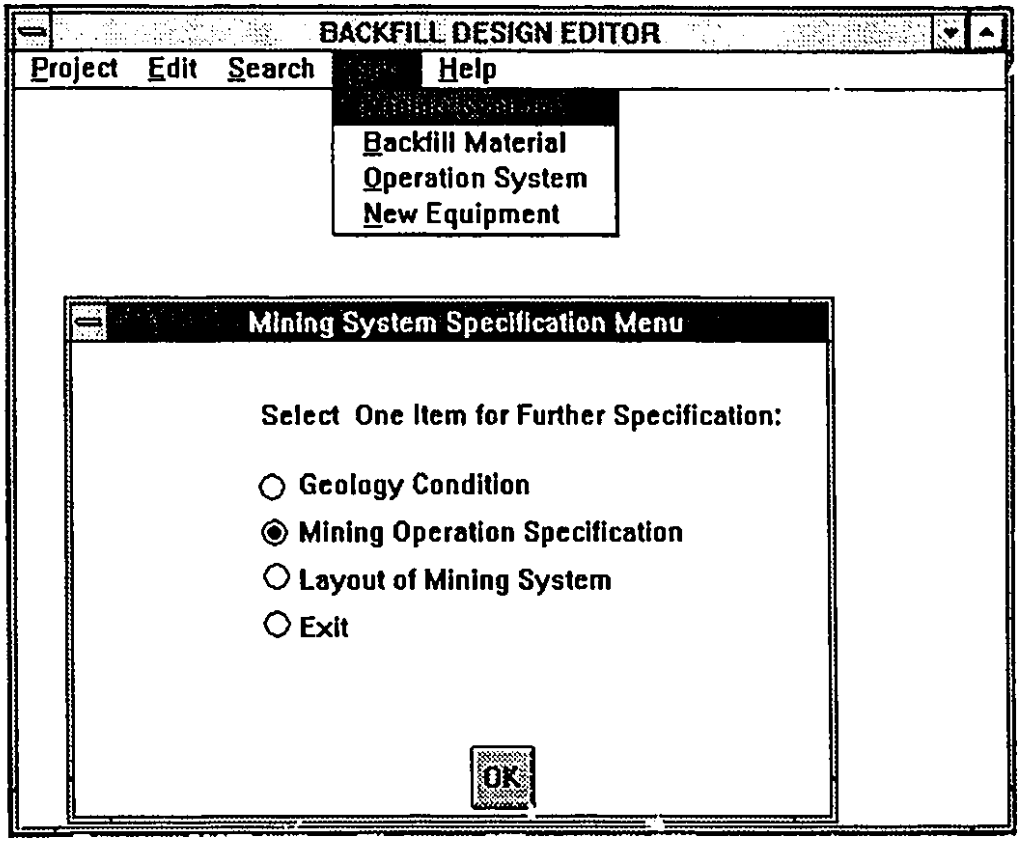



Figure 8-13 Mining system specification submenu related to 'Task' option

If user select the option 'mining system', the pop up window will display the mining specification menu for further process as shown in Figure 8-13. In the input mode, user mainly work with 'task' option. Since different mining method and backfill type requires different specification tasks, and hence different objects specification, the menu organized as such will provide a flexible interface to map different requirements.

The option 'Help' is another important facility which provides users with high level tools. As shown in Figure 8-14, a backfill design hypermedia reference manual is implemented as knowledge source, and mining method selection expert system is implemented as decision support consultation tools. If user wants information of dewatering techniques, he should simply select the hypermedia reference manual and follow the navigational information browsing procedure to retrieve diagram of filter and related information as shown in Figure 8-14.

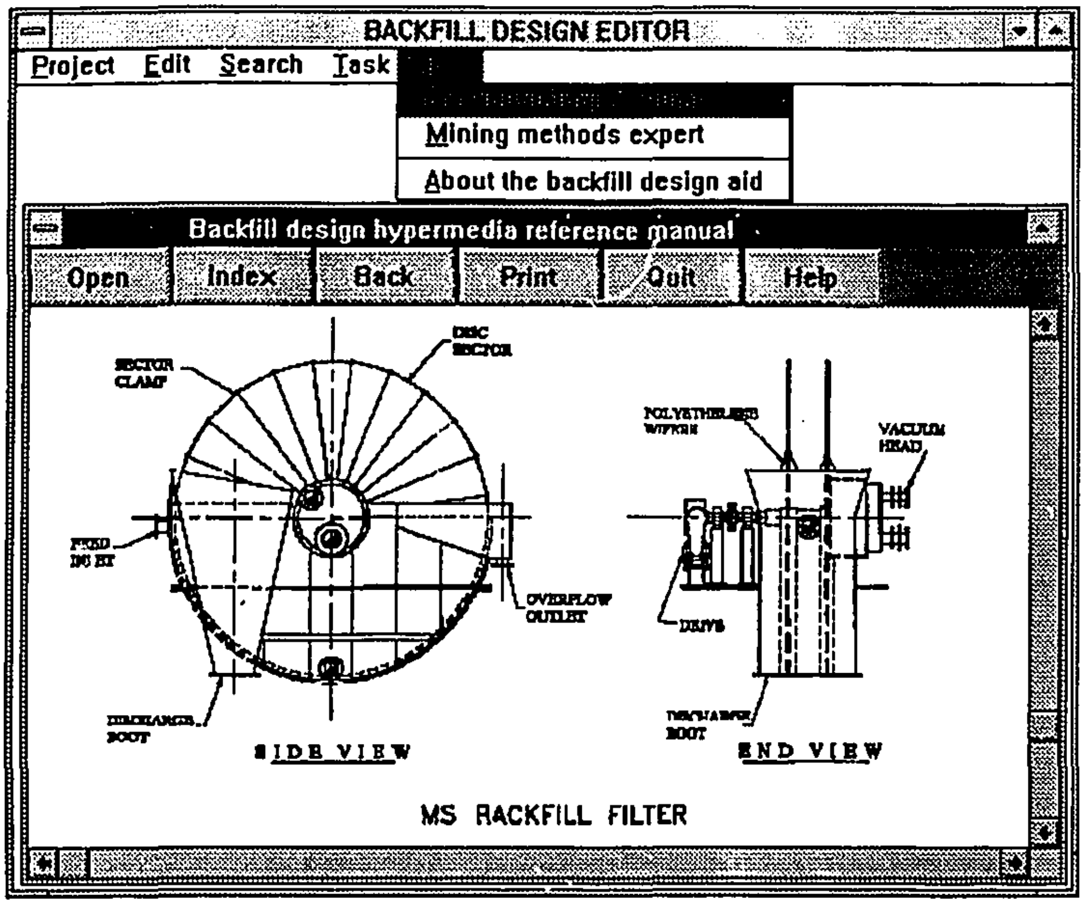



Figure 8-14 Hypermedia system connected to 'Help' menu

The mining methods expert system is another high level tools implemented accessible from user interface. The underlying development tools of mining method expert is CLIPS expert system development shell, which is developed from NASA using C language. Under MS-Windows 386 enhanced mode, multi-tasking process is fully supported no matter it is a window application or non-window application. The mining method expert system is a stand alone program with its own knowledge representation scheme. Under blackboard architecture, a public bulletin is defined for information sharing. So the solution reached through the mining method expert consultation process is published to the bulletin and then read by other programs. Figure 8-15 shows an example of the consultation window with mining method expert system:

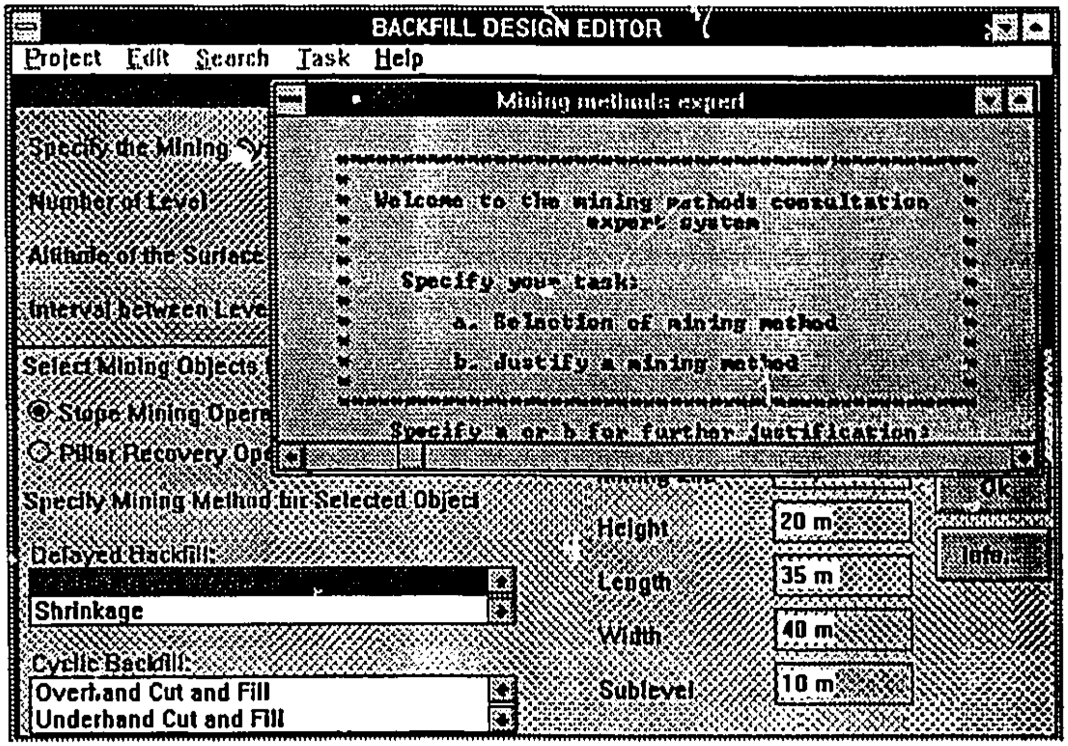



Figure 8-15 Expert system tools activated to select mining method

Suppose a backfill designer is specifying a mining system and have to select a mining method. He can make a decision either based on his own knowledge or knowledge of the mining method expert system from 'Help' menu. If a mining method expert system is activated, a special window will be pop up for consultation as shown in the figure. Once the conclusion is reached, the recommended mining method is written to public bulletin, and then read into specification window as a default method (Vertical Crater Retreat mining method in this example). The designer finally decides whether accept the system recommendation or try other tools.

The integrity constrains are implemented as attached predicate with attributes of objects. Using earlier example to specify the pipe diameter, suppose the designer is specifying the hydraulic transportation system and wants to select pipe diameter. As a general rule, the pipe diameter has to fall into the range between 50-200 mm. This is implemented as an attached predicate as shown in that example. If for any reason the designer inputs a number outside of the range, a warning message is presented for clarification. Figure 8-16 shows the basic window to specify pipe line system of hydraulic transportation. If designer specifies that the diameter of horizontal pipe line is 300 mm which is over 200 mm limit, the system will present a warning message to draw users attention. Users can either ignore the warning message or consider other alternatives.

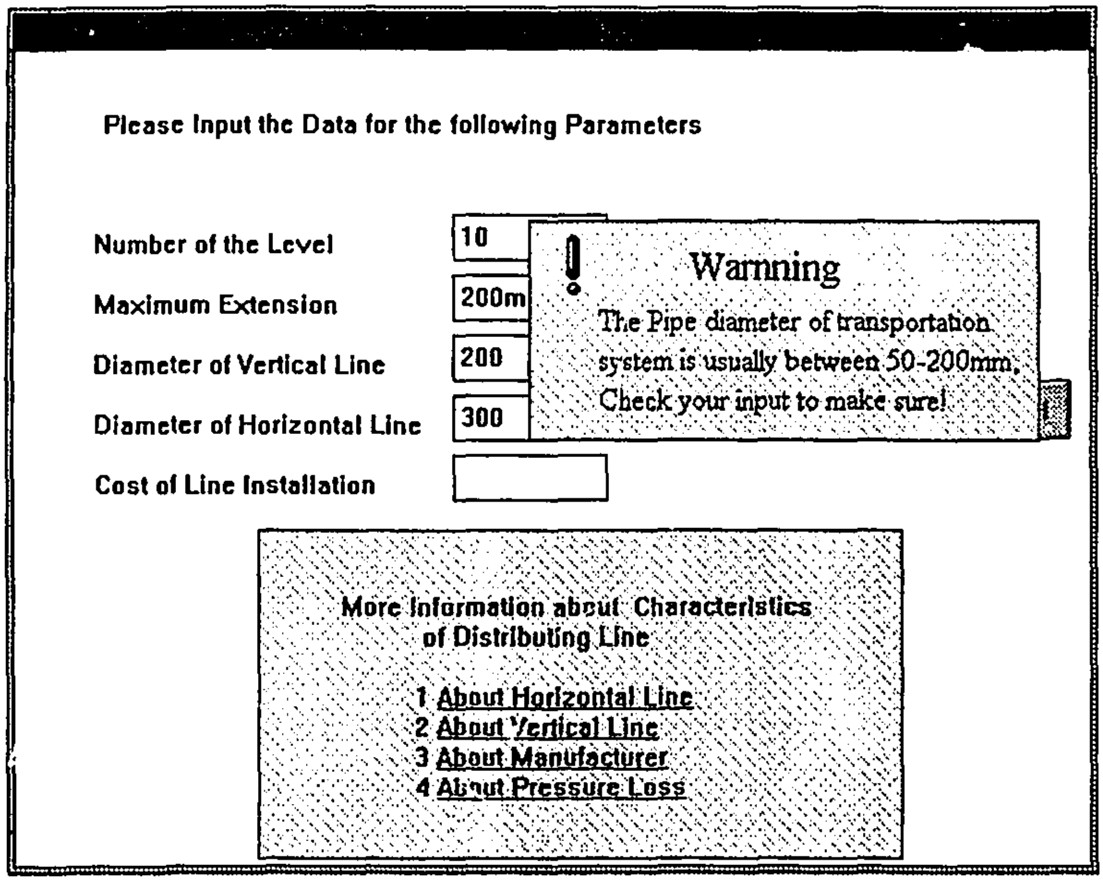



Figure 8-16 The attached predicate to maintain integrity constrain for pipe diameter

In doing so, various integrity constrains can be implemented by different approaches. Also shown in these examples are that information is organized and presented as objects of real world. So the designation procedure simply asks users to specify and select certain objects and configure into a proper complex object according to the best knowledge of designer. The hypermedia database system provides the basic concepts and techniques of various backfill technologies, while expert systems, knowledge base management system, and other conventional programming tools provide various tools to solve more complicated problems.

## 8.6. SUMMARY

Backfill design is a multi-level data/information/expertise intensive procedure as outlined in the previous chapters. To solve these problems, various techniques can be used. Based on the context diagrams Figure 2-4 and level-Zero data flow diagram Figure 2-8, which specified the major requirements, tasks and basic data flow regarding backfill design process, this chapter defines the main models of computer system that can integrate knowledge base management system, expert system and hypermedia system together to support decision makings involved. A blackboard architecture is presented for different computer technologies to cooperate each other in a single representation scheme. In addition, the conceptual model of the system is discussed. Moreover, the basic functions of user interface is discussed in details. The example given in the chapter illustrates the basic principles of object-oriented approach to enforce the integrity constrains and semantic knowledge of backfill operation when anomaly specification occurs. The following chapters will present various technologies and the implementations discussed in this chapter, and demonstrate how they help the mining engineers.

# CHAPTER 9

# MINING METHOD SELECTION EXPERT SYSTEM

## 9.0. INTRODUCTION

The selection of mining method depends largely on the overall developing system adopted and geological condition. According to context diagram Figure 2-4, the selection of mining methods is the first decision encountered by backfill designers. In this chapter, mining methods are further classified according to various considerations. Based on the analysis, a mining method expert system is proposed and implemented using CLIPS expert system shell. At lower level, knowledge representation scheme is presented as decision tree. In the implementation level, the nodes of the decision tree are implemented as facts and traverse of the tree is guided by rules. In doing so, a learning mechanism is enforced. Mining method experts can easily teach the system to understand more complicated mining techniques, which result in the expanding of decision trees. Finally, we raise the issue of truth maintenance of knowledge base of this representation scheme for further research.

## 9.1. EXPERT SYSTEM FOR MINING METHOD SELECTION

Within the framework of context diagram representation, the mining method selection is the first process according to Figure 2-4 and level-zero data flow diagram of Figure 2-8. The mining method selection was the main concern of backfill design rationale which influences other backfill operations. Based on Figure 2-8, the process 1 of mining method selection concerns of input of geological information and expects for a recommendation of mining method. This process is further decomposed to the level-1 data flow diagram as shown in Figure 9-1:

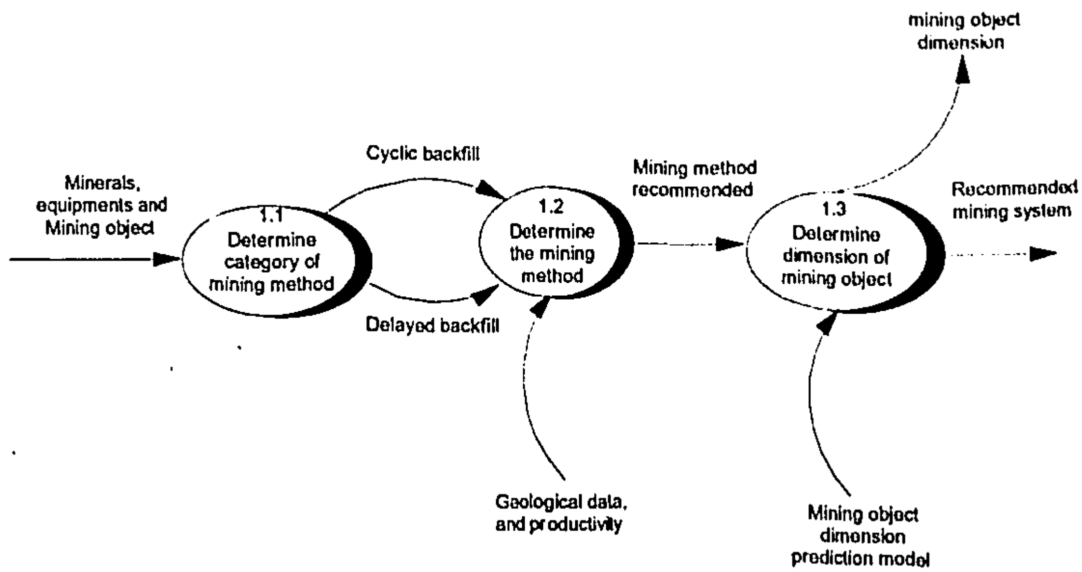



Figure 9-1 The level-1 data flow diagram for process 1

The process 1 starts from the process 1.1 which requests information of mineral, equipment and mining objects (primary mining operation, or secondary mining operation). The result of the process 1.1 is the recommended backfill type which is either delayed backfill or cyclic backfill. The process 1.2 further requests the geological information and productivity requirement to select a specific mining method for further specification. The process 1.3 asks for the mining object dimension prediction model and finally output the recommended mining system which includes the mining method and the dimension of each mining objects. The process 1.1 and 1.2 are the typical heuristic classification problem and can be suitably modeled in the context of expert system.

### 9.1.1. Applications of expert system in mineral industry

Artificial intelligence is an emerging technology in the field of computer application. Only recently has artificial intelligence advanced to the point that AI projects are accomplishing practical results. Most of these results can be attributed to the design and use of expert systems, problem - solving computer programs that can reach a level of performance comparable to that of a human expert in some specialized problem domain. It appears now that the artificial intelligence has emerged from research labs to industrial practice[87].

As an important field of AI, the expert system has the greatest potential for meeting the decision-making needs of many engineering problems. Expert systems have much scope of application in the decision making process in mining engineering, especially when a large amount of information is involved as in the case of backfill design. Instead of using computer as mere a tool for numerical computation, the expert system has been designed to make the computers participate in the decision making process[88].

Historically, expert systems have been used in a wide range of fields from military to medicine. One of the first expert systems was MYCIN, developed in 1974, in which subjective and heuristic knowledge of expert physicians was used to diagnose infectious diseases and provide antimicrobial therapy[89]. Since then, expert system applications have appeared in other areas such as mineral prospecting[90], configuration of computers[91] and structural engineering[92]. In review of literature, the application of the expert system in mining industry covers many aspects such as, the transportation analysis in mineral resources management[93], mine planning[94], [95], surface mining equipment selection[96], slope stability assessment[97], feasibility study and mining operation evaluation[98], [99], [100], consultation on coal mine dust control[101], orebody modeling[102], etc. Among many other important application domains, heuristic classification is a successful field that has been intensively studied in the history of development of expert system as is done for MYCIN system.

The mining method selection is another application domain fallen into the category of heuristic classification. Mining method selection is a crucial decision encountered by mine designers, which has increasingly drawn the interest of mining research groups[103]. Recent publications indicate a strong consensus on applying expert systems to select proper mining method. Bandopadhyay[104] presented a small prolog-based mining method selection expert system following the classification criteria presented by Nicholas[105]. In his discuss, certainty factors are introduced to deal with uncertainty and human expertise is implemented as heuristic rules. Thomas W. and Martin L.[106] demonstrated an object-oriented expert system for prefeasibility evaluation of hard rock mines which integrates the Cost Estimating System (CES)[107] as part of evaluation process. In this presentation, the problem of mining method selection and mining cost estimation are separated and evaluated using different approaches -- expert system approach and procedural approach. Therefore two fundamental problems remain unsolved: 1. conceptual model of mining method selection, and, 2. knowledge acquisition. We will discuss the basic principles of expert system and its application scopes. Based on the principles, we argue that the mining method selection, at prefeasibility study stage, belongs to the category of heuristic classification, in which the solution set is finite, and suitable for decision tree approach or backward chaining inference. Therefore, at the feasibility study stage, the solution set is infinite, hence the decision tree approach is no longer suitable. At the prefeasibility study level, we demonstrate a prototyping mining method selection expert system with a learning mechanism as a partial solution to provide knowledge acquisition facility.

### 9.1.2. Basic principles of expert system

The typical architecture of expert system is shown in Figure 9-2. In a rule-based system, the knowledge base contains the domain knowledge needed to solve problems coded in the form of rules. While rules are a popular paradigm for representing knowledge, other types of expert system use different representation scheme such as semantic network and frame.

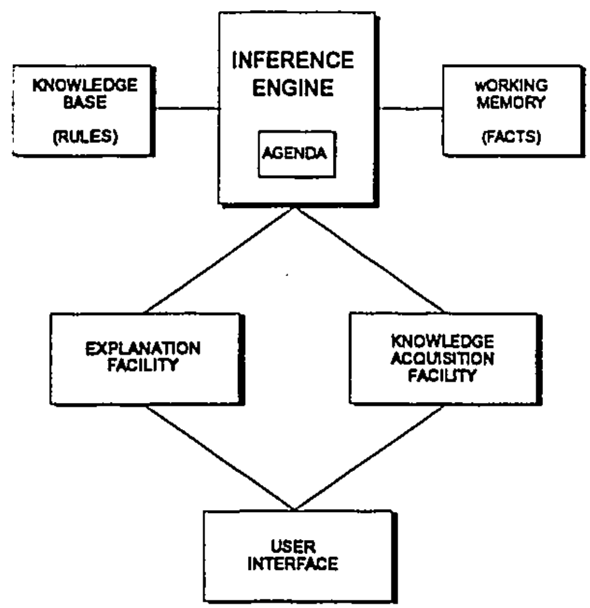



Figure 9-2 Architecture of expert system[108]

- User interface - the mechanism by which the user and the expert system communicate.
- Explanation facility - explain the reasoning of the system to a user.
- working memory - a global database of facts used by the rules.
- Inference engine - make inferences by deciding which rules are satisfied by facts, prioritizes the satisfied rules, and executes the rule with the highest priority.
- Agenda - a prioritized list of rules created by the inference engine, whose patterns are satisfied by facts in working memory.
- Knowledge acquisition facility - an automatic or semi-automatic way for the user to enter knowledge in the system rather than by having the knowledge engineer explicitly code the knowledge.

In the architecture, the key parts are the inference engine and knowledge base. Others are facilities to make the system talk with its user more friendly and efficiently. As one of the key component of the expert system, the inference engine is responsible for interpreting the contents of the knowledge base in context of user-specified input or hypothesis in order to reach a goal or a conclusion. The system can be divided into two parts. The knowledge base and the working memory constitute one part of the system. The inference engine and all of the subsystems and interfaces constitute the second part. The knowledge base of an expert system consists of "facts". The facts constitute a body of information that is widely shared, publicly available and generally agreed upon by experts in the field. The inferential processes -- the ways in which knowledge is used to solve a particular problem -- are generic and nonchanging for similar classes of problems. Once the inference engine of an expert system is designed, it will be suitable for similar reasoning problems of various area which could be totally irrelevant. The reasoning mechanisms such as backward chaining, forward chaining and reasoning with uncertainty, etc. are the example of commonly used inferential schemes.

The software available for developing expert systems can be divided into three broad categories: 1) standard programming languages such as LISP, PROLOG, PASCAL, C, and FORTRAN; 2) expert system programming environments such as OPS83, RLL, ROSIE, KEYSTONE etc.; and 3) expert system shells such as RULEMASTER, EXSYS, etc.

Standard Programming Language: These are high level programming languages with their own compiler or interpreter and a run time environment for writing and debugging programs. These languages are general and allow the developer flexibility in developing knowledge representation and inference schemes. The major disadvantage of developing a system using a language is the time and effort involved in programming a system from scratch.

Expert System Programming Environments: These are one level higher than the standard programming languages. Usually these are rule based programming languages with these preprogrammed ways of representing rules and facts. These environments offer high level utilities to build customized inference schemes and user interface. They offer the flexibility along with utilities for developing features which are standard in expert system shells. With environments, a system development team consisting of knowledge engineers and domain experts is generally sufficient.

Expert System Shells: Expert system shells are skeletal systems containing the knowledge representation and inference schemes and in many cases the user interface and means of representing uncertainty. The developer can concentrate on the knowledge acquisition phase and input the rules into the shells to have a working prototype quickly. By insulating the users from the internal representation and logic, shells permit development of systems by people who are not conversant with programming.

The following attractive features of an expert system are generally accepted:

- Increased Availability. Expertise is available on any suitable computer hardware. In a very real sense, an expert system is the mass production of expertise.
- Reduced Danger. Expert systems can be used in environments that might be hazardous for a human.
- Permanence. The expertise is permanent. Unlike human experts who may retire, quit or die, the expert system's knowledge will last indefinitely.
- Multiple expertise. The knowledge of multiple experts can be made available to work simultaneously and continuously on a problem at any time of day or night. The level of expertise combined from several experts may exceed that of a single human expert.
- Increased reliability. Expert systems increase confidence that the correct decision was made by providing a second opinion to a human expert or break a tie in case of disagreements by multiple human experts. Of course, this method probably won't work if the expert system was programmed by one of the experts. The expert system should always agree with the expert unless a mistake was made by the expert. However, this may happen if the human expert is tired or under stress.

For our purpose, expert systems are used to support certain decision makings involved in backfill design, like mining method selection, when the problems are suitable to be modeled by expert system technique. Although the original purpose of defining the integrated decision supporting system is to solve the backfill related problems, the expert system solution to the mining method selection in this chapter is not confined within the backfill related mining method selection, but rather as a general tools for various mining method selection. Therefore, the discussion followed presents the principles of mining method selection as a whole regardless of their application limits.

### 9.1.3. Knowledge engineering

The knowledge base is an essential component of expert systems, which contain the formal representation of the information provided by domain expert as encoded by the knowledge engineer. To encode the knowledge, it is necessary to make use of one or more of the knowledge representation methods for different purposes. The building of the knowledge base requires the understanding of the specific area that the expert system is going to be used. It is the domain expert who works with knowledge experts to encode the knowledge into knowledge base. In real practical application, the knowledge is hardly represented as is needed, and in some more complex system the knowledge is not even explicitly expressed. Instead, the knowledge is expressed implicitly in a large amount of information and statistic data. So certain mechanism is needed to acquire knowledge from discrete piece of data automatically or manually. If decided to use expert system shells as implementation tool, the task of building a application expert system is mainly to build knowledge base as completely as possible.

The choice of inference mechanism and knowledge representation scheme depends on the problems to be solved. Different approaches have restricted application fields. Before discussing the knowledge representation issues of mining method expert system, a profound understanding related to mining method selection is essential.

## 9.2. MINING METHOD SELECTION

The mining method selection is one of the most demanding and complex tasks that the mining engineer has to deal with. The selected method has serious impact on the conclusion reached in prefeasibility and feasibility study. This decision is usually one of judgment based on personal experience and knowledge of operations conducted in similar geologic settings. Therefore, the distinguish between mining method and mining operation should be made. The term mining method refers to a set of mining techniques, which are characterized by similar classification criteria. While mining operation stands for very detailed specified excavation operations including the adopted mining method. The difference between both terms is defined by the precision of specification. Corresponding to different mining design stage, mining method selection is related to pre-selection of a first step restriction of alternatives, while mining operation specification deals with the rather detailed determination of mining method layout, including dimensions, machinery, etc. The overall process of mining method evaluation combines both stages, which includes the following three elements: 1). the alternative mining methods, 2). the objectives, and 3). the influential factors. Of course, the lowest cost is the ultimate objective for selection, but cannot be calculated at early project stages. We now formally define those parameters and draw possible conclusions.

Lets define $M=\{m_1, m_2, m_3, ...m_n\}$ as a set of mining methods, $F=\{f_1, f_2, f_3, ... f_n\}$ as a set of factors affecting the selection of certain mining method, i.e. dip, deep, rock strength etc., and $P=\{p_1, p_2, p_3, ...p_n\}$ as the set of mining operation parameters such as dimension, machinery, capacity etc. Finally, we define G(M, F, P) as the objective function of certain mining operation, which is functional dependent on the sets M, F, and P. However, at the pre-selection stage, the G can not be calculated precisely because P set has not involved, and the objective function can only be qualitatively defined as qualified or unqualified. So the G is function of M and F. For certain mining project, the M and F sets are finite, therefore the solution for mining method selection at this stage is finite.

When the process extends to the detailed specification stage, P set starts taking effective on the G function. Some members of P set vary continuously and G function should be quantitatively defined. If not properly specified, a qualified method may be disqualified. For example, sublevel stoping method can be applied with or without delayed backfilling operation. If backfilling operation is specified, the cost (or G) will increase. Further more, backfilling operation may involve cement addition, which is especially cost effective. Sublevel stoping method can only be qualified for certain mining operation provided the cement ratio falls within certain range. As we all know, mining operation is a multi-disciplines process involving various specifications. These parameters are interrelated. Restriction on one parameter may relax the restriction on other parameters. For example, increasing of the stope dimension will improve mining condition, and hence decrease the mining operation cost. Therefore, it may also increase the strength requirement of backfill material and probably cement ratio which will increase the backfilling cost. Obviously, the search for a solution of a mining operation at this stage is not simply selection of a possible mining method, but rather a process of determining the boundary of each operation parameters and best possible combination for certain mining method. In this sense, the solution to mining method selection at detailed specification stage is infinite.

The above discussions conceptually divided the mining design process into two different stages which closely simulates the real world practice. Theoretically, there is only one goal to achieve, i.e. specify a mining system with lowest cost. However, in real mining practice, there is no algorithm to automatically evaluate all possible mining scheme and optimize the design. A prefeasibility study has to be carried out to eliminate obvious mining methods and concentrate on two or three qualified scheme for further evaluation. This approach may not guarantee to reach the best mining method. But it will provide a qualified one. Based on personal judgment, the more experience the mining designer has, the better the mining design will be. The previous attempts to solve the mining method selection problem by using expert system made no clear distinguish between the two design stages which is a crucial concern to choose reasoning strategy when designing expert systems. In general, the decision tree approach or backward chaining has more advantages than forward chaining if the search space is finite. But in the case that the solution set is infinite, the forward chaining approach will be more efficient. So the understanding of different design stage and corresponding characteristics does provide solid foundation for further design of expert system.

Under all circumstances, effective evaluation of the mining methods depends upon the information available. Rarely is it possible to do more than a preliminary study from core drilling observation and other surface investigations. Information from actual underground working can suffice for a final development plan, so a combination of surface investigation and detailed studies of underground conditions is necessary to avoid mistakes in the early stages of mine development.

**1. Geological conditions.**

Most of the factors that physically influence the choice of a mining method are included in the concept of geology; i.e., the ore situation in the rock and the behavior that can be expected of the ore and the surrounding rock.

**DIP:** The dip of the ore body is a factor influencing the mining method. Normally, the dip is classified as either steep or flat, with a rather undefined medium range between the two extremes. Steep dips range from the angle controlling gravity flow, about $50^{o}$ to vertical. Flat dips are more difficult to define, since they are connected with equipment capabilities, they normally range between horizontal and an inclination of $20^{o}$. The medium dips are difficult to fit into the description of mining methods but range from $20^{o}$ -$50^{o}$. Table 9-1 defines the relationship between the dip of the ore body and the mining methods that may be applicable.

Table 9-1 Relationship between dip of ore body and mining methods that applicable.

| | | |
|---|---|---|
| Flat | Room-and-pillar mining | Competent horizontal ore body |
| Flat | Longwall mining | Thin seam-type ore body |
| Medium | Room-and-pillar | Competent ore body |
| Medium | inclined room-and-pillar mining | Slope precludes mechanization |
| Medium | Stope room-and pillar mining | Stepping allows mechanization |
| Medium | Longwall mining | Thin seam-type ore body |
| Medium | Cut-and-fill mining | Firm ore body; selectivity; mechanization |
| Medium | Square-set mining | High -grade ore; labor-intensive |
| Steep | Sublevel stoping | Competent ore regular boundary |
| Steep | Shrinkage stoping | competent ore, regular boundaries, delayed ore recovery |
| Steep | Cut-and-fill mining | Firm ore body, selectivity; mechanization |
| Steep | Sublevel caving | Large ore body, extensive development effort |
| Steep | Block caving | Massive ore body; extensive development effort |
| Steep | Longwall mining | Thin seam-type ore body |
| Steep | Square-set mining | High-grade ore; labor intensive |

**2. Ore reserves and grades**

The relationships between production capacity, ore grade, and available reserves are factors that must be included in the selection of a mining method. High grade ore should consider the high recovery methods and low grade large scale ore permits massive low recovery mining operation. A method known to require more labor than another may allow selective mining, thus producing ore of a higher grade and yielding a more valuable product.

### 3. Productivity and Mechanization

Productivity in mining has become synonymous with mechanization, replacing manual labor with powerful machines. Over the last few decades, a tremendous development has taken place, rationalizing underground mining methods with the introduction of new machinery of increasing size and capacity. In principle, the capacity of a machine is related to its size, so it is advantageous to select the largest units possible. However, there are limitations to the choice. Underground openings are not of unrestricted size, and operating within the available space limits the physical dimensions of a machine. Another factor is the capacity that can be utilized effectively.

According to the principles above, the mining methods can be classified to the following three category:

1. Stope requiring minimum support
2. Stopes requiring some additional support other than pillar.
3. Caving methods

The characteristics of each individual mining methods are the following.

### 1. Stope requiring minimum support

An open stope is an underground cavity from which the initial ore has been mined. Caving of the opening is prevented (at least temporarily) by supporting from the unmined ore or waste left in the stope in the form of pillars and the stope walls (also called ribs or abutments). In addition to this, primary support may also be required using rockbolts, reinforcing rods, split pipesets, or shotcrete to stabilize the rock surface immediately adjacent to the opening. The secondary reinforcement procedure does not preclude the method classified as open stoping.

Condition to apply the open stope:

- Waste rock is competent enough to use an open-stoping method;
- Assuming that the reserve is not classified as gassy, the form which the method will take is primarily determined by the dip and thickness of the reserve.

Depending on whether dry, broken material flows by gravity, or whether it must be moved by non gravity methods where energy must be supplied to move the material, the open-stope mining system can be further classified as the following methods.

### a). Room-and-pillar

Rock conditions: Room-and-pillar mining is an open-stoping method where mining progresses in a nearly horizontal or low angle direction by opening multiple stopes or rooms, leaving solid material to act as pillars to support the vertical load. Since the direction of excavation (angle of dip) is below that which would cause the dry material to flow by gravity to a drawpoint or gathering point, the material must be loaded on the room where it was extracted and transported to a point where it will flow, either by gravity or mechanical means, to a central gathering point to be taken out of the mine. This is an important aspect of room-and-pillar mining which differentiates the system from other open-stope mining methods which rely heavily upon gravity to transport ore from where it was broken to a lower elevation, usually through a drawpoint. There are many variations of the method which go by a number of names in local districts: breast stoping, breast-and bench stoping, board-and-pillar, stall-and-pillar, and panel-and-pillar are all basically open-stope room-and-pillar mining. We distinguish two methods for further specification: single sliced room-and-pillar and multi-sliced room-and-pillar.

**b). Sublevel stoping**

The sublevel stoping mining method is usually applied to a relatively steeply dipping, competent ore body, surrounded by competent wall rock. Ore is produced by drilling and blasting longholes, which can range from 50 mm (2 in) to 200 mm (71/2 in) diam., with lengths up to 90 m (300ft). Longholes can be inclined in any direction, but the ring or pattern usually from a plane, and the holes are blasted as a unit. Recently developed mobile drilling and loading machinery, as well as new explosives products, blasting techniques, cemented sand and rock fill have make sublevel stoping a highly efficient and versatile mining method. When designing a sublevel stoping production system, it should be kept in mind that accurate, realistic scheduling is essential to smooth production rates. Planning of pillar recovery, representing the majority of ore tonnage in a production block, must be done during early mine planning. Sublevel stoping can be further classified as: 1). sublevel stoping. 2). blasthole stoping, and 3). VCR. (large massive ore deposit)

**2. Stopes requiring some additional support other than pillar.**

The backfill techniques are usually used to deal with stopes requiring some additional support other than pillar, which is applied to the condition where the rock is not strong enough to stand alone for long time after the excavation. There are virtually two kinds of backfill methods, 1) the delayed backfill, 2) cyclic backfill, each of which has their specific applications.

**1). delayed backfill**

The application conditions of the delayed backfill is that rock should be strong enough to stand alone for the period of mining operation. The void formed by the excavation will be filled by rock, sand, or tailing with or without cement. The following is a list of the commonly used mining methods.

**a). Shrinkage stoping**

The idea dip is 70-90$^{o}$, as dip fall below 70$^{o}$, the shrinkage draw begins to strongly favor the hanging wall side, thus leaving a poor working platform for conventional overhand work. This is particularly true in relatively wide stopes. The support afforded to the hanging wall also diminishes with decreasing dip, reaching nil as the dip approaches the repose angle of broken ore. Dips below 45 -50 are not generally shrinkable except by open stope "semishrinkage" methods. The minimum mining width is fixed by working space requirements in the stope-generally about 1 m. Shrinkage in narrower veins requires that waste rock from one or both walls be broken with the ore and the attendant dilution accepted to achieve the minimum width. Narrow stopes are less suitable, encouraging hang-ups and gridging of broken ore, with the attendant problems of erratic draw and incomplete recovery of broken ore. Maximum practical width may be 3 m or less to over 30 m, depending upon the competency of the ore and its ability to stand unsupported across the stope back. Regularity along the dip is a prerequisition of shrinkage as there must be no serious obstruction to the flow of broken ore downward through the stope to the sill level. Gentle rolls along the dip are acceptable if the local footwall dip everywhere exceeds 45-50$^{o}$. Off-dip hanging wall and/or footwall splits can generally be mined selectively from a conventional shrink stope as they are encountered without adversely affecting subsequent continuation of shrinkage mining up dip on the main vein. Vertical offsets or major rolls along the dip which cannot be "smoothed over" generally require that a sublevel be established with new draw control development.

**b). Cut-and-Fill stoping**

Open cut-and-fill stoping for many years was probably the most widely used mining method in underground metal mines. Then for a time this method was largely supplanted by the blasthole stope. It again became popular as many mines reached depths or grades where methods requiring large open voids to remain open for extended periods of time became unsuccessful, often as a result of excessive dilution.

The open cut-a-fill method is very flexible and is readily adaptable to almost any ore body. The standard application requires that a slice of ore usually 2.4 to 3 m stope, and as the ore is taken down, the back is dressed and rockbolted. After the back is secured, the broken rock is removed through rock passed to the level bellow. When the rock has been removed the rock passes are extended upward the height of the ore removed, the stope is backfilled, and another cycle is mined. This methods is best employed in plunging ore bodies with considerable vertical extent, ore area that require selective mining, ore area where weak wall conditions exist, and ore bodies that have an ore value that will carry this relatively expensive mining method. Blasthole stoping, shrinkage stoping, and other mining methods that do not employ rock passes in a stope are not efficient in plunging or flatly dipping ore bodies because the footwall makes ore removal quite difficult. Since mining is accomplished by taking down slices of the back, only small areas of the wall rock are exposed at any one time, and these only for short periods.

**c). VCR.**

The introduction of 165-mm holes to underground mining operations has made possible the application of Canadian Industries Ltd's (CIL) vertical crater retreat (VCR) mining method. This unique and revolutionary new application of spherical charge technology, when applied to primary stopes and pillar recovery, eliminates raise boring, slot cutting, and dilution of ore by backfill; greatly improves fragmentation; reduces labor and time requirements; eliminates upholes drilling and blasting; and minimizes or completely eliminates damages by blasting to the walls and retreating backs of the stope or pillar.

**d). Sublevel stoping.**

The sublevel stoping miming method is originated from open stope mining. It requires backfill after mining activity.

**e). Blasthole stopping.**

Also comes from open stope with delayed backfill operation.

**2). Cyclic backfill**

The application condition of the cyclic backfill is that both host and ore rock are very weak to stand alone for even short period, so the backfill operation has to be conducted along with other operation to provide sufficient safety support. Following is a list of mining methods used in cyclic backfill.

**a). Undercut-and-fill mining**

Undercut-and-fill mining is usually used for pillar recovery in very poor rock. The undercut-and-fill method mining was developed by Inco Ltd. in the Sudbury district of Ontario, countered in pillar recovery. Although it is now sometimes utilized as primary stoping method, the main application continues to be in pillar recovery. Undercut-and-fill is a method of extracting a block of ore by mining successive cuts, working from the top down. After a cut of ore is completely mined out, laminated timber stringers are constructed along the sides for the full length of the cut. Round logs covered by a wire screen are laid across the stringers to form a mat. The cut is then tightly filled with hydraulically placed immediately below the mat. Drilling, blasting, muck removal, and timbering operations are repeated until all the ore in the cut has been mined from level to level. The principles of undercut-and-fill can be adapted to a variety of mining situations. It can be applied transversely or longitudinally in ore bodies of varying width, that dip from vertical to flat lying. The method can be applied using conventional equipment.

Undercut-and-fill is a selective method, ore recovery is high, and dilution can be controlled to an acceptable level. However, it is a high-cost, labor-intensive method and as such its use is restricted to areas where the less costly methods of ground support are not successful, and bulk mining methods are not adaptable. One basic requirement for undercut-and-fill is an adequate source of hydraulically placed cemented sand fill and a dependable sand plant and distribution system.

**b). Overcut-and-fill mining**

Overcut-and-fill mining is used mostly for pillar recovery and poor rock but the ore condition is better than the undercut-and-fill mining methods since it requires a temporary space for mucking drilling operation. It also differ from the undercut-and-fill that the cost is cheaper compare to the undercut-and-fill.

**c). Cut and Fill with Post pillar**

Cut and fill with post pillar is used to the situation where the rock is poor but strong enough to stand alone with minor support by post pillar which is permanent lost. It is mostly applied to the dip steep ore deposit.

**d). AVOCA**

AVOCA method worked very much like the undercut-and-fill method. But the blasting is designed to form steep face along with the ore body so that the ore can be drawn out through the side raise. The ore deposit should be dip and strong enough for short period. The productivity is limited.

**3. Caving methods**

The original application of caving methods was in ground so weak that it would collapse even in small headings when the support was removed. Caving methods are usually massive, highly mechanized operation. No artificial support is needed. Following is a list of caving method

**a). sublevel caving**

Depending on the stability of rock, the sublevel caving can be further classified as 'Induced caving' and 'Blasted caving'. The major problems with sublevel caving are the control and minimization of dilution. In case of induced caving, the waste should be weak enough to cave. While for blasted caving minimum waste blasting is acceptable. In the case of flat dip the potential disadvantages is inefficient development and drilling of hanging wall drifts because of low ore height above the drift; and on the top sublevel only about 50% of the ore is drawn because the ore takes up its angle of repose and cannot be reached by the loaders. In the case of vertical dip there is always another level underneath to reclaim the ore left behind (except for the bottom level of the mine), so recovery is relative good. But for rather flat dip, only a small quantity of the lost ore is drawn on levels below, and portions of the ore body are not mined at all. This results in high development cost per ton and low recovery. On the medium dip, geometry is still quite favorable and although a little ore will be lost in the footwall, recoveries are relatively good. From this explanation, then, it is desirable to have fairly compact ore, weak walls, and a steep dip. Advantages: It can be applied to both hard and moderately weak ground; it is flexible so it can be applied to irregular ore bodies and wide or narrow ore bodies down to about 3.7 m; all operations take place in drift-size headings that can be well-supported and provide good conditions for accident prevention; it is suitable for a high degree of mechanization; activities can be specialized, simplifying training of personnel and reducing the number of miners required. The method has been successfully applied to pillar recovery. Disadvantages: The major disadvantages are high dilution and the problems of controlling it. Control includes brow support, good drilling and blasting practice, and an organization for strict draw control. High development cost is also a factor.

**b). Block caving**

Block caving is a distinct caving method applied mostly to large, massive, ore bodies because of its inherent low cost and high production capacities. Areas of sufficient size are removed by undercutting so that the mass above will cave naturally. Drawing of the caved ore at the bottom of the ore column causes the caving action to continue upward until all of the ore handling. When properly applied, block caving results in a lower mining cost per ton than any other under ground methods. There are three distinct forms: 1) dividing the horizontal area into rectangular or preferably square of nearly square blocks, drawing evenly over the entire area to maintain an approximately horizontal plane of contact between broken ore and caved capping; 2) dividing the horizontal area into panel across the ore body, retreating by undercutting manageable areas from one end of the panel to the other and maintaining an in inclined plane of contact between the broken ore and caved capping (thus the name panel caving); and 3). no division of horizontal area of the ore body into definite blocks ore panels (this is termed mass caving). Undercutting may be from wall to wall, retreating room on end of the ore body to the other, maintaining an inclined plane of contact between the broken ore and caved capping. The total active caved area is determined by the size of block that will not produce undue stress on workings below the undercutting level and by total production requirements. This type of operation is also referred to as panel caving at some properties.

Block caving can be used where rock mass has sufficient fractures, planes of weakness, as in some types of hematite, so that the mass will break if the support of an area of sufficient size is removed by some method of undercutting. The material caves from the bottom of the block, broken material is drawn off, and the caving of the mass progressed upward through the ore. There is a limit to the rate that this caving action progresses which is in relation to the structure of the material being caved. Block caving in its various forms is applicable to deposits of various shapes and ores of various strengths. Its success is governed by rigid requirements and limitations. In unsuitable deposits ore where improperly employed, the loss of ore may exceed that of any other mining methods. Good planning, systematic work procedures, careful supervision, and good judgment contribute to its success [9].

The above discussion covers most of the issues related to mining method selection from mining engineering point of view. The next step is to find an approach best suitable to address these concerns based on the knowledge representation and inference mechanism from expert system point of view.

## 9.3. DECISION TREE APPROACH

Decision trees provide a useful paradigm for solving certain types of classification problems. The problems suitable for decision trees are typified by two primary characteristics. First, they provide the answer to a problem from a predetermined set of possible answers. Taxonomy and diagnosis problems generally meet this requirement. For example, a taxonomy problem might require the identification of a gem from the set of all known gems. A diagnosis problem might require the identification of a possible remedy from a set of remedies or the selection of the cause of a failure from a set of possible causes. Because the answer set must be predetermined, decision trees do not work well for problems which must generate solutions in addition to selecting them. The second characteristic of decision tree is the manner in which they derive a solution by reducing the set of possible solutions with a series of decisions or questions that prune the search space of the decision tree. Within the context of the mining method selection, the problems meet exactly with these requirements. Therefore we propose the decision tree approach to build the underlying expert system.

### 9.3.1 Characteristics of decision trees

A decision tree is composed of nodes and branches. Nodes represent locations in the tree. These locations can either be decision nodes or answer nodes. Branches represent connections between nodes. Branches connect parent nodes to child nodes when moving from bottom to top. The node at the top of the tree which has no parent is called the root node. Note that in a tree every node has only one parent, with the exception of the root node, which has none. Nodes with no children are called leaves. The leaf nodes of a decision tree represents each of the possible solutions that can be derived from the tree. These nodes will be refereed to as answer nodes. All other nodes in the tree will be referred to as decision nodes. Each decision node represents a question or decision which when answered or decided determines the appropriate branch of the decision tree to follow. In simple binary decision tree, the question could be a yes/no question. The left branch of the node would represent the path to follow if the question is answered yes, and the right branch of the node would represent the path to follow if no is answered. In general, a process yields a single branch to follow. Thus, decision nodes may select a branch corresponding to a set or range of values, a series of cases, or functions mapping from the state at the decision node to the branches of the decision node. Figure 9-3 is an example of a decision tree.

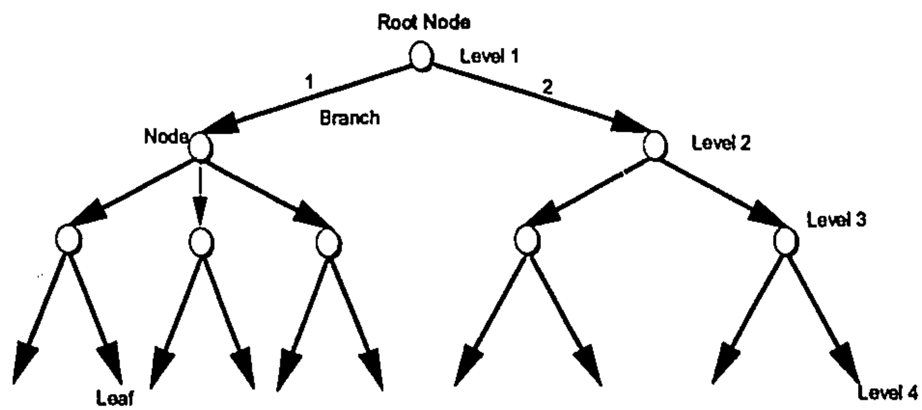



Figure 9-3 Example of decision tree

In general, each node can have arbitrary branches. But any tree can be simplified as a binary tree shown as Figure 9-4. The simplified tree represents reasoning process equivalent to the previous tree but higher. The height of a tree is referred to as the depth of the tree. Binary trees are simple and easy to understand and implement. In our prototype expert system, we chose to use binary tree to model the mining method selection process.

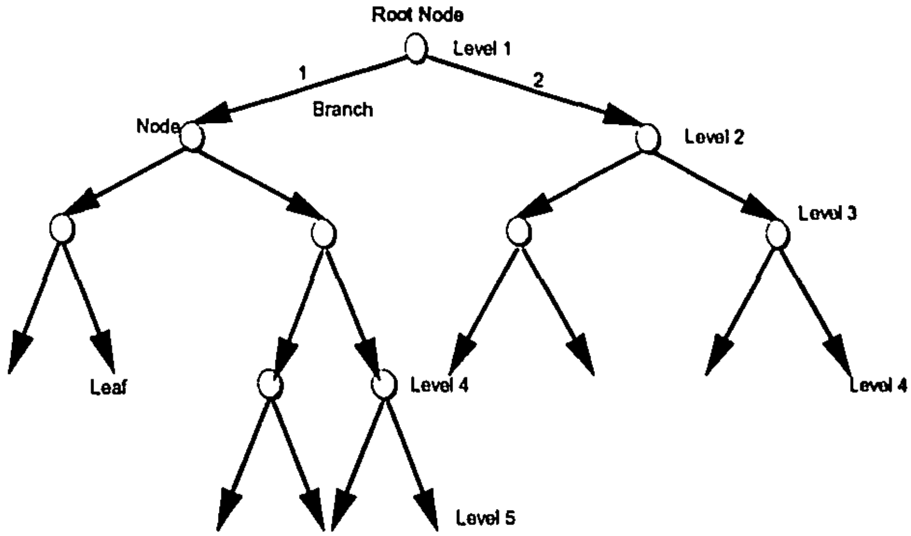



Figure 9-4 The simplified binary decision tree

### 9.3.2 Mining method selection decision tree

Based on our knowledge of mining methods and the decision tree structure introduced earlier, the representation of mining method selection heuristics as a binary tree is shown in Figure 9-5. The whole tree represents the entire searching space of the knowledge base and the knowledge of the tree designer. Other mining method expert may not fully agree with this tree and may draw another one to represent their understanding of mining method selection. The search of the tree starts from the root to leaves. Except from the leaves, all nodes are decision nodes. Each decision node contains a question for user to answer and each question can only be answered with a yes or no. According to the user's answer, the tree decides which branch to explore. Each leaf is a answer node which contains a solution to the search of the tree. In our case, each leaf contains a mining method. Once a leaf is reached, the mining method contained in that node is the recommended mining method.

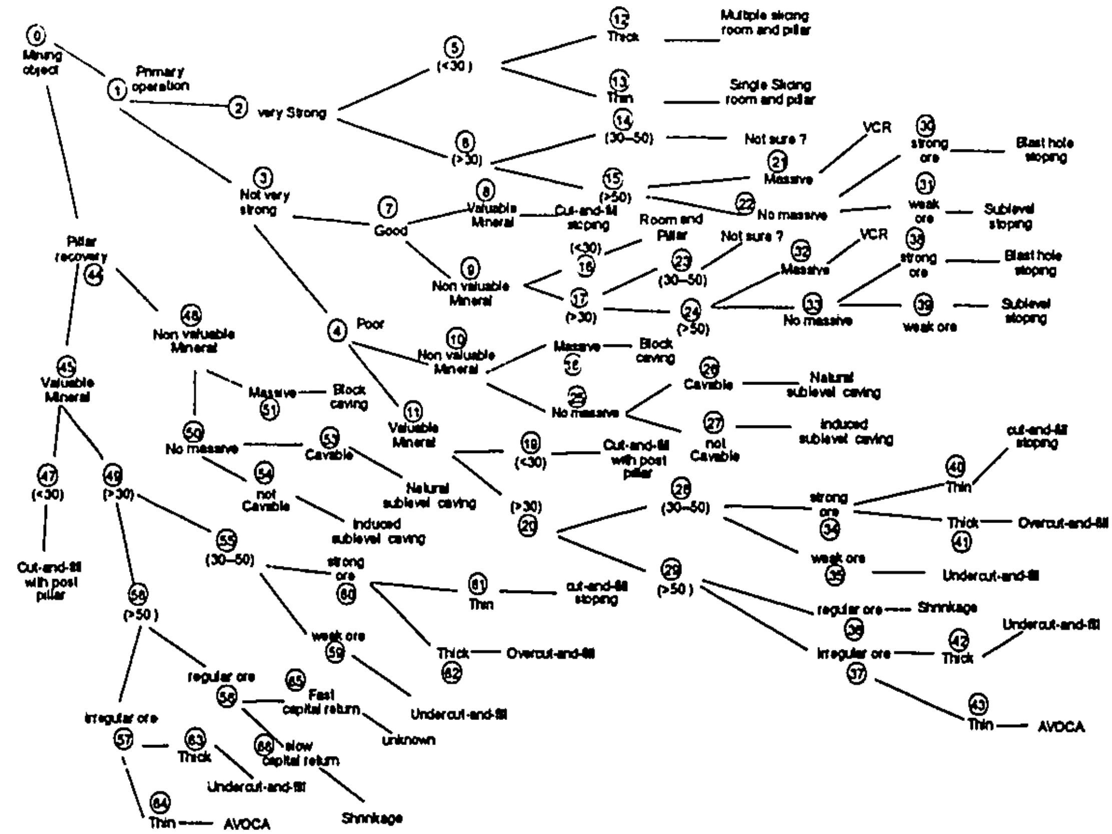



Figure 9-5 Representation of mining method selection heuristics as binary tree

Each node is assigned with an integer number as an identity. The tag printed beside the node is the possible answer to the question contained in the parents node. For instance, the root node is a decision node containing the question: 'It this method for primary mining operation?'. If the answer is yes, the search is directed to the node 1 since the tag beside the node is the primary operation.

Based on the decision tree, at least two functions can be implemented.

1. Select a mining method from the known list.

Assume a backfill designer is not occupied by any mining method and fully rely on the expert system. The decision tree can be used to provide a consultation service by asking several questions related to mining conditions and finally recommend a method. In this process, the search of the tree starts from root to leaf.

2. Verify a mining method

Assume a backfill designer is knowledgeable with mining method and has some ideas in mind. He might want to make sure the method he has in mind is feasible. The decision tree can be used to verify a speculation. In this case, the search of the decision tree starts from the leaf to root. If all questions are answered correctly, the speculated method is verified.

As stated earlier, the decision tree presented in Figure 9-5 is far from the complete knowledge of the mining method selection. It represents only expertise of one expert. Knowledge engineering is a bottleneck of expert system in general and has drawn lots of attentions in academic community. No intention has been made to explore the whole issue of knowledge acquisition. We propose a mechanism that adds new knowledge to a decision tree as it is learned. For example, the mining method decision tree may ask a series of questions to select a mining method. Once the decision tree has reached an answer, it asks if it is the correct answer. If the user is a mining method expert and sure it is a correct answer, then nothing more is done. If the answer is incorrect, however, the decision tree is modified to accommodate the correct answer. The answer node is replaced with a decision node containing a question that will differentiate between the old answer that was at the node and the answer that was not correctly guessed. In doing this, the decision tree will grow, and the knowledge base will contain more nodes and questions. It can be considered as a tool for knowledge engineering. The initial decision tree should not necessarily be complete. Certain knowledge can be missing like 'unknown' in decision tree. The expert system can be delivered to some real mining method experts and ask them to teach the system with their knowledge. Each mining expert will teach the system by their own experience. Some conflicts might quite possibly arise, and other tools are need to maintain the knowledge base, which is another issue out of the scope.

## 9.4. SYSTEM IMPLEMENTATION AND CLIPS

In last chapter, we introduced the basic concepts of expert systems and the expert system development shell or tools CLIPS. The earlier section of this chapter outlined the main concerns of mining method selection and formulated the decision process within the context of decision tree. We now demonstrate the techniques and methodology to implement the decision tree for mining method selection.

### 9.4.1 CLIPS

CLIPS is an acronym for C language Integrated Production System designed at NASA/Jonhnson Space Centre with the specific purpose of producing high portability, low cost, and easy integration with external systems. CLIPS was written using the C programming language to facilitate these objectives. CLIPS is a forward chaining rule-based language that has inference and representation capabilities similar to other expert system. Syntactically, CLIPS very close resembles the LISP language. Because of this high portability, CLIPS has been installed on a wide variety of computers ranging from PCs to CRAY supercomputers. We choose the CLIPS as a language to implement our mining method selection expert system.

The basic components of CLIPS are:

1. fact-list: global memory for data
2. knowledge-base: contains all the rules
3. inference engine: controls overall execution

**1. FACTS**

In order to solve a problem, a CLIPS program must have data or information with which it can reason. A "chunk" of information in CLIPS is called a fact. In CLIPS syntax, a fact consists of one or more fields enclosed in matching left and right parentheses. The following are all examples of facts:

(a "Multiple slicing room-and-pillar")
(b "Single slicing room-and-pillar")
(c VCR)
(d "Blasthole stoping")
(e "Sublevel stoping")
(f "Induced sublevel caving")
(g "Natural sublevel caving")
(h "Cut-and-fill stoping")
(i Overcut-and-fill)
(j Undercut-and-fill)
(k Shrinkage)
(l AVOCA)

This fact list states a relationship between the alphabet a, b, c,... and the name of a mining methods. For example, the fact (c VCR) states that the character 'c' represents 'VCR' method. The fact list can be created by any text editor such as EDIT of MS-DOS5.0 provided the CLIPS syntax is enforced.

**2. RULES**

In order to accomplish useful work, an expert system must have rules as well as facts. Apart from stating certain fact, rules state the relationships between facts. The general format of a rule in CLIPS syntax is:

```
(defrule <rule name> [<optional comment>]
        <<patterns>>  ;Left-Hand Side (LHS) of the rule
        =>
        <<actions>>)  ; Right-Hand Side (RHS) of the rule
```

The entire rule must be surrounded by parentheses. Each of the patterns and actions of the rule must be surrounded by parentheses. A rule may have multiple patterns and actions.

If all patterns of a rule match facts, the rule is activated and put on the agenda, which is a collection of activated rules. The symbol => that follows the patterns in a rule is called an arrow representing the beginning of the THEN part of an IF-THEN rule. The part of the rule before the arrow is called the left-hand side (LHS) and the part after the arrow is called the right-hand side (RHS).

The last part of a rule is the list of actions that will be executed when the rule is fired. The term fires means that CLIPS executes the actions of a rule from the agenda. A program normally ceases execution when there are no rules on the agenda. When there are multiple rules on the agenda, CLIPS automatically determines which is the appropriate rule to fire. CLIPS orders the rules on the agenda in terms of increasing priority and fires the rule with the highest priority, called salience. Salience can be assigned to a rule when a rule is defined. Like facts, a rule can either be typed into CLIPS or loaded in from a file of rules created by an editor as facts. As parts of knowledge base which needs permanent maintenance and repetitive use, facts and rules are normally saved in a text file and loaded to CLIPS when running the expert system.

### 9.4.2 Algorithm of traversing decision tree

With all the domain knowledge and implementation tools available, it is ready to build application program for mining method selection. As stated earlier, the traverse through the decision tree can go either from root to leaves or vice versa, and certain learning mechanism should be enforced. The first step in determining how a learning decision tree can be implemented in CLIPS is to decide how the knowledge should be represented. Since the decision tree must learn, it will probably be worthwhile to represent the decision tree as facts instead of rules. Facts can easily be added and removed to update the decision tree as it learns. A set of CLIPS rules can be used to traverse the decision tree by implementing the Solve_Tree_and_Learn algorithm using a rule-based approach[110].

Each node of the decision tree will be represented by a fact. Since the answer nodes and decision nodes hold different types of information, it will be necessary to use different templates for the different nodes. An answer node (leaf) will use the following template

(node <node number> answer <name of mining method>)

Where <node number> is the unique name for the node and <value> represents a mining method stored in the node. The word answer is used to indicate an answer node.

A decision node will use the following template

(node <node number> decision <question>
<yes-node> <no-node>)

where <node number> is the unique name for the node, <question> is the question that is asked when this node is traversed, <yes-node> is the node to proceed to if the question is answered affirmatively, and <no-node> is the node to proceed to if the question is answered negatively. The word decision is used to indicate a decision node.

Because the mining method selection program will learn, it will be necessary to store information about what has been learned from one run of the program to the next. Since the decision tree will be structured as a collection of facts, it will be useful to store them in a file in the load-facts command format and assert them using the load-facts command when the program begins and saves then using save-facts command when the program finishes. For our program, the facts will be stored in a file called "method.fac".

The procedure to traverse the tree represented this way is very simple depending on the purpose of running the program, starts either from root to select a mining method, or from a leaf to verify a speculation. For instance, assuming the purpose is to select a mining method, the inference process is started by setting the current location in the root of decision tree. If current location is a decision node, then the question associated with the decision node must be answered in some manner. If the answer to the question is yes, then the current location is set to the child node connected to the yes branch of the current location. If the answer to the question is no, then the current location is set to the child node connected to the no branch of the current location. If at any time an answer node becomes the current location, then the value contained in the answer node is derived through consultation with the decision tree. Otherwise the procedure for handling a decision node is repeated until an answer node is finally reached. If the finally answer is not correct by user's knowledge, the answer node contains the message 'unknown' as shown on decision tree (Figure 9-5), however, the decision tree can be modified to accommodate the correct answer. The answer node is replaced with a decision node and the answer that was not correctly guessed. The following is the pseudocode of Solve-Tree-and-Learn algorithm:

```
Procedure Solve_Tree_and_Learn
        Set the current location in the tree
                to the root node
         while the current location is a decision node do
         Ask the question at the current node.
         If the reply to the question is yes
                Set the current node to the yes branch.
         else
                Set the current node to the no branch.
         end do

         Ask if the answer at the current node is correct.
         if the answer is correct
                return the correct answer.
         else Determine the correct answer
         determine a question which when answered yes
                will distinguish the answer at the current
                node from the correct answer.
         Replace the answer node with a decision node
                that has as its no branch the current answer
                node and as its yes branch an answer node
                with the correct answer. The decision node's
                question should be the question which
                distinguishes the two answer nodes.
         end if
end procedure.
```

Following is the examples of basic rules to initialize, explore decision nodes and answer node. The sentence started by a semicolon ';' is comments and ignored by CLIPS.

```
;Now start the definition of the rule to traverse the tree
(defrule initialize
     (specification)
     =>
    (load-facts "method.fac")
     (assert (current-node root)))
```

```
(defrule do-decision-node
        (specification)
    ?node <- (current-node ?name)
    (node ?name decision ?question
            ?yes-branch ?no-branch)
    =>
    (retract ?node)
    (printout t
"****************************************************************" )
(format t "         %s" ?question)
(printout t crlf
"****************************************************************"
"Please answer the question by typing yes or no:" )
(bind ?answer (read))
(if (eq ?answer y)
     then (assert (current-node ?yes-branch))
     else (assert (current-node ?no-branch))))

(defrule do-answer-node
    (specification)
    ?node<-(current-node ?name)
    (node ?name answer ?value ?explanation)
    =>
    (printout t
"****************************************************************"
"      I guess it is a " ?value  crlf
     ?explanation  crlf)
(printout t
"      Do you Agree? (yes or no)   )
(printout t
"      If you are mining expert and don't agree, answer      "  crlf)
(printout t
"      no and teach the system what method should it be      " crlf)
(printout t
"      Otherwise type yes to accept the recommendation       " crlf
"****************************************************************"
```

```
"      Answer the question by typing yes or no: ")
(bind ?answer (read))
(if (eq ?answer y)
      then (assert(ask-try-again))
           (assert(method ?value))
           (retract ?node)
      else    (assert (replace-answer-node))))
```

### 9.4.3 Performance of the expert system

The program runs as a consultation conversation between a mining designer and a computer expert system. As an individual stand alone program, it runs on full screen in DOS operating system. As a mining design tool to support intelligent links in hypermedia backfill design manual, it is integrated as part of hypermedia system and can run within MS-Window environment. In MS-Window environment, mining method selection expert system can run in a window background. Figure 9-6 shows the starting window for user to specify the task to perform under MS-Window environment.

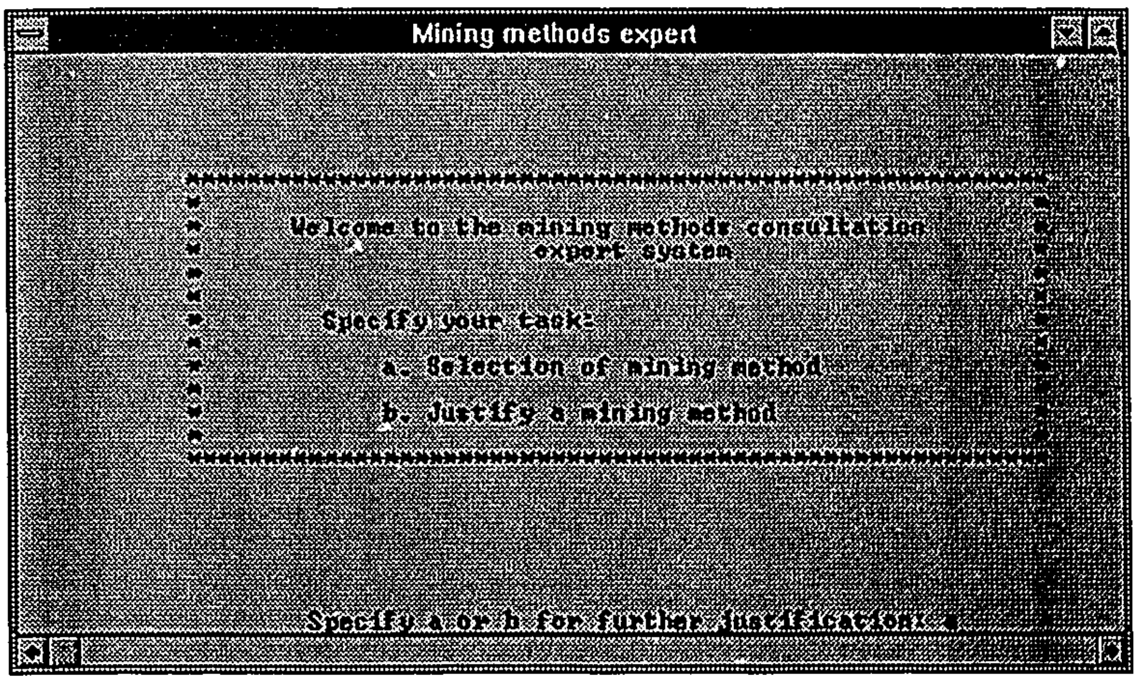



Figure 9-6 The starting window of mining methods selection expert system

If the user answers a, the following conversation is shown in Figure 9-7, which sets the current node on decision tree to the root node and asks the question contained in that node.

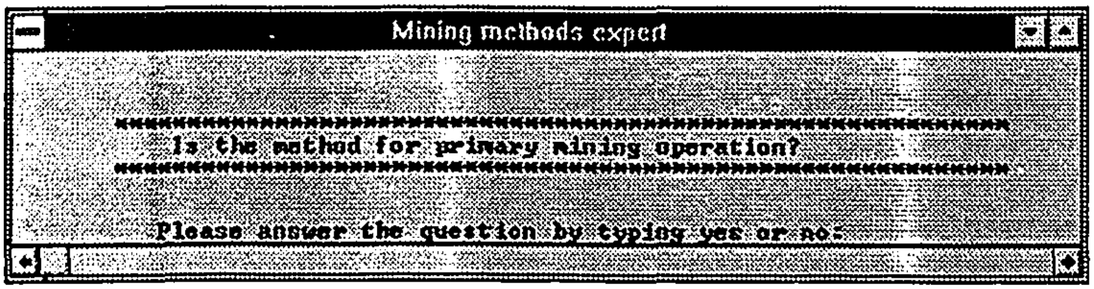



Figure 9-7 The conversation between expert system and mining designer

After several questions, a leaf node will be reached and a mining method recommended as shown in Figure 9-8. The message displayed on the window includes a recommended mining method by the expert system and a simple explanation why the method is recommended. At this point, the user has the choice to decide whether accept the recommendation or try another session. If the user is a mining method expert and do not agree with the solution reached, he can teach the system by answering no and continue the conversation.

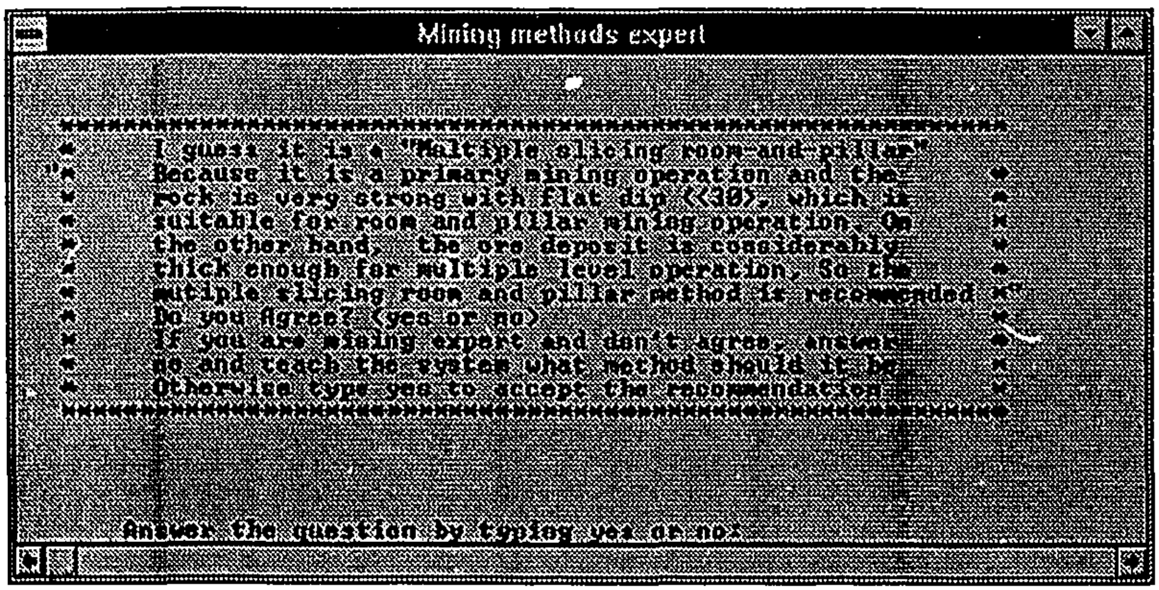



Figure 9-8 The recommended mining method by expert system

Figure 9-9 shows the conversation when a user disagree with the solution reached, and try to teach the system with new knowledge. The system ask user's justification of new mining method followed by a question that will distinguish the original one with the new one. In this example, the human expert recommends the VCR method with his justifications. The newly taught method and justification are saved permanently in facts file as knowledge base for further use.

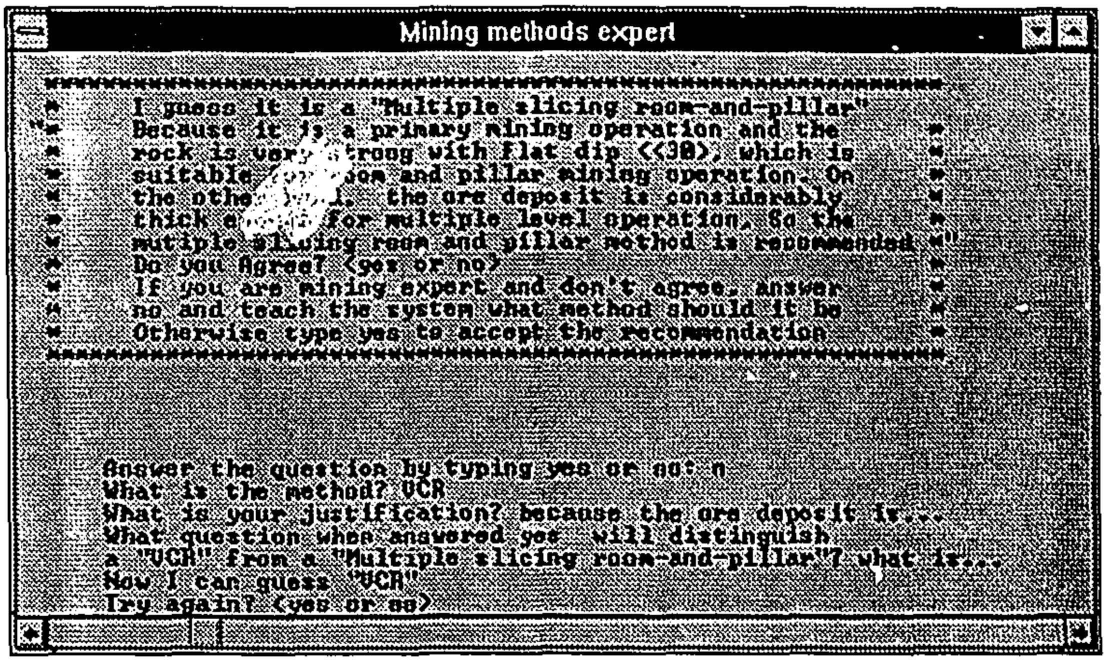



Figure 9-9 The teaching conversation between mining expert and the expert system

At this point, the user can either accept the recommendation or start the session to justification his suggestion.

### 9.4.4. Case test of the expert system

Even though the decision tree shown in Figure 9-5 is not a complete knowledge of mining method selection in practical mining operation, the main principles and general rules of mining method selection are very well respected in general, therefore it embraces many practical mining operations in reality. As an example, we present the mining practice using cut-and-fill stoping in Mount Isa Mines Ltd, and demonstrate how the solution reached by mining expert system complies with the real world activity.

According to B. Hornsby and Staff[111], the general description of the ground condition is as the following:

The ground is fairly competent, but requires extensive rockbolting for secondary support. The ore values are 6.5 zinc by weight, 6.4 lead by weight, and 150g/t (4.5g per st) silver. The ore bodies are restricted to the Urquhart shale, a formation of thinly bedded pyritic, dolomitic, and volcanic shale. Towards top of the Mount Isa group sediments, copper and silver-lead-zinc ore bodies occur as contiguous but discrete entities which need the selective mining operation. The dips varies between 40 to 55$^0$.

The current mining method is basically cut-and-fill stoping. The reasons for adopting the cut-and-fill stoping method are various. Among the reasons are the method for the silver-lead-zinc ore bodies and its suitability for selective extraction of ore, ability to cope with poor wall rock conditions, high production rate, and continuity of production. The cut-and-fill stoping is used in the footwall ore bodies, whose hanging walls are, in general, not as competent as the ore bodies in the hanging wall of the sequence, and whose widths are less. The method is restricted to a maximum horizontal miming width of 11 m for ground control and a minimum horizontal mining because of the size of equipment used.

In our decision tree, there are two paths leading to the cut-and-fill stoping method:

1. node0 - node1 - node3 - node4 - node11 - node20 - node28 - node34 - node40
   - "cut-and-fill stoping";
2. node0 - node44 - node45 - node49 - node55 - node60 - node61
   -"cut-and-fill stoping"

The questions corresponding to the chain are:

1). Is the mining method used for primary operation?

2). Is it used for very strong host rock?

3). Is the host rock condition poor?

4). Is it used to mine valuable mineral?

5). Is the dip of ore body less than 30$^0$?

6). Is dip of ore body greater than 50$^0$?

7). Is the ore rock competent?

8). Is the ore body thin?

Apparently, these questions are not clearly defined, therefore, the answers to the questions will be very subjective. However, the potential users of the mining methods selection expert are expected to be the mining engineer. These questions such as the 'Is the ore rock competent?' , or 'Is the ore body thin?' should make clear sense to them. Based on the description of Mont Isa mine, the following answers to these questions should be acceptable:

1). Yes, the mining method is used for primary mining operation;

2). No, the host rock is not very competent;

3). Yes, the host rock is poor;

4). Yes, the mineral is valuable;

5). No, the dip of ore body is greater than 30$^0$;

6). No, the dip of body is not greater than 50$^0$;

7). Yes, the ore rock is competent;

8). No, the ore body is not thick;

Running the mining methods expert system and answering the questions with these answers, the system will recommend a cut-and-fill stoping method as the solution, which is expected. Since the answers are not unique depending on the understanding of the questions of individual user, the final recommendation might be different. But the recommended alternative methods will have similar applicable conditions and can be used as a candidate method for further evaluation.

## 9.5 SUMMARY

The mining method selection problem falls into the category of the heuristic classification, which is suitable to be modeled by decision tree approach. To be able to represent the mining method selection process as a decision tree, the mining methods are classified into three categories in general: 1). Stope requiring minimum support, 2). Stopes requiring some additional support other than pillar, and 3). Caving methods. Meanwhile, the basic characteristics of different mining methods in each group are discussed as the first step for knowledge acquisition. The formal representation of mining method selection process is represented as a binary tree which is defined for prototype mining methods selection expert system.

Based on the decision tree representation scheme, CLIPS expert system shell is used as underlying implementation tools of prototype system. Some examples of CLIPS syntax are illustrated to demonstrate the different language feature comparing with conventional procedural language. In addition, the Solve-Tree-and-Learn algorithm is introduced to traverse the decision. Three basic functions are implemented with the prototype mining method selection expert system. 1). Traverse from root to leaves to select a mining method; 2). Traverse form a leaf to root to justify a hypothesis; 3). Learn knowledge from end users.

The performance of the system as a individual tools demonstrate a consultation procedure between mining designer and computer expert system under MS-Windows environment. The results show the consistency of the solution reached by expert system and human experts in real mining operation. Other uses of mining methods selection expert rather than an individual tools will be discussed in next chapter.

As a technique remarks, it is worth to indicate that the learning mechanism introduced in the Solve-Tree-and-Learn algorithm serves only as the technique notion of using expert system. It is not intended to introduce this kind of learning mechanism as implementation techniques in general. One apparent and severe drawback of this algorithm is that it does not provide any mechanism to resolve the conflicts possibly input by different experts to maintain the truth of the knowledge base. The management of knowledge bases from each individual mining experts is still heavily relied on the human intervention.

# CHAPTER 10

# A HYPERMEDIA-BASED SYSTEM FOR BACKFILL DESIGN AND TRAINING

## 10.0. INTRODUCTION

The backfill design rationale defined through chapter 2 to chapter 7 outlines the basic scope and disciplines involved in the backfill design process. In general, it deals with the following aspects:

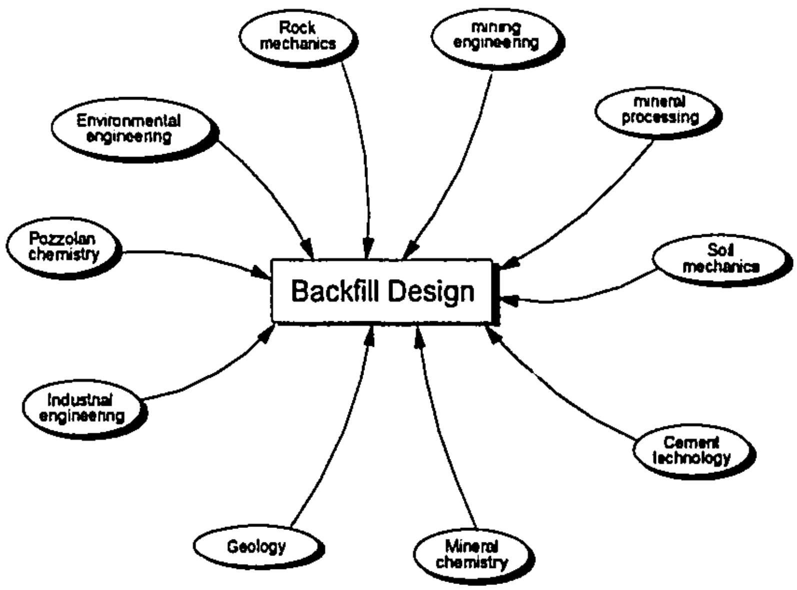



Figure 10-1 The merging technologies into backfill design

As a relatively new mining technique, the approach to backfill design and evaluation in practical mining operation is still not beyond trial and error stage, although some rules and design rationales have been established through recent research [6]'. The traditional backfill design is a multi-phase information intensive procedure. Bringing a wide range of relevant data and knowledge from different fields to support the necessary decision making is very time consuming and expensive. It is more cost effective to have a computer system that can manage information related to certain design problem in various formats such as text, graphics, video picture or sounds, and one which provides flexible facilities to access the information from different considerations. The conventional database technology supports only a structured data format, i.e. flat records, for example. Other unformated information such as graphics, sound or video can not be integrated within a single database environment[113], which is not adequate to meet with the needs of most engineering applications like backfill design.

The full scale support of paper-based information system on computer-based system, lies to the advent of the hypermedia technology. Hypermedia system supports the unformated information and non-linear access and, therefore opens the possibility to build various application tools for backfill design in real mining practice. In the context of integrated decision support system for backfill design, the hypermedia database system is defined to provide various information related to backfill design. More specifically, in our prototype system, the hypermedia system is designed to serve as a reference manual book for quick information retrieval. This chapter discusses the basic concepts of hypermedia technology and its possible applications in mining engineering. As an example, a prototyping system of backfill design reference manual was implemented using KnowledgePro, which supports both text and graphic representations of information. In addition, the integration of the expert system to support dynamic linking is successfully achieved by blackboard architecture. Finally, we compiled the potential applications of hypermedia system to mining design, mining training, and simulation.

## 10.1. BASIC CONCEPTS OF HYPERMEDIA SYSTEM

### 10.1.1. The definition of hypermedia and its features

Hypermedia are some times referred to as hypertext. In this chapter the term hypertext and hypermedia are used interchangeably. The simplest way to define hypermedia is to contrast it with traditional text like a book. All traditional text, whether in printed form or in computer files, is sequential, meaning that there is a single linear sequence defining the order in which the text is to be read. Hypermedia is nonsequential. There is no single order that determines the sequence in which the text is to be read[114]. The discrete pieces of data/information are connected by links. When the data can be graphics or sound or other media, the structure such defined is referred to as hypermedia. Conceptually, a hypermedia database consists of a network of nodes and links where data (document text, graphics, etc.) are the nodes, and the links are cross-references. The hypermedia presents several different options to readers, and the individual reader determines which of them to follow at the time of reading information. This actually allows users to navigate through a network of chunk of information. Figure 10-2 shows an example of this network. Each rectangle marked by A, B...F represents a node which contains document of text (node B), figure(node A), sound(node D), photo(node C), animation(node E), and film(node F).

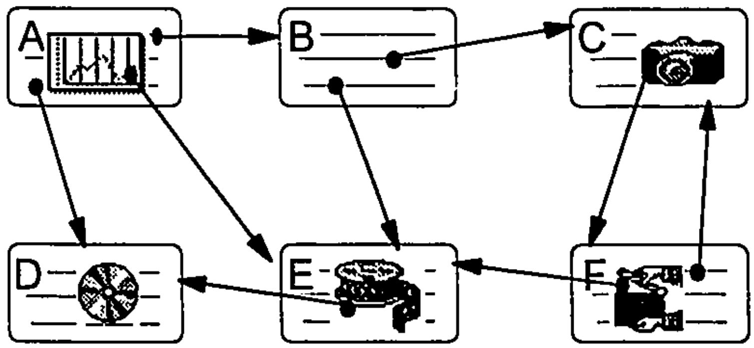



Figure 10-2 A simple hypermedia structure

This kind of unrestricted navigation among information coincides with the associative nature of thought in which people think. The theoretical properties of associative memories have been studied for a number of years, and the fundamental advantage and disadvantage of free association from both viewpoints of cognition and information retrieval have been discussed by many researchers[115], [116], [117]. For our purpose, we claim that hypermedia introduces two fundamental changes in the way the information is stored and retrieved. The first is the capability to move rapidly from one part of a document to another by means of an associative link. The sequential pattern of reading so familiar from the print world is replaced by a truly interactive format. The second change lies in the sharing of information across different machines and systems in different format of data[118].

### 10.1.2. The essential concepts of hypermedia system

As shown in figure 10-2, the hypermedia database consists of **nodes** and **links**. A single document in a hypermedia database is called a node. Each node in a hypermedia system corresponds to one or more screen displays. A document usually describes a single concept or topic. Hypermedia documents are usually written so that they are self-contained and do not depend upon the user's having viewed other documents. Continuity to other documents is provided by links. Some hypermedia systems allow nodes to be different types (e.g. references, annotations, illustrations, etc.). Typed nodes are usually identified by different colors, fonts, or icons when viewed by users. Nodes can also be composites, that is, several nodes are linked together and referenced by a single name. This is useful if the screen size is fixed and no scrolling is allowed. Because of the modularity inherent in hypermedia documents, they are not well suited to information that has a strong sequential nature (such as traditional prose or poetry). On the other hand, for information that has a richer network organization (such as reference books, texts, guidelines, catalogs, and technical manuals), hypermedia provides many advantages over traditional linear formats.

Link is another important concept in hypermedia. Links are the labels that connect one node (document, article, topic) with another. When a link is activated (e.g. by selecting it with a mouse or arrow keys), a jump is made to the document the link points to. A link may be embedded in the body of the document, embedded as part of a graphic or video image, listed at the end of the document, or contained in an index. In the application level, links are usually denoted by words of phrases that are highlighted in some fashion, but they can also be graphics or icons. For example, each component of a schematic diagram may be a link to a more detailed schematic of that component or to a text description. Links can produce a variety of results such as:

- transfer to a new topic
- show a reference (or go from a reference to the article)
- provide ancillary information, such as a footnote, definition, or annotation
- display an illustration, schematic, photograph, or video sequence
- display an index
- run another program (any procedural or non-procedural call)

A hypermedia system may have only one type of link or many types. In some system, the user may be able to select which link types are active. Links are usually given names that may or may not be shown to the user. The link name can indicate two nodes connected and its type. In addition to the standard links connecting two nodes, some hypermedia systems also have super-links to connect a larger number of nodes. There are several possibilities for dealing with having a single link connected to several nodes. The two simplest options are either to show a menu of the links or go to all the destinations at the same time. The alternative and obviously intelligent way to deal with multiple destination would be to have the system choose for the user in some way. We will discuss the integration of the expert system as inference engine for automated decision making and demonstrate a prototyping hypermedia system enhanced with this mechanism.

Although claimed to provide flexible and free navigation among information, hypermedia system easily leads to disorientation and chaos if the information is not properly organized or if tools are not provided to support the information browsing. A common technique to reduce the disorientation during the navigation is to impose certain structure on hypermedia database such as hierarchy. In a hierarchy, each node has a parent (superordinate concept) and a child (subordinate concept) unless the node is a starting point (root) or an end point (leaf). A hypermedia database organized as a hierarchy can be drawn as a tree structure. Figure 10-3a shows a network structure hypermedia database system with arbitrary cross-over links. The figure 10-3b shows a hierarchy structure of hypermedia database with no cross-over links.

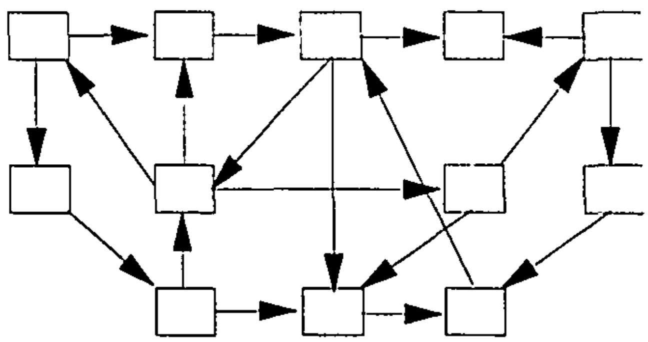


Figure 10-3a Network structure of hypermedia database

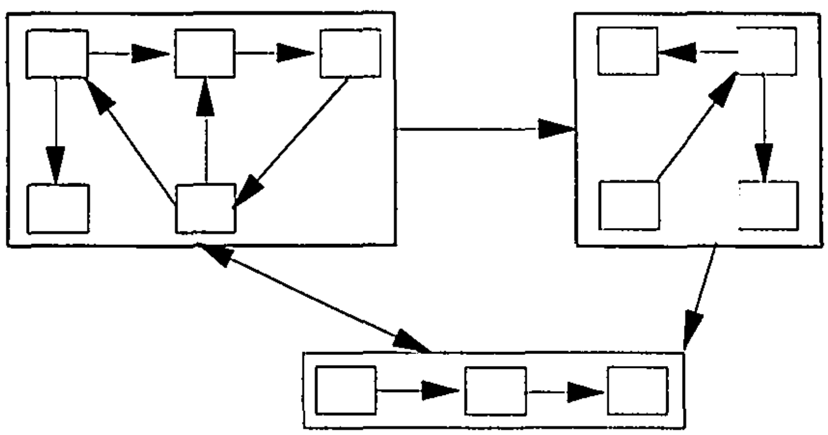


Figure 10-3b Hierarchy structure hypermedia database

The advantage of a hierarchical structure is that all links must follow an orderly route through the tree, connecting superordinate and subordinate nodes. So the data are well defined, and various display techniques such as fisheye or hierarchical views can be applied. The disadvantage of a hierarchical structure is that the flexibility of the links among nodes is limited. All nodes must be linked together via some super/subordinate concepts, not in an arbitrary manner. Compromise between the flexibility and usability has to be made when designing a hypermedia database.

The basic operation on the hypermedia system is browsing. With little training in computer concepts and with little knowledge of subject domain, hypermedia users can casually traverse nodes and links looking for something of interest. Direct manipulation enables an easy-to-use mode of interaction. The above discussion presents the basic concepts of hypermedia system. Other auxiliaries such as searching, indexing, windowing, filtering, bookmarks and path history are also important components to make the hypermedia system useful[119], [120].

### 10.1.3. The architecture of hypermedia systems

One can distinguish three levels of a hypermedia system[121]:

- Presentation level user interface
- Hypermedia abstract machine level: nodes and links
- Database level: storage, shared data, and network access

The database level is at the bottom of the three-level model and deals with all the traditional issues of information storage that does not really have anything specifically to do with hypermedia. The hypermedia nodes and links are just data objects with no particular meaning to database level. Each of them forms a unit that only one user can modify at the same time and that takes up so many bits of storage space. The hypermedia abstract machine level sits in the middle between the database and user interface levels. This centre is where the hypermedia system determines the basic nature of its nodes and links and where it maintains the relations among them. It is in this level that the import-export formats for hypermedia are standardized. Since the database level has to be heavily machine dependent in its storage format, the user interface level is highly different from one hypermedia system to the other. The user interface level deals with the presentation of the information in the hypermedia abstract machine, including such issues as what commands should be made available to the user, how to show nodes and links, and whether to include overview diagrams or not. Assume that the hypermedia abstract machine level defines the links as being typed. The user interface level might decide not to display that information at all to some novice users and to make typing information available only in an authoring mode. Since most hypermedia systems support direct manipulations, the user interface has to provide all the interactive facilities for users to manipulate, which is another reason that makes the hypermedia system a suitable application for graphics user interface design tools.

## 10.2. APPLICATIONS OF THE HYPERMEDIA SYSTEM

A number of traditional hypermedia application areas have been reported, which include dictionaries, encyclopedias, medical textbooks, product catalogs, creative writing, help systems, technical documentation, instruction, software engineering, religious studies, and museum exhibits. More recent investigations illustrated some ground areas where hypermedia system works as a domain specific tool for problem solver and adviser. One interesting area this chapter will discuss intensively is using hypermedia system as a decision support system for engineering design and training in mining engineering. As mentioned early in the introduction section, backfill design is data intensive involving different format of data. Access to the knowledge base is essential for backfill operation design, which is usually non-linear, and cross-referencing is common. Hypermedia systems are fully capable of handling this type of information store and retrieval. Beyond that, different user interfaces can be built to provide interactive reference and advice program for use in the backfill design both as a design tool and training tutorial.

### 10.2.1. Hypermedia database as backfill design reference manual

A primary area of application of hypermedia systems is as reference manuals to aid in the design and control of backfill design and operation. The chapter 2 to chapter 7 present a collection of distributed information of backfill design and operation, which cover most of issues related to backfill technology and is still growing[122]. For a specific mining operation designer, the access to this information base is most likely non-linear. The task-driven and experienced designer, of course, wants faster access to only relevant information. One may refer to a specific portion of information casually. But for a novice designer, one may need to get acquainted with the basic concepts first and then go into specific area concerned. In any cases, access to the information as a whole and following certain sequence is highly unlikely to happen if not possible. In a paper-based information system, users under time pressure often find themselves flipping from one section of a printed manual to another in a number of time-consuming iterations and, at the same time, trying to keep a finger in four different sections at once. This is very costly in terms of cognitive overhead and information retrieval.

The purposes of accessing the information base, as far as backfill design is concerned, are:

- Looking for basic concepts with regarding to the specific engineering problem;
- Looking for technology already available for problem solving;
- Looking for methodologies already known in general;
- Looking for regulations to follow;
- Looking for successful examples to draw analogy and conclusions;
- Looking for tools to make decisions;

To meet with these needs, we need to distinguish two primary areas of information. One is the reference documents such as standard specification described in the backfill design handbook, literature and other government regulation. The other one is the practical operation information and the knowledge of experienced personnel. For the first one, the knowledge is represented explicitly by certain format which could be text, graphic and so on. However, the second one needs personal judgment, and implicitly exists among the successful designs. Proper process of the knowledge, such as automated reasoning, or expert systems, is needed to make it useful. We proposed an integrated system to support both hypermedia reference manual and inference engine as backfill design tool. The prototype application was developed to provide a reference manual as a help facility as part of design tool. It should provide basic concepts of backfill design and the main technologies available.

### 10.2.2. Hypermedia system design

The process of creating hypermedia documents and databases is called authoring. The hypermedia author must make a series of decisions about how to organize the entire database and individual documents. As stated in the previous section, one of the strategies is to impose certain structure on the document like hierarchy which we adopted for our system. Then the challenge will be to structure the knowledge in a way that an overview can be presented to the reader in the root document or introductory article. The overview should identify the key subsidiary ideas and the breadth of coverage. Figure 10-4 shows a hypergraph representation of top level hierarchy of the backfill design reference manual:

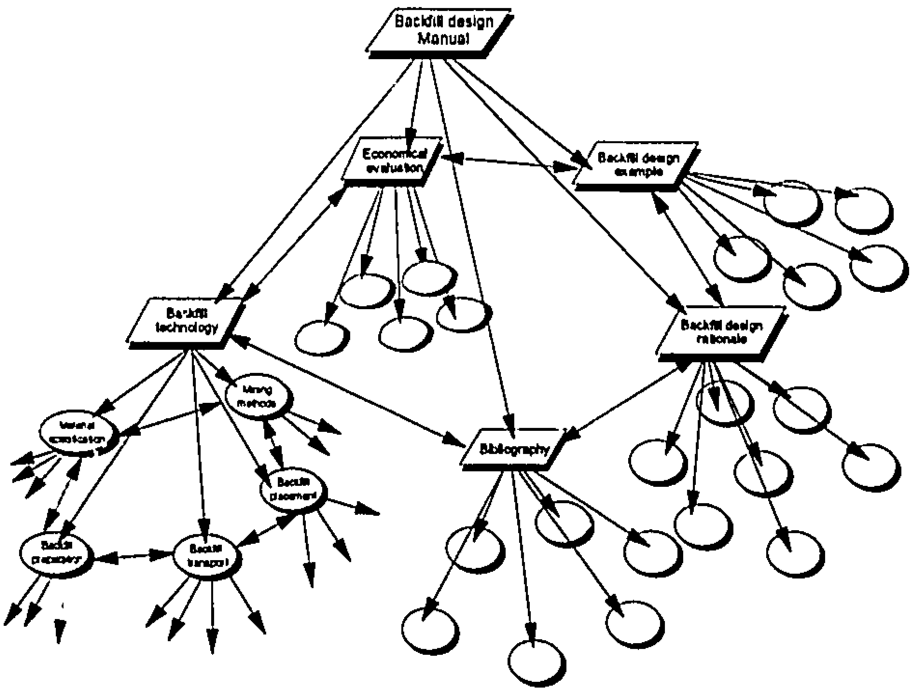



Figure 10-4 The top level hierarchy of backfill design reference manual

On the top of the hypergraph is the root document, '**backfill design manual**', which serves as an entry to the hypermedia system. The prototyping system does not support structural browsing facility, but rather provides an embedded table of contents as the overview of the scope of the hypermedia system. "KnowledgePro" is used to develop the hypermedia system which is a PC-based programming tool, and fully integrated within the MS-Windows environment. Other features and advantages of KnowledgePro become evident when integrating expert system to support intelligent links. Figure 10-5 shows the root document which gives a short introduction to the backfill design reference manual and its organization. The document is displayed on a window facilitated with scrolling button. The underlined texts are links pointing to other nodes. Other type of links and commands is implemented as buttons on the top of the windows. These buttons are implemented as standard objects for root display window and automatically inherited by the lower level of nodes display windows. The button '**Open**' lets the users select different hypermedia database to explore. In other words, it is a special type of link that leads to other hypermedia documents. The button '**Index**' is a special link which always points to the root document. Whenever the user gets lost or wants to go back to the root document, simply by clicking the button, the system will return to the original entry of the hypermedia system. The '**Back**' button always points to the previous node to simulate bi-directional link. The data structure used to implement history path is a List which is a built-in data type supported by KnowledgePro. This history path list keeps track of the browsing chain that the user has explored and can be used to monitor the user's intention of interaction to the hypermedia system, which may provide information to make heuristic inference if necessary. The rest buttons perform the functions implied by the button name. The system supports direct manipulation of the window objects (links, buttons,...etc.). For example, if reader is interested in backfill design rationale, one simply needs to point the mouse to the text **The backfill design rationale**, the linked node is presented as shown in figure 10-6.

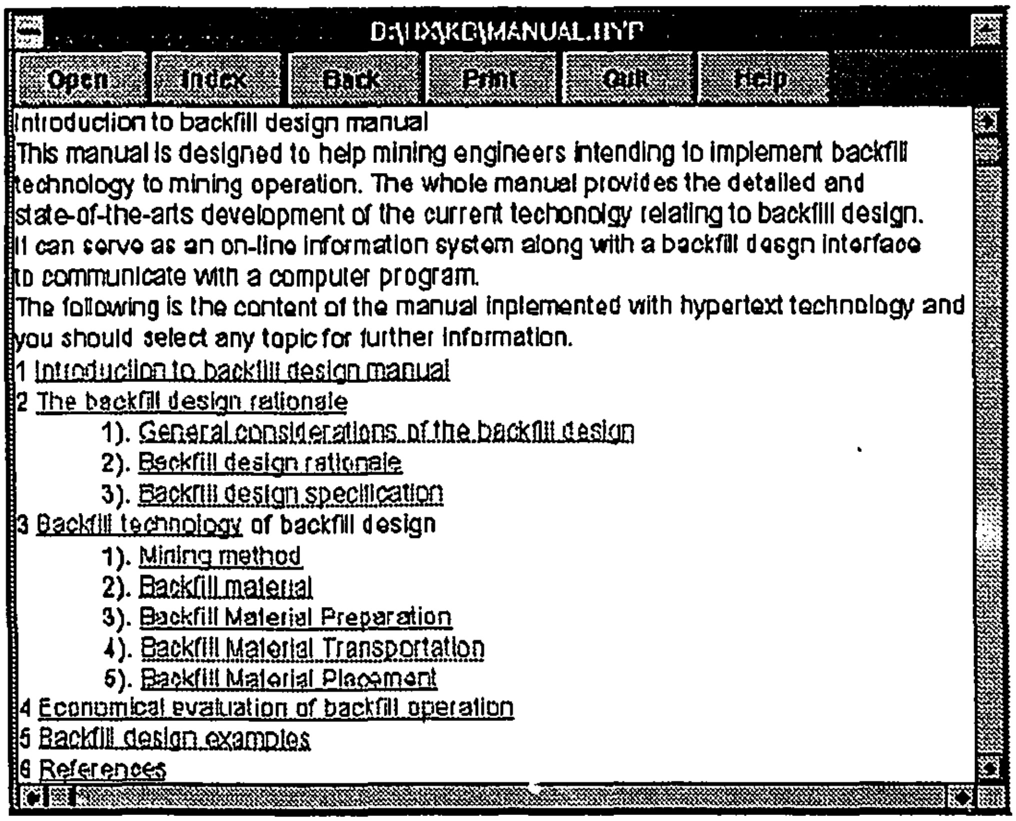



Figure 10-5 The root document of the hypermedia system as entry node

Under the root are the second level documents consisting of five subnodes, namely, **The backfill design rationale**, **Backfill technology**, **Economical evaluation of backfill operation**, **Backfill design examples** and **References**. Each node groups the closely interrelated ideas and frequently cross-referenced concepts together. Nodes of the same level are linked as shown by the arrows. The nodes belonging to different parents are not directly linked but rather linked through the higher level nodes. This kind of hierarchy does sacrifice the flexibility of the hypermedia system. The user will no longer be able to jump freely from one idea to another. However, the authoring process of hypermedia document does take into account the most possible exploring route and structures the hypermedia document to present the information as it should be. The loss of freedom is compensated by a well organized structure. This is the decision made in the design stage.

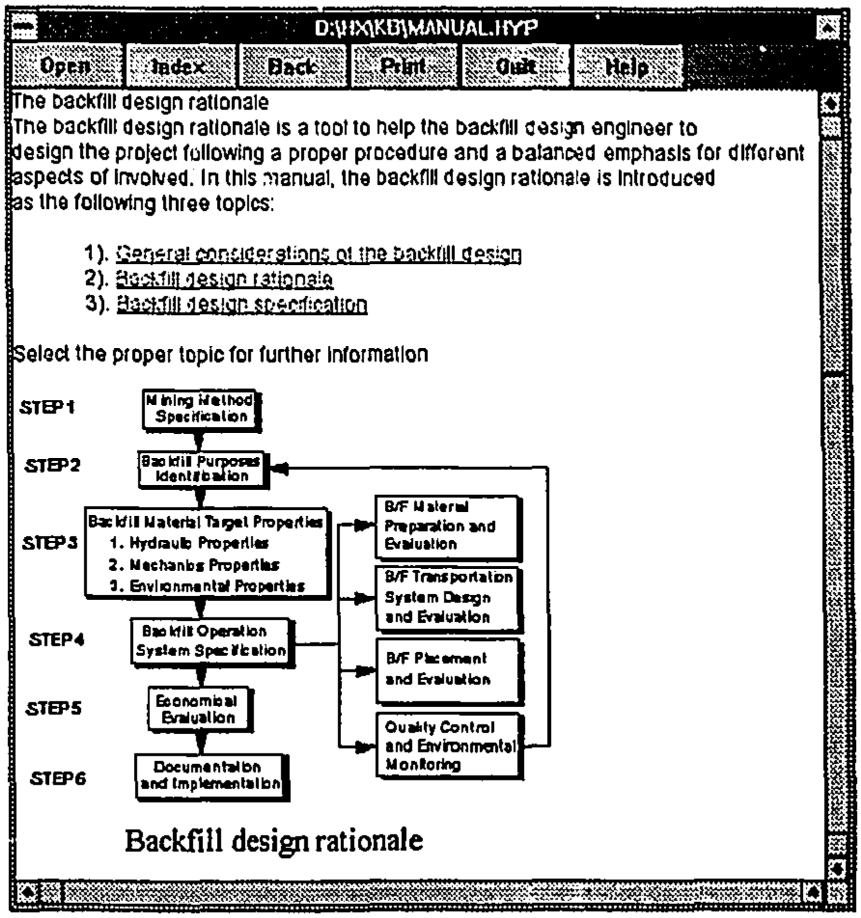



Figure 10-6 The second level nodes linked by the link <u>The backfill design rationale</u>

In addition to present the link in text and button, the link can also be presented by certain graphics on the screen such as icons and any graphical figures, which is also supported by KnowledgePro. Our prototype system implemented graphic link for graphics interface to the hypermedia system, which is another important application growing out of the hypermedia technology. Shown in figure 10-7a is a node document containing the link **'Cyclone'** and **'Pump'** pointing to other hypermedia node 'cyclone' and 'pump' respectively, As a convention, we use terms started with capital letters to refer to link names, while the terms started with lowercase letter to refer to node names.

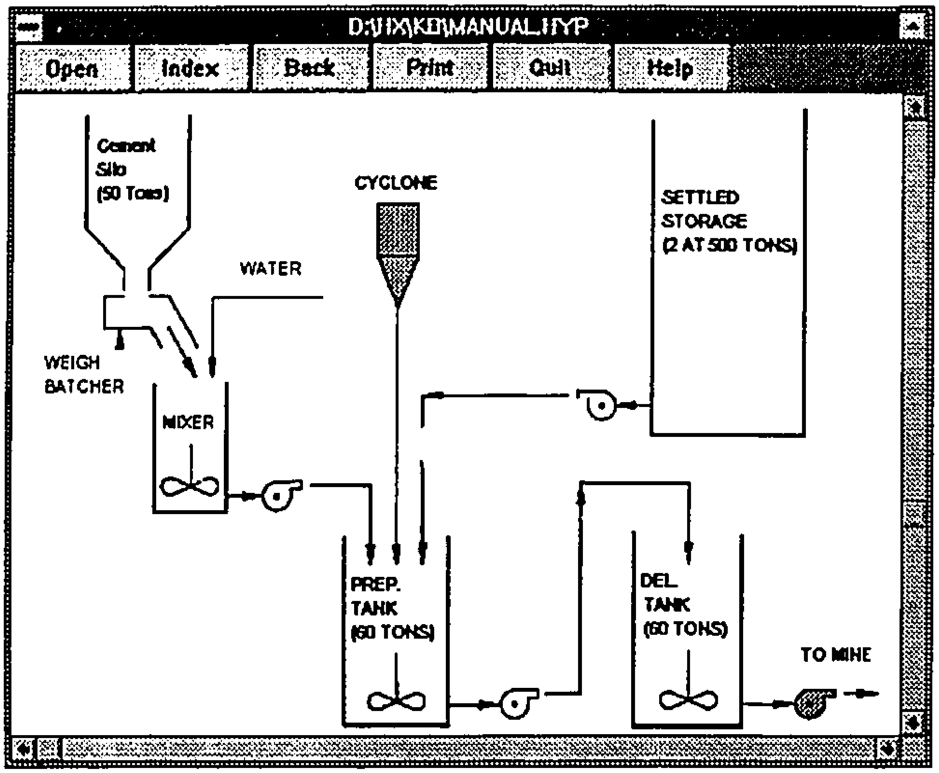



Figure 10-7a The node document contain graphics objects as links

The gray colored objects cyclone on the top middle of the figure and pump at the right bottom corner imply that these are graphical object links pointing to other nodes containing information about the objects. Manipulation of the graphical links is the same as the text and other objects by using mouse to click on the object interested. Figure 10-7b is the result document of clicking the link 'Cyclone' on figure 10-7a.

This example clearly illustrated the associative style of information retrieval. When a backfill designer starts accessing to the hypermedia database, information is presented as if he is reading through various paper-based reference manual books relating to several fields at the same time. The user is released from the burden of reading different manual at the same time, not to mention the frustration of case to case searching for information from various sources. The information stored in hypermedia system can be easily updated to the state-of-the-art, which is another advantage over paper-based system.

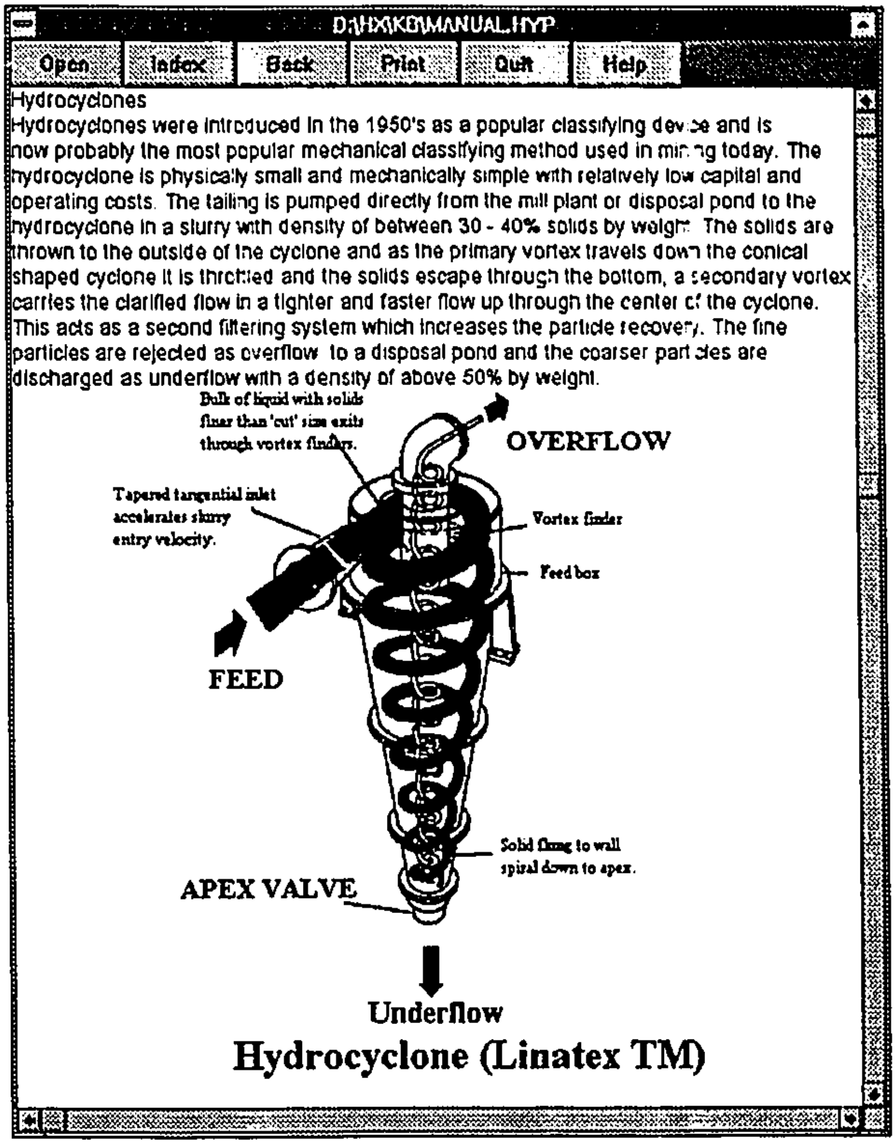



Figure 10-7b The Hydrocyclone node linked by link Cyclone

## 10.2.3. Integrate expert system within hypermedia

The backfill design process goes beyond simple information retrieval process. More complicated tools are needed to perform the task of decision support functionality. Integrating certain AI models with information retrieval research has produced intriguing hybrid systems. Of particular interest are expert systems which constitute intelligent interfaces to a hypermedia database. Adding the intelligence of an expert system requires the use of additional components -- a knowledge base and an inference engine. The expert system embodies the facts, information, knowledge, rules-of-thumb, and other elements of heuristic expertise. The user may consult this expertise as an expert advisor or a consultant to solve problems or make decisions. Three approaches to conjoining expert systems and hypermedia had been proposed[123] :

1. Separate knowledge base and hypermedia system
2. Merge knowledge base with the hypermedia system
3. Embed or distribute expert systems within the hypermedia system

For our purpose, we proposed distributing expert systems within the hypergraph to support intelligent links both as search strategy and decision making tools. The KnowledgePro combines the capability of expert system, object-oriented programming language, and a hypermedia product into one seamless and highly modular environment[124] and therefore provides the basic building blocks. In the interface level, the expert systems were implemented as problems solving tools as discussed in last chapter, which can be consulted directly from the user interface for decision making alone. In the hypermedia abstract machine level, an expert system is integrated as an intelligent links to connect nodes. A simple example is the multi-destination problem. One link may connect to more than one nodes. The dummy implementation simply authors an intermediate node and lists all possible links for further browsing. The disadvantage of this strategy is that the user has to try all possible links to find relevant information. This is especially frustrating if too many possible links involved. For example, in the process of backfill design, the first important decision faced by backfill designer is to specify a proper mining method. There exist many mining methods, and each of them works for specific mining conditions and requirements. The '**Mining method**' link shown in figure 10-5 might connect to several mining methods. As an option, the mining methods selection expert system is integrated to provide intelligent link. Implemented using CLIPS expert system shell, the expert system is initially designed as a separate decision support tool and then integrated as intelligent link. The KnowledgePro and CLIPS are different tools. The integration is accomplished by blackboard approach which allows the information of multiple sources to be represented in a single representation system. The blackboard approach has been intensively discussed in chapter 8 as the underlying architecture of overall decision support backfill design system. We introduced the basic concepts in the abstract level at that time. We are now introduce the techniques in implementation level. In essence, the blackboard approach opens a unique publication board for all systems involved, namely, KnowledgePro and CLIPS in our example. Each individual system agrees to interact with a coordinator or scheduler, and read from and write on the publication board. In our implementation, the publication board is a ASCII file. Each node in hyper document has a unique name attached as its identity. The result of running expert system will be a name of node reached. This name is a short ASCII text which is a standard code and readable my most of machine. The KnowledgePro and CLIPS provide functions for each I/O access at run time. Therefore, write and read to or from both systems are easily supported. Since knowledge base of mining method selection system is implemented as an individual component out of hypermedia system, there is no need for expert system to access hypermedia database. So the only function here is that the CLIPS write the reached identity of hyper node to public board and hypermedia system read the identity from the board.

Figure 10-8a illustrates the dialog window when a user clicks the link '**Mining method**'. Two options are prompted. If a user is performing a training course without specific task in mind, he might want to read the basic mining techniques in general. The first option provides him with a link to intermediate node where the mining method specification is explained in general, and individual methods are further associated through generic links. For a experienced backfill designer, the second option provides an intelligent link which initiates mining selection expert for consultation (shown in figure 10-8b). The process of the consultation leads to a specific mining method which is published to the publication board from CLIPS. Then the coordinator informs KnowledgePro to read from the board for further process. Figure 10-8c presents the resulted mining methods in the context of hypermedia system. In the windows 386 Enhanced model, CLIPS and KnowledgePro cooperated seamlessly. This approach separates knowledge base of expert system from hypermedia database. We believe a more advanced architecture should integrate the knowledge base of the expert system within hypermedia database, therefore the operation on the expert system will be the same as operation on hypermedia with the same knowledge representation scheme. By doing so, the application of hypermedia system for education training and backfill operation simulation will be a natural extension of the current hypermedia presented.

The performance of the system is dramatically improved by indexing the name of a link in hypermedia document. The sequential access to the hypermedia, as initially designed, runs too slow when the searched nodes appear to the end of hypermedia document and results in a sensible time elapse after mouse clicking. The performance deteriorates when the searched hypermedia document grows. For a small to middle size hypermedia system, single indexing technique provides a simple solution. By doing so, the access to hypermedia database in our prototyping system runs approximately in a constant time to the search space. The search time can be ignored, and the sensible time elapse only depends on the hardware of the machine and the size of the node document.

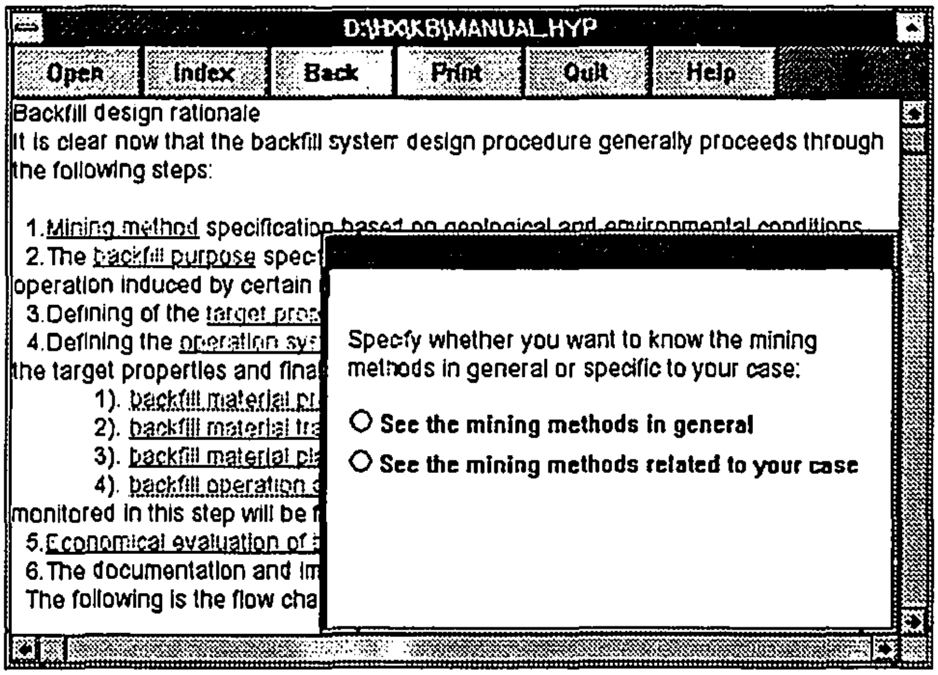



Figure 10-8a The dialog window of intelligent link 'Mining method'

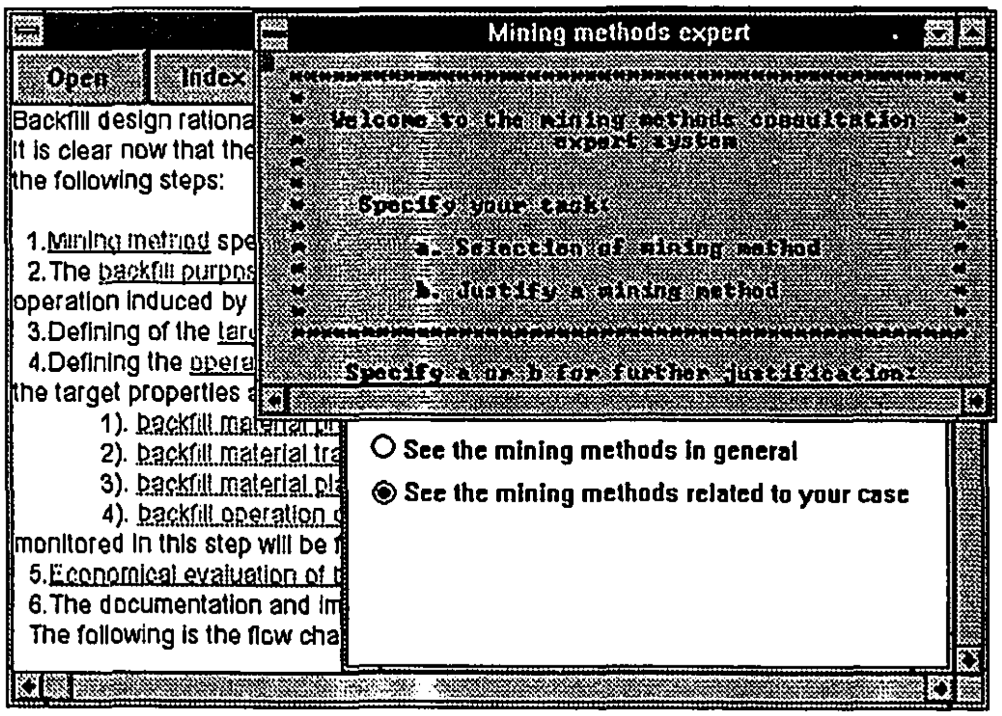



Figure 10-8b Consultation window with mining methods expert system

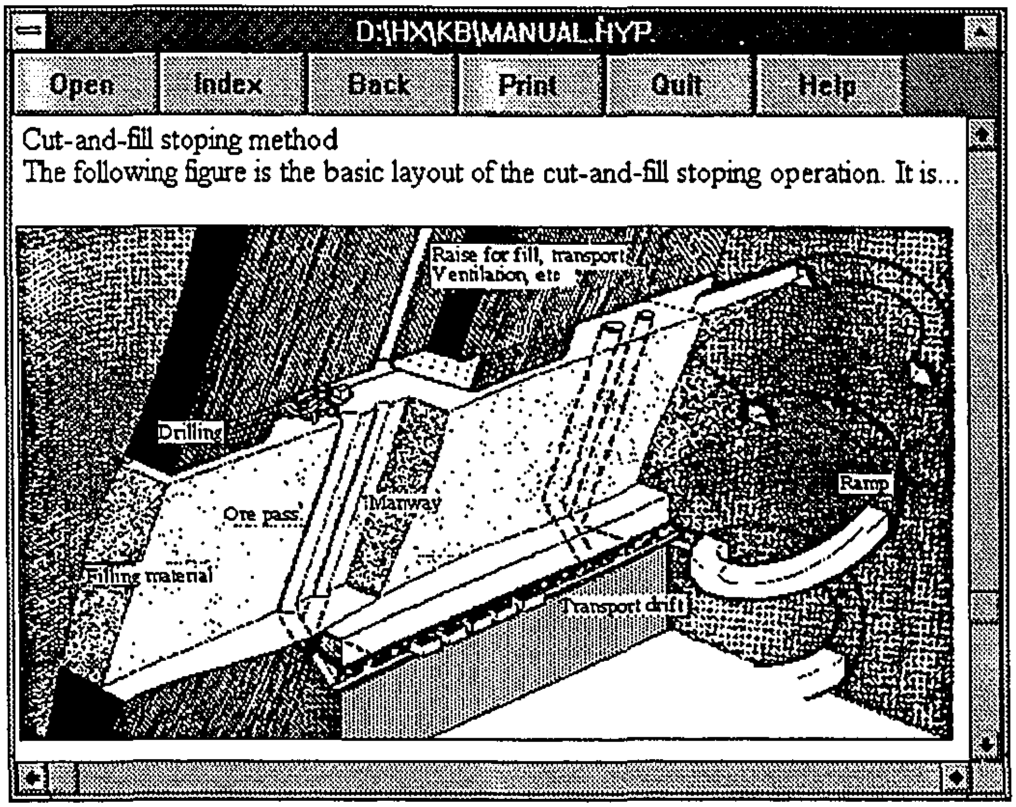



Figure 10-8c The recommended mining method through consultation

## 10.3. CONCLUSIONS AND FUTURE WORK

Hypermedia technology is providing us with capability to greatly increase the amount and kind of data we store electronically. The non-linear access to the hypermedia database gives rise to associative information retrieval, which coincides with cognitive intuition of human beings and many engineering design processes. We have presented the basic concepts and principles of hypermedia system and its feature. In particular, we argued that the backfill design is a multi-phase information intensive process. The hypermedia system provides a nice paradigm to meet with this kind of information requirement.

The prototyping illustrated provides a good example that combines various technologies together to support a dynamic linked hypermedia system within the framework of blackboard architecture. It is believed that the expert system techniques can be further combined with hypermedia system to generate more powerful decision supporting tools. The key issue is to merge the knowledge base of the expert system within hypermedia by a uniform representation. The initial development of prototyping hypermedia system for backfill design demonstrated its ability to provide a multi-source reference manual and decision supporting tool. The following are considered cases of further research for hypermedia system development in mining engineering:

- The development of high level graphic user interface to integrate various design tools into a single environment based on hypermedia technology. In essence, the hypermedia system does not cope only with information process, but also serves as user interface development tools. By doing so, the links will no longer be always linked to static node, but rather to other links or procedural calls to impose modularity.
- In the application level, one natural extension of the current prototyping system is to implement a tour mechanism, which will provide more sophisticated facilities to support tutorial features of hypermedia system for novice backfill designers and operators. A profound understanding of domain knowledge of backfill design is crucial in designing the touring chain.
- The hypermedia system also opens possibility to simulate backfill design process under the supervision of expert system. In this process, the backfill designer has opportunity to try different options, and the hypermedia system will cooperate dynamically to demonstrate possible consequences from various information source. This requires more complex searching strategies which we believe will be a role played by an expert system.

# CHAPTER 11

# KNOWLEDGE BASE MANAGEMENT SYSTEM FOR BACKFILL DESIGN

## 11.0. INTRODUCTION

As indicated in previous chapters, the traditional backfill mining operation design is a multi-level data/information/knowledge intensive procedure. A large amount of time and money is spent to bring a wide range of relevant data and knowledge together to support decision makings involved, from the specification of the mining method and backfill purpose to the identification of backfill material target properties, preparation, transportation, placement, quality control, and economic evaluation of the backfill venture as well.

The solution of most mining engineering problems requires, in addition to vast numerical calculations, substantial use of practical judgment and expertise based on experience[125]. Generally speaking, the feasibility study of certain project is based on the general rules. The optimization issues, on the other hand, relies on information and experience the designers have in mind. So it would be appreciated to have a computer system that can:

1) store knowledge of mine design, especially the previous experience of other successful designed mining operation;
2) intelligent process of the knowledge and information related to certain design problem. Based on the information the computer system provided, the designer can make decision more efficiently and precisely.

The chapter 10 discussed the hypermedia database system to manage the source of knowledge related backfill design in general. In hypermedia database system, the information or knowledge is stored and presented to users as passive data. The semantics of knowledge are general ignored. The processing of knowledge is not address. Users are responsible for decision making. Therefore, the hypermedia database system provides only partial solution of the integrated decision support system. It is believed that certain knowledge included in the backfill design rationale specified in the previous chapters can be processed to support automated decision making for certain problems. The mining method selection expert system, for example, provides an automated approach to mining method selection. This chapter presents another approach towards efficient feasibility study of mining operation based on the knowledge base management system (KBMS) technology. Special attention is given to the backfill mining operation design. The basic technology and concepts of knowledge base management system are discussed in the context of Logical Data Language (LDL) framework. In particular, the main issues of backfill design using LDL are illustrated by an example of hydraulic transportation system design of backfill operation. Finally, we concluded that the knowledge base management system is a useful tool to solve mine design problems, and recommend further extension of the technology to the full scope of the mining operation design.

## 11.1. BASIC CONCEPT OF KNOWLEDGE BASE SYSTEM

Solution to problems in knowledge/data intensive engineering application depends on the characteristics of these problems which can be classified according to three attributes: 1) completeness of knowledge of data in the environment; 2) accuracy of knowledge or data available in the environment; and 3) knowledge about the objective and or specifications of the problem[126]. Backfill design is well defined, but inexact and incomplete data process. Heuristics must first be applied to find a restricted set of data/knowledge that can be used in solving the problem. We propose the knowledge base management system approach for solve certain backfill design problems.

The knowledge base management system is a programming system that has the capacities of both database management system (DBMS) and declarative language to serve the role played by data manipulation language and host language[127]. Declarative language is a language in which one can express what one wants, without explaining exactly how the desired result is to be computed. A language that is not declarative is procedural such as Pascal, C, Lisp. Like the DBMS, the KBMS supports for efficient data access and manipulation. Above that, the KBMS provides the expressive power with declarative language based on logic as has been successfully demonstrated in expert system, production system and logical programming language. Figure 11-1 illustrates a primitive architecture of the KBMS.

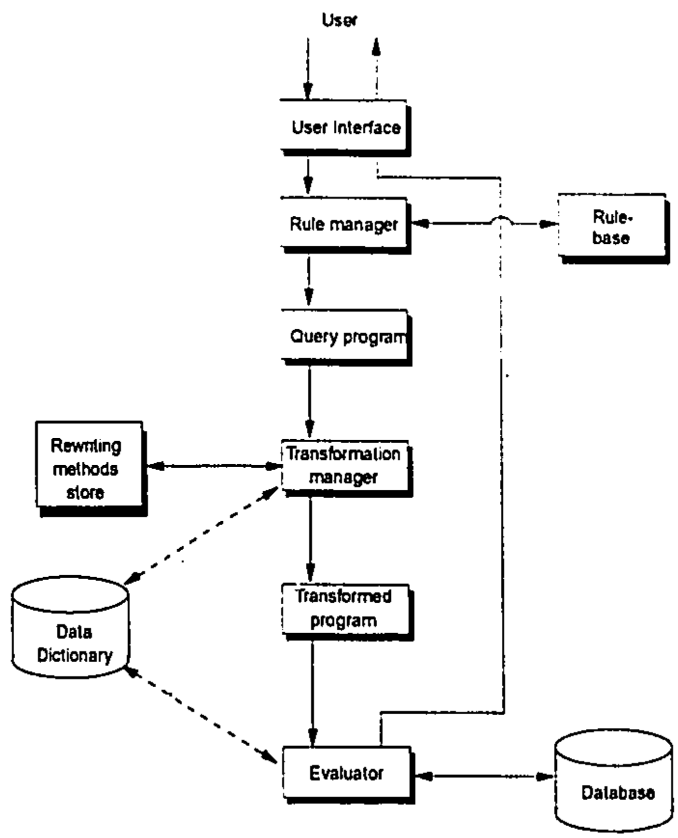



Figure 11-1 The Architecture of Knowledge Base Management System

The combination of the database and declarative language provides the useful power for real application beyond traditional DBMS and logic rules system alone. The traditional database technology, i.e. relational database for example, provides limited ability to address inference of reasonable complexity, transitive closure for instance. Therefore the semantics of knowledge of application domains can not be well understood and properly process to provide meaningful solution to end users. On the other hand, the logical system such as expert system does provide the power of complex inference, but fails to address the ability to manage and process data of massive beyond the working memory. In most expert system, domain knowledge has to be loaded to working memory as a whole, or as sequentially file. Efficient secondary storage which is well supported by database technology is not provided. The development of knowledge base management system is to address both capacity and therefore provide more powerful expressive language for real application. Over the last ten years, several models have been intensively investigated[128], [129], [130], [131] and concluded with a simple powerful language Datalog[132], [133] , which leads to the design and prototyping of LDL knowledge base management system.

The LDL system provides a declarative logic-based language and integrates relational database and logic programming technologies so as to support advanced data and knowledge-based application. The LDL system is to develop the technology for a new generation of database systems that support the rapid development of sophisticated applications--such as expert system and advanced scientific and engineering applications[134]. After about ten years of research and development, the LDL system has been developed as an efficient and portable SALAD, standing for System for Advanced Logical Application on Data, under UNIX system. Shown in figure 11-2 is the conceptual architecture of the current LDL. There are six basic components or modules in the current system: 1). the user interface, 2). the fact manager, 3). the schema manager, 4). the rule manager, 5). the query form manager and 6). query form manager. Since the underlying philosophy of LDL extends that of relational database systems, the facts are represented as relations. Above the facts, the rules are defined using Horn Clause with limited extensions to allow function symbol and negation as part of the language. Meanwhile, the newly released version of LDL provides convenience for linking with external functions and relation, which made possible the massive numerical calculation[135].

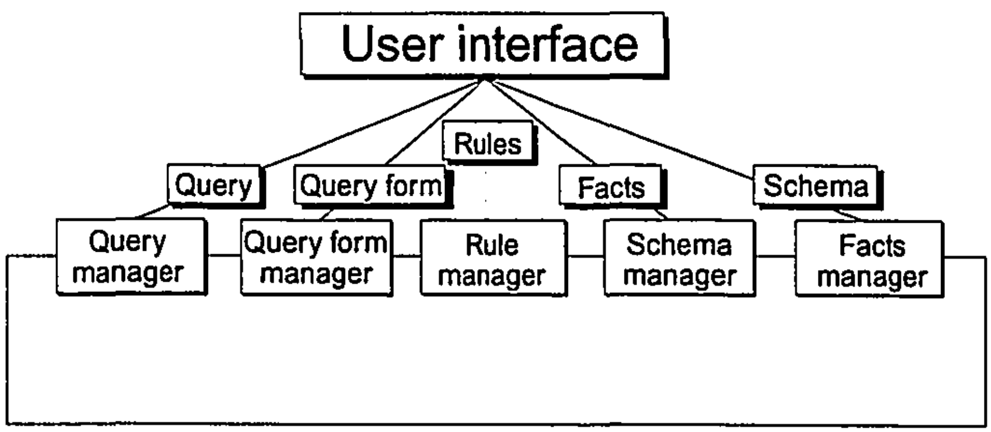



Figure 11-2 The conceptual architecture of LDL

The LDL application programs consist of four components:

1. A schema;
2. The data;
3. A rule set;
4. Query form;

Information about schema, facts, rules, and query forms is created and contained in ASCII text files by any text editor.

1. A schema is a description of the stored data and its organization into stored facts. The schema specifies, among other things, the format in which the data is stored and the type of each constant. The schema itself is stored in a *.sch file. Complex terms are also defined within the schema as shown later in the example.
2. The data consist of a set of facts. These facts must be consistent in structure and type with the relations declared in the schema and are stored in a *.fac file.
3. The rules component of the program contains a set of one or more rules that serve to define the derived predicates required for queries. Rules are stored in a *.rul file.
4. A query form is a generic query that specifies for the LDL compiler which of the arguments will be given and which are expected as output when a query of that type is issued. Query forms are stored in *.qf file.

## 11.2. KNOWLEDGE BASE SYSTEM FOR MINE DESIGN

Based on the power provided by LDL, certain mine design problems addressed in previous chapters can be formalized to be solved with the assistance of knowledge base management system. We now present a formal representation of hydraulic transportation system design. The methodology and implementation is illustrated by the example of pump selection, which is a crucial decision faced to backfill designers.

### 11.2.1. The general description of design problems

As an example, the hydraulic transportation system is taken for close analysis and implementation. The figure 11-3 illustrates basic specifications and information flow concerned. One key issue of hydraulic transportation system is the pump selection. As with most mining engineering problems, the backfill design requires substantial use of practical judgment and expertise based on previous experience. Successfully designed mining operation are used as a source of good example, and certain decisions are made simply by drawing the comparison from case to case. Since the conditions inherited for different deposits are never identical, an apparent drawback of this method is the burden of collecting the related mining operation for comparison. Even though the general information is available, close review of information and case by case analysis manually is also an expensive process. Therefor, the knowledge base management system approach provides the technology for efficient information storage and logic reasoning, and can be used as an automated or semi-automated tool for decision making involved in backfill design. The following example uses LDL as a tool to implement a knowledge-based application requiring efficient access to large collection of data related to hydraulic transportation system design in mining operation which includes the following goals:

1. Provide quick access to the similar mining operation for the user.
2. Based on certain domain knowledge, the system can recommend some design options for users to verify, which are: 1). The pressure requirement justification; 2). Pump recommendation; 3). Economical overview of the mining project.
3. Provide certain procedural function call to access traditional language like C for numerical analysis which is also importation towards the success of the mine design.

In particular, the implementation focuses on the issue of pump selection which is crucial for hydraulic transportation system. The key problem in determining the proper pump relies on the prediction of pressure loss along the pipe line when fill material is delivered. It is still an active research topic in hydraulic transportation. Several mathematical models have been proposed with restrictions and a loop test is almost inevitable. On the other hand, the practical operation of pump is far from optimized. Previous experience may also leads to malfunction of pump selection. To overcome the unreliability of the pressure loss prediction method mentioned above, this example proposed a multi-phase pressure loss prediction model based on both analytical (empirical) pressure loss prediction model available (as discussed in chapter 4) and statistics model of previous applications.

A statistics model of pressure loss is defined as the average pressure loss used in similar previous hydraulic transportation system. Suppose that the information of previous hydraulic transportation operation is available and stored in knowledge base management system, the pump pressures used previously can be retrieved from database to calculate the average pressure. The pressure losses predicted from the two models may not agree each other. If conflict is encountered, a compromised pressure loss should be recommended. The details of the rules to define these domain knowledge will be given later.

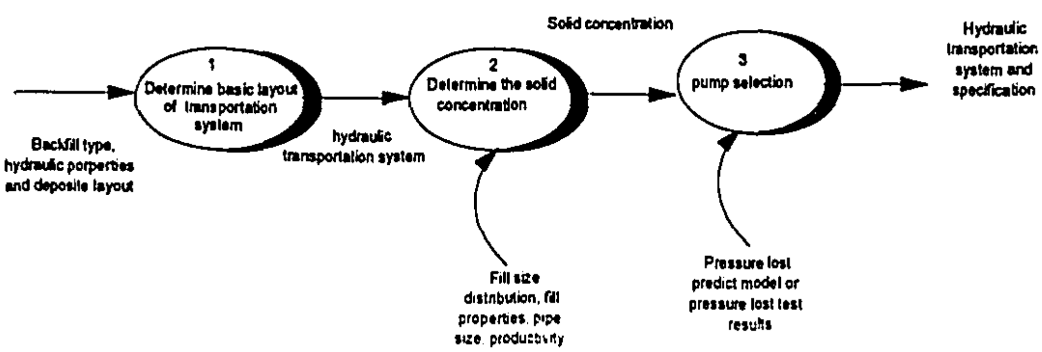



Figure 11-3 The information flow of hydraulic transportation system design

### 11.2.2. Data representation

Relational representation is essential in LDL. In the context of relational representation, information is represented in terms of relations which consists of various attributes. Each relation groups certain information closely related together to represent a real world object. In chapter 8, we discussed in brief the conceptual modeling and presented the entity-relationship diagram as the global picture to capture the major information needs conceptually. We now focus more specifically on the local information need of hydraulic transportation system. The result of the modeling leads to a logical representation of information as relations. Bearing in mind, this modeling is intended to address the information need only for feasibility study. Therefore data of interesting may not be sufficient to capture the other issues of backfill design such as the detailed specification and evaluation of backfill operation.

Following the LDL convention, relations and attributes have a name. A relation name is represented as lower case string and attribute name is represented with strings starting with up case letter. The formal representation of relation is:

**relation**(Attribute1, Attribute2, Attribute3,,,AttributeN)

Where relation is any name assigned to relation, and Attribute1, 2, ,,,N is the name of attributes. Each attribute belong to certain type of data such as real, integer, string etc.. Normally, Relations and attributes are named with self explanation. For example, the mine relation stands for a relation to hold the information of mine, which includes attributes such as "MineID" for mine identification, "Production" for annual production of the mine etc.. A function symbol is presented as up case string by LDL. Following the name of a function symbol is the arguments related to the function. The following information has been identified to be pertinent to hydraulic transportation system design. In this specification, the relation is defined by a short description and a formal format. Following the definition is a table specifying the type of each attribute.

**1. pump relation:** relation pump is defined to represent the information related to the pump of the real world entity.

**pump**(Pumpid, Pumptype, Purpose, Capacity, Pressure, Manufacturer, Cost);

| Attributes | Domain |
|---|---|
| 1). Pumpid | . {string} |
| 2). Pumptype | {reciprocating, centrifugal) |
| 3). Purpose | {string: water, slurry} |
| 4). Capacity | {real: capacity of pump} |
| 5). Pressure | {real: pressure of pump} |
| 6). Manufacturer | {string: manufacturer ID} |
| 7). Cost | {real: cost of pump} |

2. manufacturer: relation manufacturer is defined to represent the information related to the manufacturer of pump and other equipment.

**manufacturer**(Name, ADDRESS(City, State, Country, Telephone));

| Attributes | Domain |
|---|---|
| 1). Name | {string: name of manufacturer) |
| 2). Address(City, State, Country, Telephone) | {Function symbol} |

3, transport: relation transport is defined to represent the information related to hydraulic transportation system. It captures the operation feature of hydraulic transportation system rather than its structure.

**transport**(Tranid, Mineid, Type, Cap, LHratio, Height, Fillid, Dia, Flow, Pumpid, Cost);

| Attributes | Domain |
|---|---|
| 1). Tranid | {string: transportation system identification} |
| 2). Mineid | {string: mine identification} |
| 3). Type | {string: Hydraulic, pneumatic, conveyer, truck} |
| 4). Capacity | {real: capacity of transportation system} |
| 5). LHratio | {real: ratio of total length to vertical length} |
| 6). Height | {real: vertical difference of transportation system} |
| 7). Fillid | {string: identity of backfill material} |
| 8). Dia | {real: pipe diameter} |
| 9). Flow | {real: flow velocity} |
| 10). Pumpid | {string: identity of pump} |
| 11). Cost | {real: cost of overall transportation operation} |

4. fill: relation fill is defined to represent the information related to certain fill material. It is intended to capture the information related to hydraulic transportation rather the mechanical properties.

**fill**(Fillid, Filltype, Concent., Sgrav, Fgrav, Max, Avesize, Cratio, Cvelocity, Cost);

| Attributes | Domain |
|---|---|
| 1). Fillid | {string: to identify backfill material} |
| 2). Filltype | {string: Tailing, Sand, Rock} |
| 3). Concent | {real: backfill concentration} |
| 4). Sgrav. | {real: slurry gravity} |
| 5). Fgrav. | {real: dry material gravity} |
| 6). Max | {real: Maximum size of particle} |
| 7). Avesize | {real: average size of fill material} |
| 8). Cratio | (real: cement ratio; 0.0-1.0} |
| 9). Cvelocity | {real: critical velocity} |
| 10). Cost | {real: cost of fill material per ton} |

5. mine: relation mine is defined to represent the information related to mine operation which captures the operation feature of a mine.

**mine**(Mineid, Production, Mineral, Method, Purpose, Rockcondition, Cost, Address);

| Attributes | Domain |
|---|---|
| 1). Mineid | {string: to identify a mine} |
| 2). Production | {real: annual production of a mine} |
| 3). Mineral | {string: name of the major mineral} |
| 4). Method | {string: mining method} |
| 5). Purpose | {string: Backfill purpose} |
| 6). Rockcondition | {real: very stable, stable, good, poor, very poor} |
| 7). Cost | {real: cost of ore per tone} |
| 8). ADDRESS | {Function symbol} |

6. **modelfill:** relation modelfill is defined to represent information related to classification fill material. In essence, the fill material is classified by its pressure loss prediction model. The attribute Mfillid specifies the model that fill material belongs to. The rest attribute specifies the range that qualify each fill material to certain model.

**modelfill**(Mfillid, Maxconcent, Minconcent, Maxsize, Minsize, Maxcratio, Mincratio);

| Attributes | Domain |
|---|---|
| 1). MFillID | {string: to identify backfill material model} |
| 2). Maxconcent | {real: Maximum solid concentration of slurry} |
| 3). Minconcent | {real: minimum solid concentration of slurry} |
| 4). Maxsize | {real: maximum size of particle} |
| 5). Minsize | {real: minimum size of particle} |
| 6). Maxcratio | {real: maximum cement ratio, 10.0-1.0 % } |
| 7). Mincratio | (real: minimum cement ratio; 0.0-1.0%} |

7. **costmodel:** relation costmodel represents the information of cost model of real world. In chapter 7, we presented a regional cost prediction model. This relation captures the information needed to perform the evaluation based on that model. The attribute Rank stands for the rank classified for certain mining operation. It could be very good, good, acceptable, unacceptable. The rest of attributes stands for certain ranges that qualify certain backfill operation to a rank.

**costmodel**(Rank, Maslurry, Mislurry, Maxtran, Mintran, Masum, Minsum, Filltype);

| Attributes | Domain |
|---|---|
| 1). Rank | {string: verygood, good, acceptable, poor, unacceptable} |
| 2). Maslurry | {real: Maximum cost of slurry} |
| 3). Mislurry | {real: minimum cost of slurry} |
| 4). Maxtran | {real: maximum cost of slurry transportation} |
| 5). Mintran | {real: minimum cost of slurry transportation} |
| 6). Masum | {real: maximum sum of backfill cost } |
| 7). Mincratio | (real: minimum sum of backfill cost} |
| 7). Filltype | (string: fill type} |

8. **ADDRESS**: function symbol ADDRESS is defined to represent the address of any organization. As an extension of Datalog, LDL allows using function symbol as a term constructor of complex term. It is not a relation. So the operation on ADDRESS will be different from the relations.

**ADDRESS**(City: string, State: string, Country: string, Telephone: string)

| Attributes | Domain |
|---|---|
| 1). City | {string: name of city} |
| 2). State | {real: name of state} |
| 3). Country | {string: name of country} |
| 4). Telephone | {string: telephone number represented as string} |

### 11.2.3. Definition of rules

Based on the relations defined, rule bases are defined by Horne Clause to capture the logic feature of information need. The details of the rule implementation is accessible through the rule file "mine.rul". In addition to the basic LDL facilities, some advanced techniques are also used which will become clear in the following discussions.

**1. Definition of compatible mines**

To be able to draw the comparison among different mines, three levels of similarity have to be defined:

1). Very similar mine,
2). Similar mine, and
3).Compatible mine.

The basic rules to define the three levels similarity is:

A. If mine MINE1 and MINE has 5% percent difference in production and produce the same mineral using the mining method falling into same category under same rock conditions, then the two mines are considered be to very similar mines;
B. If mine MINE1 and Z are very similar mines, and mine Z and MINE2 are very similar mines, then MINE1 and MINE2 are considered similar mines;
C. If mine MINE1 and Z are very similar mines and mine Z and MINE2 are similar mines, then the MINE1 and MINE2 are considered to be compatible mines.

These rules can be represented as the following Horn Clause by LDL syntax such as:

1). Predicate VerySimilarMine(MINE1, MINE2) means that the mine MINE1 and MINE2 are very similar mines. Above the <- sign is the rule head meaning the conclusion, the rest is the rule body meaning the conjunctive condition that make the head true.

```
VerySimilarMine(MINE1, MINE2)
        <-
mine(MINE1, Prod1, Min1,Method1,_, Rock1),
mine(MINE2, Prod2, Min2, Method2,_, Rock2),
abs(Prod1, Prod2, Pdaviation),
Pdaviation<=0.05, Pdaviation>=0.0,
Method1=Method2, Min1=Min2, Rock1=Rock2.
MINE1~=MINE2.
```

where the ~ sign negates the built-in predicate =, means not equal. The underscore refers to the arguments not concerned by the rule.

2). Predicate SimilarMine(MINE1, MINE2) means that mine MINE1 and MINE2 are similar, which can be defined as the following rule:

```
SimilarMine(MINE1,MINE2)
        <-
mine( Mine2, _, _, _, _, _, _)
if (VerySimilarMine(MINE1,MINE2) then true
        else SimilarMine(MINE1,MINE2),
         VerySimilarMine(MINE1, Z),
         MINE1~= MINE2).
```

3). Predicate CompatibleMine(MINE1,MINE2) means that mine MINE1 and MINE2 are comparable in the sense that they can be compared each other to draw conclusion for feasibility study, which can be defined as following:

```
CompatibleMine(MINE1,MINE2)
        <-
mine(MINE2, _, _, _, _, _, _),
if (verysimilarmine(MINE1, MINE2) then true
else if (Similarmine((MINE1, Z) then true
else verysimilarmine(MINE1, Z),
similarmine(Z, MINE2), MINE1~=MINE2)).
```

In defining the CompatibleMine, an advanced IF-THEN-ELSE operator provided by LDL is used to express certain procedural constructs. External call "abs" to calculate the relative difference of two attributes such as production is implemented as external function imported to the LDL, which is used as a built-in predicate.

The figure 11-4 illustrates the graphic representation of the relationship defined by the rules:

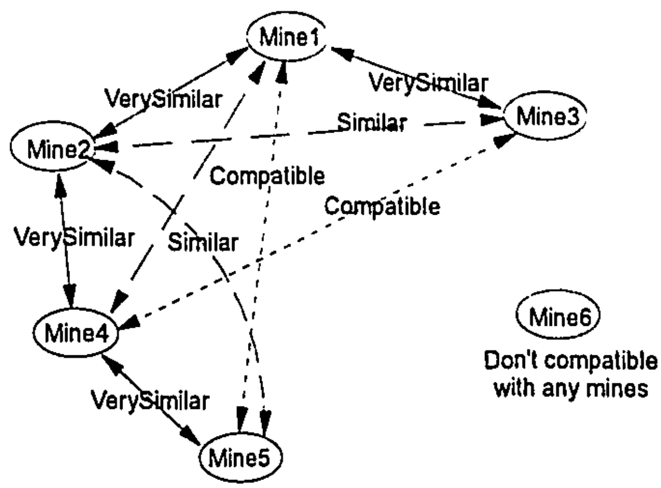



Figure 11-4 Graphic representation of relationship between mines defined by rules

**2. Definition of the similarity of Backfill material**

Two levels of similarity are defined by the following rules:

A. If fill FILL1 and FILL2 have 5% percent difference in concentration, fine particle, gravity and belong to the same type of backfill, then the two fill materials are considered be to be similar;

B. If fill FILL1 and Z are similar, and Z and FILL2 are similar, then FILL1 and FILL2 are considered to be compatible;

Figure 11-5 is the graphic representation of relationship between fill material defined by above rules.

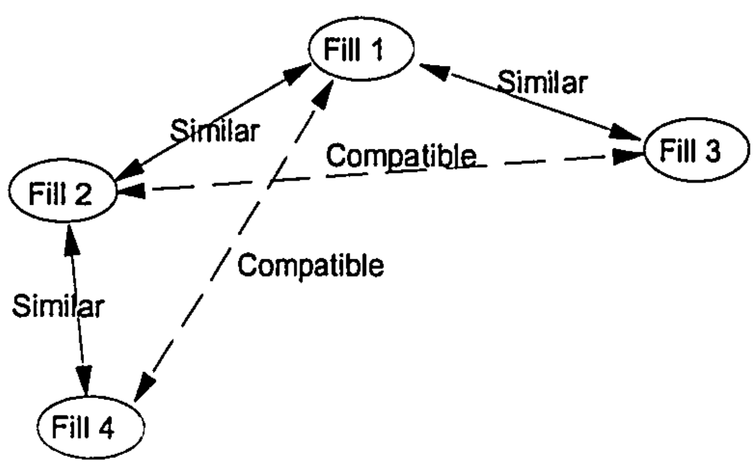



Figure 11-5 Graphic representation of relationship between fills defined by rules

**3. Definition of compatible transportation system**

Three levels of similarity of transportation systems need to be defined based on the following rules:

A. If transportation system TRANS1 and TRANS2 has 5% percent difference in capacity, L/H ratio (the ratio of total length to height of pipe line), and pipe diameter regardless of the fill material handled, then the two systems are considered to be relevant.

B. If system TRANS1 and TRANS2 are relevant and designed for compatible mine to handle the compatible fill material, then the two transportation systems are considered to be similar. Notice that in this rule, the two transportation systems are considered to be similar provided that the related mining operation and fill material have to be compatible, which take into account both the mining operation and fill material as a whole.

C. If the system TRANS1 and Z are similar, and Z and TRANS2 are similar, then TRANS1 and TRANS2 are considered to be compatible transportation system;

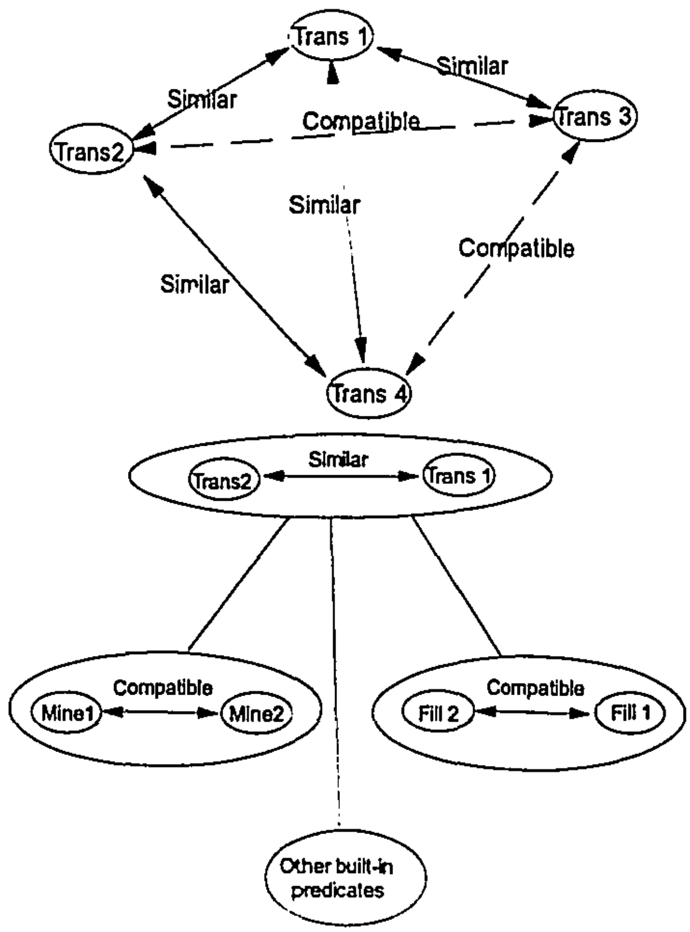



Figure 11-6 Graphic representation of similar transportation system defined by rules

## 4. Definition of pressure loss prediction model

Every fill material belongs to a category based on certain pressure loss prediction model. The rules to define the material category is the following:

A. If fill FILL1 and model fill of category MFill has 5% percent difference in concentration, size range, and cement ratio with the same fill type, then FILL1 are considered belonging to MFill category with same pressure loss prediction model PM;

B. If fill FILL1 and Z are similar fill material with the same backfill type, and Z belongs to MFill category and , then FILL1 belongs to MFill as well.

Figure 11-7 is the graphic representation of the relationship between the fill and fill model defined by the rules.

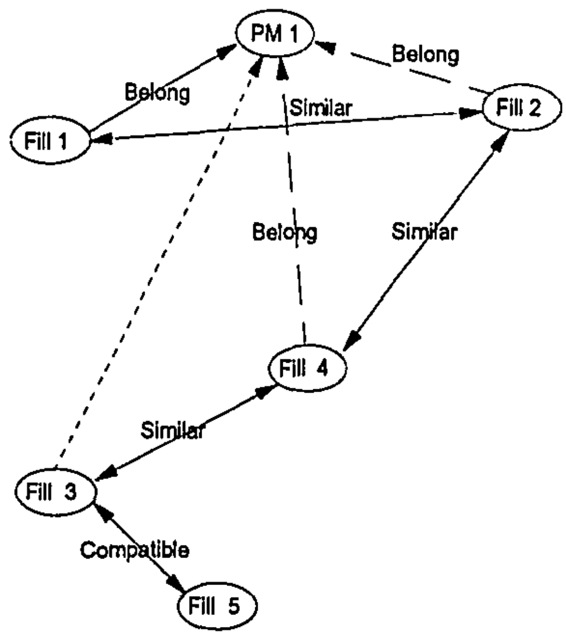
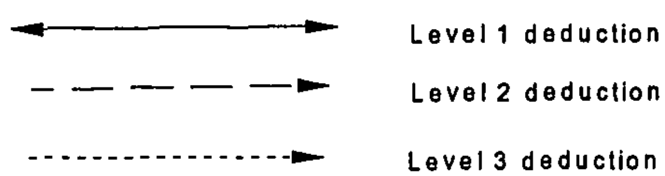



Figure 11-7 Graphic representation of belong relationship




It is noticed that the above rules defining the fill model involve the recursive definition which is not reliable to capture the similarity of the fill material theoretically. If the inference chain is too long, the similarity defined by first rule will not exist any more. But in practice, the fill material do fall naturally into certain category with causal exception. Most of the fill models inferred by the above rules will closely meet with the characters defined by certain model. Meanwhile, the pressure calculation followed will be justified by a statistical model, which will certainly eliminate the influence of the unreliably inferred factors by the rules. The techniques involved in these rules in case 2, 3 and 4 are similar to the case 1.

**5. Definition of pressure loss rule**

As mentioned earlier, the pressure requirement of a hydraulic system can be estimated from two different ways, 1). numerical calculation, but it is not always valid, 2). previous experience, but it is not always reliable. Our rules will generate the pressure requirement from both methods and verify one with another.

1). In this case, external predicate is used in the program. In LDL, external predicate can be any kind of procedural program implemented by conventional language such as C, Fortran, etc.. In our case, the external predicate is initially implemented as functions to calculate the pressure loss based on analytical model. These functions are stand alone programs and executable under any operating systems. Once imported to LDL, it is understood as an external predicate in the context of Datalog language. In doing this, the procedural programming is seamlessly merged into logical programming, therefore extended to the power of symbolic calculation of LDL system to handle complex numerical calculation. The following is an example:

```
pressure(Transid, P)
        <-
transport(Transid, _, _, LHratio, Height, _, Fillid, Dia, Vel, _),
fill(Fillid, _,Concent, Sgrave, Fgrave,_,_,_,_,_),
belong(Fillid, Mfill),
Mfill=p1,
pp1(Concent, Vel, Fgrave, Dia, Sgrave, LHratio, Height, P).
```

In this rule, the pressure requirement of transportation system "Transid" is specified by P which is the results of calling an external predicate pp1.

2), Similar to case one, some aggregation operations are implemented as external predicates to calculate the pressure requirement based on the previous experience as the following rule:

```
relevantpressure(Transid, P)
        <-
transportation(Trans2, _,_,_,_,_,_,_,Pumpid2),
compatibletransystem(Transid, Trans2),
pump(Pumpid2, _,_,_,P,_,_).
```

which generate all pressure requirements specified in the compatible transportation system. Because the LDL does not provide the aggregation function to calculate the average, maximum, minimum of the P set directly, the P set is output to a file called "inp" through the input/output facilities provided by the LDL. Then the following rules can use the intermediate information to generate the needed calculation through another external function call aggregation(inp, avg, max, min).

**6. Definition of the qualifiedPump rule**

In principle, the pump selected for the system should satisfy with the following conditions:

1). The pressure of the selected pump must be greater than the pressure loss predicted;

2). The capacity of the pump should be greater than the capacity of the system;

3). The cost should be as low as possible;

4). The post sales service for reciprocating pump counts for about 5% of the total cost;

5). When slurry concentration is greater than 75%, reciprocal pump is the only choice;

Based on these requirements, the pump selection rule could be expressed by rules with some external call for numerical calculation which turns out to be very efficient and powerful. The following is one of a group rules defined to describe the qualifiedpumps. By querying on the database through the rules, several pumps might be presented as the candidates. The users are allowed to make choice based on the recommendation.

```
Qualifiedpump(Trans1, PumpID)

        <-

pump(PumpID, Type, _, Capacity, Pressure, _, _),
transportation(Trans2, _ ,Capacity2, _, _, _, _,_,_, PumpID2),
transportation(Trans1, _ ,_, _, _, _, Fill1,_,_, _),
pressure(Trans1, Mp),
Compatibletransystem (Trans1,Trans2),
aggregation(inp, Pavg, Pmax, Pmin),
Pmin<Mp,
Mp<Pavg,
Pumptype='centrifugal',
MP=<pressure,
pressure>=1.05*MP,
fill(Fill1, _. Concent, _,_, _, _, _),
Concent<=0.75;
```

Because this rule involves the external function call aggregation(inp, Pavg, Pmax, Pmin) which depends on the intermediate data file 'inp', the query to qualifiedpump has to be performed by two steps: 1) generate 'inp' file, 2) call aggregation external function. The two step query is completed by LDL scripts and piping techniques using redirection command. An LDL script file is an ASCII text file containing a collection of SALAD commands, so several queries can be executed together to generate expected results. Following is an example used to perform the aggregation(inp, Pavg, Pmax, Pmin) external call. First, a script file pressure.cmd has to be created as the following:

```
?relevantpressure(tran1,P)>mining.ldl/mining.cf.0/inp
aggregation(inp, Pavg, Pmax, Pmin)
exit
no
```

Then execute the file using shell command salad<pressure.cmd. The calculation will be performed, and the results displayed on the screen. All the external function and predicate are imported to the LDL using the 'import' command provided by LDL.

**7. Definition of cost estimation rules**

Because of the diversity of the mining operation, it is not feasible at the present time to adopt any universal cost model for backfill operation. Therefore a regional cost model is possible to build up based on statistical analysis. A provincial wide survey of backfill operation has been conducted within Quebec area and a Quebec regional backfill operation cost model has been established. In this model, the main contributions to the backfill cost comes from fill cost and transportation cost, which takes about 85-90% of the total backfill cost. Accordingly, a backfill operation can be modeled by average cost level in terms of statistics. Rules to estimate the cost levels are simply defined based on the average cost level within the area. An operation can be estimated as a very good, good, acceptable, poor and unacceptable.

**8. User interface**

Above all of the discussions mentioned is the user interface, which is implemented as a stand alone facility to access the queries defined. Figure 11-8 illustrates the basic functions implemented. The interface is menu-driven facilitated with help options.

The main functions of the system are described as the following:

**Review**: the option 'Review' at the top level provides access to database to review the previous applications from various perspectives. Users can access the previous mining operation, transportation operation, characteristics of fill material as initially stored in database. In addition, the query can access rules base first and retrieve the data based on logic rule as defined. For example, suppose user wants to review other backfill operation closely related to his case, the submenu under the option 'Mine' provides a 'very similar mine' function to access the so defined mining operation. User may not be able to retrieve enough information of closely related mines. Then he may loose the condition and try to look at the similar cases. As stated earlier, three levels of similarity have been defined by rule base. In the least favorable case, user can refer to compatible mines for solution. In this process, users specify the requests, and the inference and computation will take place automatically by system.

**Evaluation**: the option 'Evaluation' at top level menu The option 'Evaluation' provides an automated tools to generate a cost report and the technical adequacy of the pump operation if any. It serves as an entry to submenu to evaluate hydraulic transportation system in terms of pump operation and overall economical performance. The option 'cost evaluation' provides function to evaluate the transportation operation based on a regional model. In this case it is a Quebec provincial regional model. The relation 'costmodel' and cost estimation rules defined earlier will qualify certain hydraulic transportation system to certain category as verygood, good, acceptable, poor and unacceptable. The option 'pump evaluation' evaluate the pump performance based on both analytical model and statistic model of pump application for certain transportation system. The results of the evaluation are all qualified pump for certain operation. If the actual operating pump of a transportation system is included in the recommended list, then the pump selected for that system is considered reasonable. Otherwise further investigation is recommended.

**Design**: Finally, the option 'Design' provides several data entry forms for specification. Under the option 'design', backfill designers have chance to access the 'review' menu to retrieve information whenever needed. The major function implemented in this prototype system is the pump selection. The user inputs the basic information of mine site, fill material, layout of transportation line as guided by the menu. At certain point, users have to determine whether a pump is needed to keep the system work safely, and if needed, what is the best choice. He should either specify his answer if he is certain, or leave it for system to make choice based on inference rules defined. He is even allowed to specify randomly. The 'evaluation' option followed will check whether the user specification is acceptable and recommend the best possible according to the knowledge of underlying system.

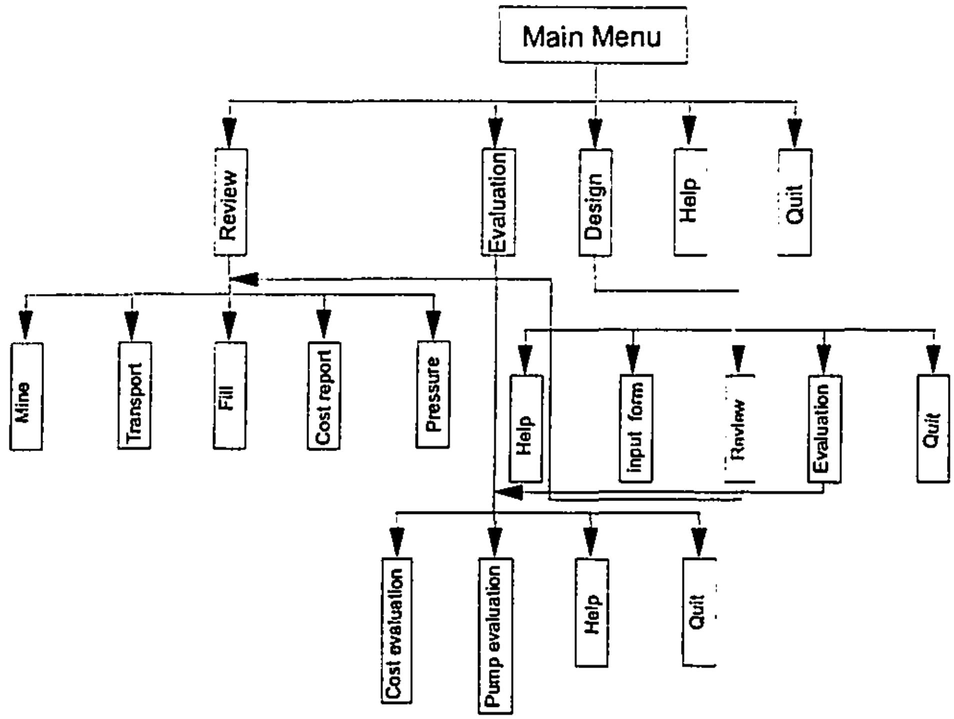



Figure 11-8 The basic functions of the system

## 11.3. SYSTEM TEST AND SAMPLE CASE STUDY

Since the LDL is implemented for UNIX operating system, the prototype of hydraulic transportation design system works only on UNIX environment. As stated in chapter 8, the knowledge base management system is initially defined as a component of overall integrated decision support system working for PC-based MS-Window environment. But knowledge base management system is not available on PC-based system at the present time. In order to be able to integrate it to the overall system, further studies are needed. This prototype is intended to illustrate the technical opportunity of using knowledge base management system for mine design. We believe, in the near future, the knowledge base management system will be comfortably shifted to PS-based system. For the time being, the sample system works as a stand-alone program.

### 11.3.1. System test

To test the usefulness of the system, sample data has been constructed. Based on the Quebec mining operation survey and literature review[136], sample data of 18 mining operations have been used for system testing. The original data do not provide all information required by the conceptual model of hydraulic transportation. For example,

most backfill operations in Quebec do not need pump in the transportation system. So the horizontal extension is extended proportionally so that the pump is apparently needed and the evaluation of pump operation is necessary. Based on the survey, more data are constructed artificially to draw correlation among different mining operations.
The following are an example of relational representation of a backfill operation:

mine(mine-1, 450000.00, nickel, cyclic, support, poor, 7.28988, address(sudbury, ontario, canada, 'Phone: (705) 674-3464')).

fill(fill-1, tailing, 65.0000, 1.61000, 2.1600, 0.0800, 0.1200, 0.125000, 1.6700, 3.1600).

modelfill(p1, 75.000, 0.000, 0.0100, 0.00, 100.00, 0.00).

transportation(trans-1, mine-1, hydraulic, 56,0000, 8.89998, 250.0000, fill-1, 76.0000, 6.70990, pump-7, 1.500).

costmodel(verygood, 64.000, 0.000, 24.5000, 0.000, 88.500, 0.000, rock).

pump(pump-7, centrifugal, slurry, 6360.00, 1608.00, 'groulds-pump', 2000.00)
These relation instances contain complete information of certain mining operation according to our conceptual model. Each mine has a unique identity, in this case, it is mine-1. It could be any name of real mining operation. The capacity of the mining operation is 450000.000 tone per year. The major mineral is nickel. The ore is mined using cyclic backfill for ground support because of poor rock condition. The mine is located in sudbury, ontario, canada, and phone number to reach is (705) 674-3464). This relation contains the general information of a mine. More specifically, the mine-1 is using fill-1 material to fulfill backfill operation. The characteristics of the fill material is contained in the relation 'fill', which includes the fill type, slurry concentration, slurry gravity, dry fill material gravity, maximum article size, percentage of fine material, cement ratio, critical flow velocity, cost of the slurry. The fill-1 is the identity of the material which can be any name used in real operation. For each backfill operation, there has to be a relation to contain the information of transportation system. In this case, it is trans-1, again the identity of the transportation system (could be any real name), it is applied to mine-1, the type of the system is hydraulic transportation to transport the backfill material fill-1. The pump used in the system is pump-7. Other data can be interpreted according to the definition of the relation discussed earlier. The pump

information is contained in the relation 'pump'. Each pump has a identity, such as pump-7 in this case, followed by pump type, it could be centrifugal or reciprocate. This relation also contains other basic information provided by manufacturer. Another relation 'modelfill' contains the information to define a pressure loss prediction model for certain fill material. p1 is the identity of a pressure loss prediction model. Other data items are ranges of certain criteria. Finally, the relation 'costmodel' contains the information to qualify a backfill operation to certain category such as very good, good, acceptable, unacceptable, poor based on the statistics of average backfill operation in certain local area or global operation. 18 backfill operations have been stored in the knowledge base as previous backfill experience. According to the functions provided by user interface, the user can review the previous backfill operation from different point of view.

Suppose a designer is interested in all similar backfill operation in general, he could do a query to the knowledge base. The system will group all mining operation according to the rules defining the similarity of mining operation. In our sample system, if a query

?verysimilarmine(A, B)

is posted to the knowledge base, the system will respond with a group of solutions:

group 1:
verysimilarmine(mine-1, mine-2)
verysimilarmine(mine-1, mine-14)
verysimilarmine(mine-1, mine-16)
group 2:
verysimilarmine(mine-2, mine-1)
verysimilarmine(mine-2, mine-3)
group 3:
verysimilarmine(mine-3, mine-2)
verysimilarmine(mine-3, mine-6)
verysimilarmine(mine-3, mine-9)

.
.
.

This query will provide information of very similar backfill operation in general. In reviewing of the very similar cases, a mine designer might be able to make decision for his own case according to the query. However, in most cases, the very similar cases hardly exist, designer might have to give up certain restriction and looking for similar or just compatible cases to help make decision. According to the rules defining the similar or compatible backfill operation, the similar case is the transitive of very similar case and compatible case is the another level transitive of similar case. This means that if A and B are very similar, B and C are very similar, then A and C are considered similar. If A and B are similar and B and C are similar, then A and C are considered compatible. Of course, the compatible cases include all very similar and similar cases. If designer do the query:

?similarmine(A, B)

the system will display the following solutions:

group 1:

similarmine(mine-1, mine-2)
similarmine(mine-1, mine-14)
similarmine(mine-1, mine-16)
similarmine(mine-1, mine-5)

group 2:

similarmine(mine-2, mine-1)
similarmine(mine-2, mine-3)
similarmine(mine-2, mine-6)
similarmine(mine-2, mine-9)

.
.
.

The solutions of the query: ?compatiblemine(A, B), is the following:

compatiblemine(mine-1, mine-2)
compatiblemine(mine-1, mine-14)
compatiblemine(mine-1, mine-16)
compatiblemine(mine-1, mine-5)
compatiblemine(mine-1, mine-3)
compatiblemine(mine-1, mine-6)
compatiblemine(mine-1, mine-9)

group 2:

.
.
.

group 3:

.
.
.

As expected, when the designer relaxes the similarity requirement, the knowledge base will provide more information for each group. Queries for other information, such as compatible fill material, compatible transportation system, demonstrate the similar inference capacity to provide information based on users' requirements.

The experimental tests displayed sensible results, and all outcomes make perfect sense as expected. The pump recommended by the system closely agreed with the manual prediction in 86% cases. The remaining 14% operation can not be justified by the system because of the incompleteness of knowledge. The basic objectives of the project were fully achieved.

### 11.3.2. Sample case study of hydraulic transportation design

As an example, we present the following sessions as a case study to demonstrate how knowledge base management system can help the decision making involved in hydraulic transportation system design. The most fundamental decision to be made when designing a hydraulic transportation system is whether extra energy is needed, if needed, how much is needed to keep the system under safe, stable operation. More specifically, the question is whether a pump is needed and what is the best possible pump in term of pressure supply and costs.

Using the last example mine-1 as basic data, we now demonstrate the design procedure. First of all, the designer should open the 'Design' menu from user interface. The menu consists of the following data entry forms:

```
************************************************************
*                                                          *
*               1. Mine site data input                    *
*                                                          *
*               2. Fill Characteristic input               *
*                                                          *
*               3. Transportation setup input              *
*                                                          *
*               4. Help                                    *
*                                                          *
*      Please specify the menu(1-4) or specify Ecs to escape: *
************************************************************
```

The user should input the mine site data first and review the similar mining operations. Select menu 1 will open a data entry form from where user input the mine identification, production, main mineral...etc. After input the mine site data, user can query the knowledge base system from review menu:

```
**********************************************************
*                                                        *
*            1. Mines                                    *
*                                                        *
*            2. Very Similar Mines                       *
*                                                        *
*            3. Similar Mines                            *
*                                                        *
*            4. Compatible Mines                         *
*                                                        *
*      Please specify the menu(1-4) or press Ecs to escape:  *
*                                                        *
**********************************************************
```

Do the query Verysimilarmine(mine-1, A), Similarmine(mine-1, A), Compatiblemine (mine-1, A), the system will bring to you various mining operations very similar, similar and compatible to the mining operation the designer just input. Here the A is a variable. mine-1 is the mine identity designer specified. Reviewing the information from similar cases, designer will feel more comfortable and confident for further specifications.

Suppose the designer has decided to use hydraulic backfill. He has to specify the characteristics of fill material and pipeline layout of the transportation system. Again, from design menu options 2. Fill Characteristic input, and 3. Transportation setup input, designer can specify the system parameters. For example, the data entry form will ask the following parameter from the user when selecting the 'Transportation setup input' menu:

1. Transportation type;
2. Capacity;
3. Ratio of total pipe line length to vertical different;
4. Vertical difference;
5. Pipe diameter;
6. Flow rate;
7. Pump ID;

At this time, the user might not be able to specify all these parameters, PumpID for example. The designer could either skip this question and let the system recommend certain pump based on the information and rules defined in knowledge base, or specify based on personal knowledge first and let the system to justify the specification later using evaluation menu. Suppose the designer skips this question and answers all other questions, he can query the knowledge base to get the recommended pump ID:

?recommendedpump(mine-1, A)

Here mine-1 is the mine ID, A is a variable which will be bound with any pump satisfy with knowledge base. The result of the query might be one pump, several pumps, nothing or a negative number. When answering is only one pump, it means there is only one pump satisfy with the condition applied. If more then one pump is retrieved, the designer has to choose one from retrieved pumps. If nothing is retrieved, it means there is no pump on the market suitable for this particular case. If a negative number is resulted, it means that the pressure needed for the system is negative, i.e. no pump is needed and the system can totally rely on gravity alone. Since the rule to define the recommended pump is based on both theoretical calculation and statistical analysis, it will be a conservative recommendation. In our case, the recommended pumps are Pump-1, Pump-4, Pump-7 and Pump-8. The original design using pump-7 which is included in the recommended pump so it is acceptable at least.

Now that the whole hydraulic transportation system has been specified, critical decision, pump selection, has been made, we need to evaluate the overall specification. The prototype knowledge base system provide a function to evaluate the system both from technological and economical point of view by accessing menu 'evaluation':

```
**************************************************************
*                                                            *
*              1. Cost evaluation                            *
*                                                            *
*              2. Pump selection evaluation                  *
*                                                            *
*              3. Help                                       *
*                                                            *
*     Please specify the menu(1-4) or press Ecs to escape:   *
*                                                            *
**************************************************************
```

The cost evaluation is based on the statistics of regional backfill survey. The system actually provide users with different model according to user's interest:

1. Quebec regional model
2. Canada nationwide model
3. Global Model

In our system only Quebec regional model is available. Querying to the knowledge base system, designer we get a rank of the specified backfill operation. In our case, the answer is 'acceptable' which means the specified backfill operation is acceptable in terms of the Quebec regional backfill operation. There is still some margin to improve the design. Technically, the designer can evaluate the critical decision made on pump selection by using menu 'pump selection evaluation' which will result in certain recommended pumps. If your specification is not contained in the recommended pump list, the specification is considered unacceptable. In this case, the pump-7 is based on system recommendation, further evaluation will always be acceptable. Repeat the same procedure manually according to empirical formula of pump pressure loss calculation, Pump-7 is still recommended pump.

This example clearly demonstrates how the knowledge base system can be used to help hydraulic transportation system and how the critical decision can be automatically or semi-automated made. Depending on the rules defined and previous mining operation data collected, the knowledge base can be substantially improved to solve more complicated problems.

## 11.4. CONCLUSIONS AND POSSIBLE FUTURE WORK

Mine design is a data intensive and time consuming procedure. Knowledge base management system is providing us with the capability to greatly increase the efficiency of information process and analysis. This paper presents the basic concepts of knowledge base management system and its possible application in mine design. In particular, the LDL deductive database system is proposed as basic machinery to implement a rule-based application for backfill design. As an example, the hydraulic transportation system was taken for close analysis. A formal representation of application knowledge base is presented fully within the LDL paradigm. Finally, the prototyping of the pump selection knowledge base system was implemented and main features demonstrated.

There is still considerable work to be done before the knowledge base management system can be fully applied into the backfill design. The collection of real operation knowledge is in the centre of the application which is by no means an easy work. Expansion of the application into the full backfill design scope is another issue worth of exploring, which needs a profound knowledge of backfill design rationale. A complete specification of functionality of the system requirement should be performed prior to any further implementation.

# CHAPTER 12

# SUMMARY, CONCLUSION AND FUTURE PROSPECTS

## 12.0. SUMMARY

This thesis presented an integrated decision supporting system for backfill design based on various computer technologies. It is recognized that the backfill design is a multi-level data/information/knowledge intensive process. Efficient information search and process plays important role in decision makings involved in backfill design. Two primary areas of information of backfill domain are distinguished: 1). Reference documents such as standard specification described in backfill handbook, literature and other government regulations; 2). The practical operation information and the knowledge of experienced personnel. The integrated decision supporting system designated in this thesis provides abilities to manage and process the information related to backfill design in feasibility study level.

A thorough literature survey is conducted to collect all aspects of backfill related information. Based on the survey and profound understanding of up to date backfill technologies, a complete specification of backfill design rationale is designated, which covers major aspects and process and information needs related to backfill operation. This rationale reflects the major development of backfill technology and serves as design reference manual. In addition, a regional backfill operation survey was conducted to collect the information of backfill operations in Quebec area. This survey provides previous backfill experience in the surveyed area and reflects the backfill operation of the real world. Accordingly, a statistics backfill cost model is established for feasibility study in the area.

Based on the understanding of the backfill operation, the basic information needs and data flow during the design process are specified in the framework of context diagram which provides a systematic approach to conceptually model the application domain. This systematic specification leads to the design and partial implementation of a computer based system to assist backfill design.

From the viewpoint of computer system, the backfill design procedure can be viewed as a series of operations on a computer program which consists of an information system combining with various decision supporting tools and relevant interactive user's interfaces to guide the user to fulfill all design tasks on computer systems. Five basic components are adopted to build the overall system.

1. User interface;
2. Hypermedia database system;
3. Expert system;
4. Knowledge base management system;
5. Conventional programming modular;

Different technologies are required to build each components. An open end architecture is adopted to integrate various technologies into a single environment based on blackboard approach.

The prototyping system, based on blackboard architecture, demonstrated the successful communication between the hypermedia system and expert system and other conventional programming modular under MS-Windows environment. The prototyping of knowledge base management system is implemented on UNIX system, which provides a efficient access to the database of previous backfill operation with the power of logical reasoning. The system test using sample data based on the survey of Quebec backfill operation shows sensitive results as expected and the potential application to other mining system design.

## 12.1. CONCLUSIONS

The following points are the significant conclusions taken as the result of studies presented in this thesis.

**1. The Rationalization of backfill design**

Backfill design is a multi-level data/information/knowledge intensive procedure. Previous experience and expertise play a key role in the decision making involved, from the specification of the mining method and backfill purpose to the identification of backfill material target properties, preparation, transportation, placement and quality control, as well as the economic evaluation of the backfill venture. This procedure is formally represented as a backfill design rationale according to various considerations. From the mining engineering point of view, the backfill design rationale could be summarized as following:

1. Mining method specification based on geological and environmental conditions.

2. The backfill purpose specification to satisfy the basic criteria of the mining operation induced by certain mining method.
3. Defining of the target properties of fill materials based on the backfill purpose.
4. Defining the operation system to make the backfill material available to meet with the target properties and finally pour to the mining voids, which includes:

   1). backfill material preparation;
   2). backfill material transportation;
   3). backfill material placement;
   4). backfill operation quality control and environmental monitoring.

5. Economical evaluation of backfill system,
6. The documentation and implementation of backfill mining operation.

From the computer system point of view, on the other hand, this procedure can be viewed as an operation on a computer program which consists of an information system combining with various supporting tools and relevant interactive user's interfaces to guide the user to fulfill these six steps on computer. The database attached provides users with two functions: 1) to record the designing parameters input by users; 2) as a knowledge base to provide user with the backfill design expertise and information needed. The formal representation of application scope and functionality are presented using three representation tools following a top-down approach:

1. Context diagram
2. Data flow diagrams
3. Process description

which laid down the foundation for further analysis and formalization of backfill design procedure at lower level.

## 2. The Backfill operation survey and regional cost model

The backfill operation survey conducted within the Quebec regional area illustrates the main characteristics of backfill practice with in the area. Based on the survey results, backfill related cost are grouped as capital cost and operation cost. Under each category, various cost items are clearly defined to represent different cost aspects. therefore build up a standard for cost modeling and economical evaluation.

A Quebec regional cost model demonstrates that among all the backfill related cost items, 85% of overall cost is contributed from backfill material cost and transportation cost. This conclusion leads to a simplified backfill operation evaluation model.

### 3. Integrated decision support system

The information process involved in backfill design procedure can be very well simulated by an integrated decision supporting system using the following technologies:

1. Object-oriented programming;
2. Hypermedia database system;
3. Expert system;
4. Knowledge base management system;

A blackboard architecture is proposed to integrate all components into a single scheme. Under this open end architecture, each component is implemented with individual data representation scheme. The communication between components is achieved through a publication board.

### 4. Mining method selection expert system

As a basic component, a mining method selection expert is designed and implemented using CLIPS expert system shell. The decision tree approach is adopted to represent search space and reason procedure. The decision tree is explored by a so called Solve-Tree-and-Learn algorithm which provides two functions:

1. Traverse from root of the tree to leaves to heuristically select a mining method;
2. Traverse from leaf to root to verify a initial speculation of certain mining method;

The learning mechanism embedded in the algorithm provides a convenient knowledge acquisition facility to expand the decision tree, and hence the knowledge base. The prototyping system reflects the mining method technology in general and verified by practical examples.

### 5. Hypermedia database system

Hypermedia based system provides an associative approach to information retrieval. Backfill designer needs to access various references and information sources frequently. The information needs can be well served by a hypermedia reference manual which provides a non-linear access to design manual of various formats. Based on backfill design rationale and literature survey, a hypermedia backfill design manual system is implemented using KnowledgePro development software. The system supports basically both text and graphic links. In addition, dynamic links are also achieved by blackboard architecture. Under this architecture, the mining method selection expert implemented using CLIPS are fully integrated into KnowledgePro environment to support dynamic link.

**6. Knowledge base management system**

Mining designers need to refer to previous mining operations and draw conclusions based on similar successfully designed cases. The knowledge base management system has the capacities of both database management system and logical inference, and therefore provides an adequate technology to store, manipulate massive operation data according to various considerations. Moreover, the query to the database can be defined by logical rules which provide an automated approach to simulate the reasoning process performed by human experts. LDL system is an adequate knowledge base management system which can be used to implement applications for mining design.

**7. Development of user interface**

Under blackboard architecture, the user interface played four roles:

1. Present the backfill design procedure in a natural and easy to understand way;
2. Provide facilities to access various design tools;
3. Maintain the integrity constrains of database;
4. As a coordinator to schedule the information exchange between different programs.

Object-oriented programming technique is used to implement these functions. The underlining programming tools is again KnowledgePro.
Finally, we claim that the proposed decision supporting system is a feasible approach for mining design.

## 12.2. FUTURE PROSPECTS

This thesis presented a new approach to mining design based on decision supporting system. The future application of introduced concepts are not limited to backfill design alone. The framework proposed is suitable for other data intensive engineering applications such as mineral exploration, geological data management and interpretation, waste management, etc.. Based on the conclusions achieved in this thesis, following work are considered for further investigation.

1. This thesis defined the information requests of backfill operation at data flow diagram level. Data requests in process description level are only introduced with the example of

mining method selection expert where a decision tree approach is used. A detailed task specification is essential to define various task solvers and integrity constrains. Further rationalization of backfill design procedure at lower level is still needed. Instead of current top-down approach, another possible alternative for further development is to implement the current specification as it is and incrementally increase task solvers when the requests appears. This alternative approach is possible because of the open end architecture specified in the thesis.

2. The mining method selection expert system presented in the thesis reflects the principles of mining method application. The decision tree approach represents only one kind of techniques. Other approaches are worth more investigation. The learn mechanism of current system does provide a convenient approach for knowledge acquisition, but lacks of the ability to maintain the consistency of the knowledge base. To represent the real mining application, a complete dynamic knowledge base and management system is obviously a further step to explore.

3. Hypermedia backfill design reference manual proposes an open structure to store and manipulate various formats of data. It can be used to integrate other technology to develop high level graphic user interface. In essence, the hypermedia system does not cope only with information process, but also serves as user interface development tools. By doing so, the links will no longer be always linked to static node, but rather to other links or procedural calls to impose modularity. In the application level, one natural extension of the current prototyping system is to implement a tour mechanism, which will provide more sophisticated facilities to support tutorial features of hypermedia system for novice backfill designers and operators. A profound understanding of domain knowledge of backfill design is crucial in designing the touring chain. The hypermedia system also opened possibility to simulate backfill design process under the supervision of expert system. In this process, the backfill designer has opportunity to try different options, and the hypermedia system will cooperate dynamically to demonstrate possible consequences from various information source. This requires more complex searching strategies which will be a role played by an expert system.

4. The knowledge base management system presented in the thesis represents a preliminary application of logical programming in engineering design. The crucial part of the application is the previous operation information. The current implementation is based on a regional survey and various assumptions. A systematic information soliciting approach is essential to collect data in a wider area. The extension of current application to full scale backfill design needs careful examination. The transfer of the knowledge

base management system from UNIX system PC based system depends on further development of computer software technology which is beyond the scope of this thesis.

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# APPENDIX

The main objects identified are the following:

The object hierarchy related to the base object EQUIPMENT is shown in Figure 8-7.

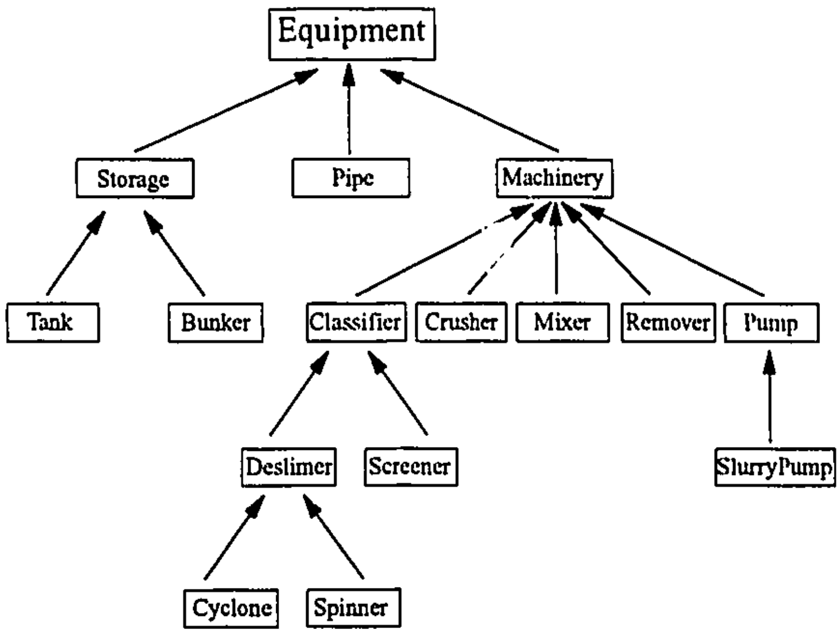


Figure 8-7 The object hierarchy of the base object EQUIPMENT

Start from the object equipment, the basic abstraction of structure tree of the objects related are shown sequentially on the figures based on the notation of generalization and specialization given in figure 8-5.

The object hierarchy related to the base object MATERIAL is represented in figure 8-8:

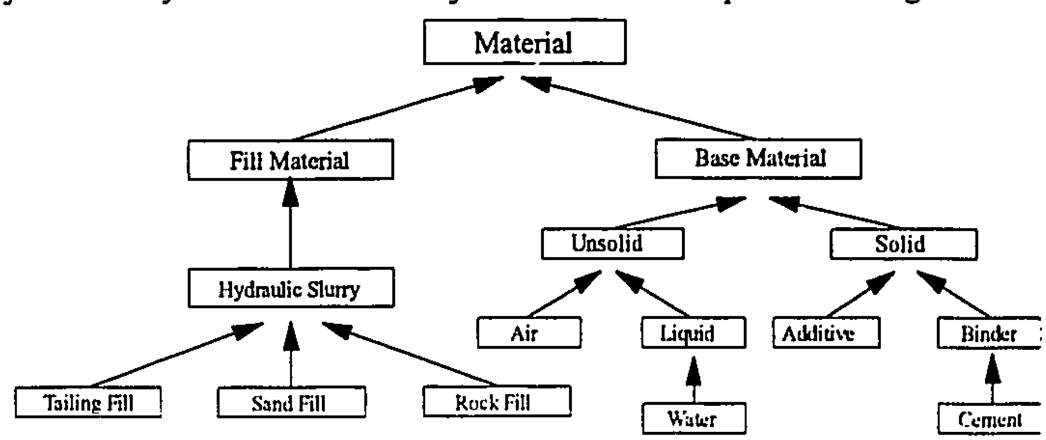


Figure 8-8 The object hierarchy of base object MATERIAL

The abstraction of the structure tree is represented based on the notations of generalization and aggregation.

The object hierarchy of the base object DEPOSIT is shown on figure 8-9. The structure consists of three objects associated by the notation of generalization.

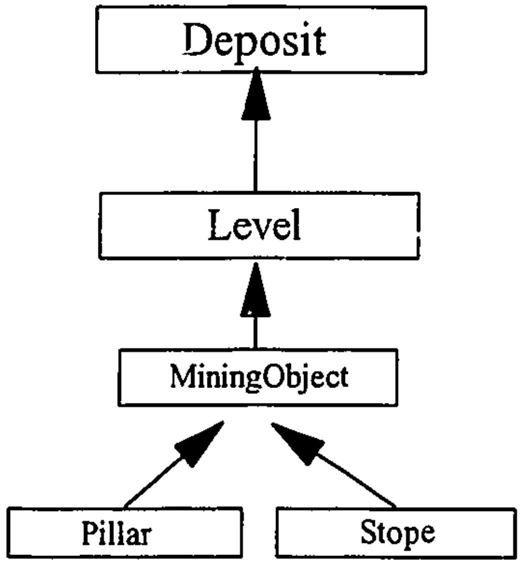


Figure 8-9 the object hierarchy of the base object DEPOSIT

shown in figure 8-10 is the hierarchy of the base object MINE, which is represented by using the notations of the generalization and aggregation.

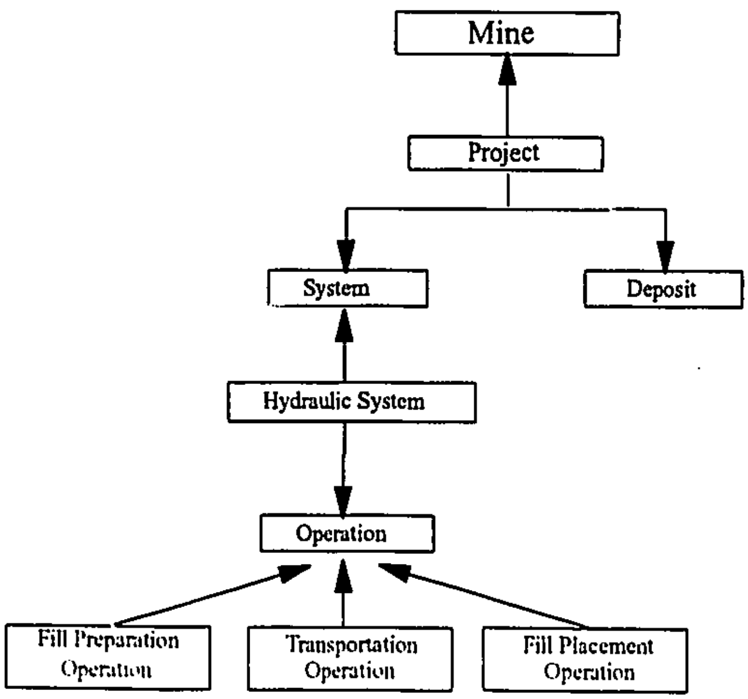


Figure 8-10 The object hierarchy of the base object MINE

Shown in figure 8-11 is the Object MiningTechniques(Mining Method):

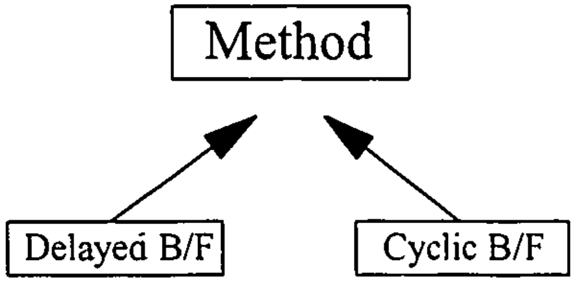


Figure 8-11 The Object Hierarchy of Base Object MiningTechnique

The main objects specifications are the following:

1 The specification of base object EQUIPMENT:

```
Type EQUIPMENT=Record Equipment ID:      Number;
                      EquipmentType:     (Storage, Pipe, Machinery);
              END.

    Subtype PIPE=Record  Equipment ID:Number;
                      Pipe ID:      Number;
                      PipeType:     String;
                      Model ID:     Number;
                      Length:       Number;
                      Roughness:    Number;
                      UnitPrice:    Number;
              Type Manufacturer=    Record(refer to 'manufacturer');
              END.

    Subtype STORAGE=Record Parents:       Equipment;
                           Storage ID:    Number;
                           Volume:        Number;
                           Structure:     Char;
                           Type  Pump=Record (refer to object pump)
                           Cost:  Number;
                  END.
    Subtype MACHINERY=Record Parents:     Equipment;
                           Machinery ID:Number;
                           MachineryType: (Crusher, Classifier, etc.);
                  END.
```

```
Subtype CRUSHER=Record Parents:      Machinery;
                       Crusher ID:   Number;
                       CrusherType:  Char.
                       Capacity:     Number;
                       Size Range:   Number
                       Operators:    Number;
                       Power:        Number;
        Type Manufacturer=Record     (refer to 'Manufacturer');
                       Cost:   Number;
            END.
Subtype CLASSIFIER=Record Parents:   Machinery;
                       Classifier ID: Number;
                       ClassifierType:(Deslimer, Screener);
                       Capacity:     Number;
                       Size Range:   Number
                       Operators:    Number;
                       Power:        Number;
        Type Manufacturer = Record (refer to 'manufacturer');
                       Cost:         Number;
            END.
Subtype DESLIMER=Record Parents:     Machinery;
                       Deslimer ID:  Number;
                       Deslimertype: (Cyclone, Spinner);
                       Rotation:     Number;
                       S.C.D.:       Number;
            END.
SubType CYCLONE=Record Parents:      Deslimer;
                       Cyclone Type: Char.
                       Overflow:     Number;
                       Underflow:    Number;
            END.
Subtype SPINNER=Record Parents:      Deslimer;
                       Rotation:     Number;
                       Diameter:     Number;
            END.
```

```
Subtype PUMP=Record Parents:    Machinery:
                    Pump ID:    Number:
                    Pumptype:   (WaterPump,SlurryPump);
                    Capacity:   Number;
                    Power:      Number;
                    Head:       Number;
               Type Manufacturer=Record(refer to 'Manufacturer').
                       Price:   Number;
                    END.
Subtype SLURRYPUMP=Record    Parents:      Pump;
                             Mode:         Record
                             Abrasivity:   Number;
                    Type  PumpingSlurry=Record(refer to 'material').
                          END.
Type MANUFACTURER=Record  Manufacturer ID:    Number;
                          Manufacturer name:  Char;
                          Address:            Char;
                          State:              Char;
                          Country:            Char;
                          Telephone:          Char:
                    END.
```

2 The specification of base object MATERIAL:

```
Type MATERIAL=Record  Material ID:   Number;
                      MaterialType:  (Rock, Sand, Tailing);
                      Binder:        (Cement, Flyash, Aggregate, Slag);
                      Additive:      Char;
                      UnitWeight:    Number;
                      U.C.S:         Number;
                      Y. Modulus:    Number;
                      I.F.A.:        Number;
                      Poisson:       Number;
                      H.T.:          Number;
                      Shrinkage:     Number;
                      VoidRatio:     Number;
                      Viscosity:     Number;
                      Cohesion:      Number;
```

```
        Liquefaction:  Number;
        Corrosion:     Number;
        Spontaneous:   Number;
        GasEmission:   Number;
        Cost:   Number;
    END.

SubType ROCKFILL=Record Parents:   Material;
                  Rock ID:         Number;
                  Rock name:       Char;
                  Shape Factor:    Number;
                  SizeRange: (Rock, Sand and Tailing size);
                  UnitWeight:      Number;
                  S.U.W.:          Number;
                  Cost:            Number;

        END.
SubType SANDFILL=Record Parents:   Material;
                  Sand ID:         Number;
                  Sand name:       Char;
                  Shape Factor:    Number;
                  SizeRange:   (Rock, Sand and Tailing size);
                  UnitWeight:      Number;
                  S.U.W.:          Number;
                  Cost:            Number;

        END.
SubType TAILINGFILL=Record   Parents:   Material;
                  Tailing ID:      Number;
                  Tailingname:     Char;
                  Shape Factor:    Number;
                  SizeRange:   (Rock, Sand and Tailing size);
                  UnitWeight:      Number;
                  S.U.W.:          Number;
                  Cost:            Number;
        END.
```

```
SubType CEMENT=Record  Parents:     Material;
                       Binder ID:   Number;
                       Cement ID:   Number;
                       Percentage:  Number;
                       Strength:    Number;
                       UnitWeight:  Number;
                       Cost:        Number;
                 END.
```

Note: S.U.W. Stand for the Saturated Unit Weight;
U.C.S. Stand for the Uniaxial Compress Strength;
Y.Modulus Stand for the Young's Modulus;
I.F.A. Stand for the Internal Friction Angle;
H.T. Stand Hardening Time of fill material;

3 The Specification of Base Object DEPOSIT:

```
Type DEPOSIT=Record  Mine ID:     Char;
                     Deposit ID:  Number;
                     Minerals:    Char;
                     Grade:       Number;
                     Gravity:     Number;
                     Reserve:     Number;
                     Price:       Number;
            END.
    SubType LEVEL=Record  Parents:     Deposit;
                          Level ID:    Number;
                          Reserve:     Number;
                          Altitude:    Number;
                          Extension:   Number;
                          Major P.S:   Number;
                          IntermediateP.S.:Number;
                          Minor P.S.:  Number;
                    END
    Subtype MININGOBJECT=Record  Parents:   Level;
                                 Height:    Number;
                                 Width:     Number;
                                 Length:    Number;
                                 BulkheadType:(Bulkhead, Fence);
```

```
                    Type  Bulkhead=Record      (Refer to 'Bulkhead');
                    Type  Method=Record        (refer to 'Method');
                               MiningObject: (Stope, Pillar);
                               ObjectLife:   Number;
                               END.
      Subtype BULKHEAD=Record  Mine ID:      Number;
                               Project ID:   Number;
                               Bulkheadtype: (Wood,Concrete);
                               Maximum P.:   Number;
                               DrainageType: (Pipe, Filter);
                               Length:       Number;
                               Width:        Number;
                               Height:       Number;
                               Cost:         Number;
                          END.
      SubType SUBLEVEL=Record Parents:         Level;
                          MiningObject:        (Stope, Pillar);
                          Subievel Height:     ·Number;
                          Sublevel Length:     Number;
                          Sublevel Width:      Number;
                    END.
```

Note: R.Q.R. Stand for the Rock Quality Requirement of the referred Mining Method, Specified either by certain text or quality index.

4 The Specification of the Base Object MINE:

```
   Type MINE=Record  Mine ID:     Number;
                     MineName:    Number;
                     Address:     Char;
                     Company:     Char;
                     MiningLife:  Number;
         END.
      Subtype PROJECT=Record  Parents:       Mine;
                              Project ID:    Number;
                              Project Name:  Char;
                              Investment:    Number;
                              Production:    Number;
                              LabourCost:    Number;
```

PowerCost: Number;
M.P.S.: Number;
W.D.Y.: Number;

Type Deposit=Record Mine ID: Number;
Project ID: Number;
Deposit ID: Number;
Mineralogy: Char;
Grade: Number;
Gravity: Number;
Reserve: Number;
Price: Number;
END.

Type Operation= Record (refer to Object Operation )
END.

Note: W.D.Y. stand for the Working Day of One Year;
M.P.S. stand for the Maximum Particle Size permitted;

Type OPERATIONSYSTEM=Record Parents: Project;
Capacity: Number;
Labour: Number;
Power: Number;
OperationCost: Number;
SystemType: Char;
END.

Subtype HYDRAULICSYSTEM=Record Parents: Operationsytem
M.C.F.: Number;
W.R.: Number;
S.C.T.: Number;
TypeOperation=Record(Refer to the Object Operation);
END.

Note: W.R.stand for the Water Requirement for the Hydraulic System;
T.C.stand for the Slurry Concentration for Transportation.

Type OPERATION=Record Mine ID: Number;
Project ID: Number;
Labour: Number;
Power: Number;
A.P.S.: Number;

```
                    OperationTask:  (Crushing, Desliming, Screening,
                                    Repumping, Mixing, Transportation
                                    Placement);
                END.
Subtype     CRUSHING=Record Parents:       Operation;
                            SizeRange:     Number;
                    Type Crusher=Record(refer to 'Crusher');
                    Type Feeder=Record  (refer to 'Feeder');

                END.
Subtype DESLIMING=Record Parents: Operation;
                DeslimingType:  (Hydrocyclone, Classifier, cte.);
            Type   Machinery=Record    (refer to Machinery);
    END.
Subtype SCREENING=Record Parents:      Operation;
                Type   Screener=Record (Refer to Screener);
            END.
Subtype REPUMPING=Record        Parents:        Operation;
                                Project ID:     Number;
                                WaterFlow:      Number;
                                WaterPressure: Number;
                        Type    Pump=Record: (Refer to  'Pump');
                END.
Subtype        MIXING=Record    Parents:        Operation;
                                Binder:         Char;
                        Type    Mixer=Record:(Refer to Mixer);
                END.
Subtype TRANSPORTATION=Record  Parents:    Operation;
                        FlowRate:      Number;
                        Flowstatus:    Char;
                        Velocity:      Number;
                        Critical V.:   Number;
                        Pressure Loss: Number;
                        PumpingType: (LocalStation,GobalStation);
                END.
```

```
Subtype     LOCALSTATION=Record  Parents:      Transportation
                                 Level ID:      Number;
                                 Maximum E.:   Number;
                            Type Pump=Record (Refer to 'Pump');
                                 A.P.S.:       Number;
                                 H.E.P.S.:     Number;
                            END.
Subtype GLOBALSTATION=Record Parents:      Transportation;
                            Maximum E.:   Number;
                            A.P.S.:       Number;
                            H.E.P.S.:     Number;
                      Type  Pump=Record (Refer to Object Pump);
                 END.
     Subtype     PLACEMENT=Record  Parents:          Operation;
                                 OperationTime:     Number;
                                 FillRequired:      Number;
                                 PlacementType:     (Pneumatic,
                                                    Hydraulic,
                                                    Mechanic);
                      Type  Machinery=Record (Refer to 'Machinery');
                 END.
```

5 The specification of the base object MiningTechniques:

```
     Type   METHOD=Record Method ID:     Number;
                      Classification:    (Delayed B/F,  Cyclic B/F);
                      Method Name:       Char;
                      Productivity:      Number;
                      R.Q.R:             Number;
                 END.
```

# GLOSSARY

| | |
|---|---|
| Agenda | Agenda is a prioritized list of rules created by the inference engine, whose patterns are satisfied by facts in working memory, 9-4. |
| Authoring | The process of creating hypermedia documents and database. The hypermedia author must make a series of decisions about how to organize the entire database and individual documents during the authoring process, 10-8. |
| Class | A class defines an abstract data type which incorporates the definition of the structure as well as the operations of the abstract data type, 8-14. |
| DBMS | Data-base management system, 11-2. |
| Declarative Language | A language that the user specifies what he wants from the database in a high-level declarative style of programming, 11-2. |
| Domain Expert | Human expert who familiar with certain professional area, 9-6. |
| DSS | Decision support system, 1-1. |
| Expert system | Expert systems are computer programs which use experience to assist in a variety of problems, 9-1. |
| Facts | General statements of certain event, 9-20. |
| Host language | Language such as PL/I, COBOL, and C are refereed to as host language when SQL is embedded in them, 11-2. |
| Hypermedia | A database has links between discrete pieces of data. The data can be graphics, numerical numbers, text and sound, 10-1. |
| Inference | The process of combining facts and rules to deduce new facts, 9-2. |
| Inference engine | Part of an expert system that carries out the function of reasoning is called inference engine. The most commonly used inference engine is forward chaining, backward chaining etc., 9-4. |
| KBMS | Knowledge-base management system, 11-2. |
| Knowledge base | Part of an expert system where the long-term memory of facts, rules that represent expert knowledge about the domain of expertise are stored, 9-4. |

| | |
|---|---|
| LDL | Logical Data Language, 11-2. |
| Links | Links are the labels that connect one node with others, 10-3. |
| Nodes | A single document in a hypermedia database is a node, describing a single concept or topic. Continuity to other documents is provided by links, 10-3. |
| Non-Sequential | Access to information sources without any pre-fixed sequence such as hypermedia system, 10-2. |
| Object | Object is the instance of class, 8-14. |
| Rules | The production rules are If-then statements that states the relationship between facts, 9-20. |
| SALAD | System for Advanced Logical Application on Data, 11-4. |
| Sequential access | Access to information sources following certain pre-fixed sequence such as reading books, 10-2 |