A study of sequencing strategy for steep, tabular, hardrock orebodies

by

Charles W. Pelley

Department of Mining and Metallurgical Engineering

McGill University, Montreal

April 1994

A thesis submitted to
the Faculty of Graduate Studies and Research
in partial fulfilment of the requirements of the degree of
Doctor of Philosophy

ISBN 0-315-94701-2

ABSTRACT

This thesis reviews, analyses and classifies extraction methods and sequences used in steep, tabular orebodies in Ontario hardrock mines and how new bulk mining techniques have affected them.

The thesis examines, as a case study, the complexities of sequencing the extraction of the Hemlo orebody through three adjacent mines and details the planning and monitoring of extracting one section of the David Bell orebody. It examines the use of numerical modelling as a tool in extraction sequence planning and demonstrates how, in conjunction with an instrumentation program, the results assist successful completion of the plan.

The thesis concludes that bulk mining sequences have increased resource extraction and productivity, lowered costs and improved ground control aspects of extraction planning; but have compromised grade control. This aspect should be improved to maximize the economic benefits. In addition, as automated or continuous mining methods are developed, their benefits should be analyzed in the context of overall sequence planning objectives.

RESUME

Cette thèse revise, analyse et classifie les méthodes d'extraction et les séquences utilisées dans les gisements tabulaires fortement inclinés des mines de l'Ontario, et comment elles ont été affectées par les nouvelles techniques d'exploitation en vrac.

La thèse examine, sous forme d'étude, la complexité de déterminer l'ordre d'extraction du gisement Hemlo à travers 3 mines adjacentes, et détaille la planification et le contrôle de l'extraction d'une partie du gisement David Bell. Elle examine l'utilisation du modelage numérique comme outii pour planifier l'ordre d'extraction et démontre comment, conjointement avec un programme d'instrumentation, les résultats aident à compléter le plan avec succès.

La thèse conclut que les séquences d'exploitation en vrac ont augmenté le pourcentage d'extraction des ressources et la productivité, diminué les coûts et amélioré le côté stabilité de la planification de l'extraction, mais ont compromis le contrôle de la teneur. Cet aspect doit être amélioré pour maximiser les avantages économiques. De plus, lorsque des méthodes d'extraction continue ou automatisées sont développées, leurs avantages devraient être analysés dans le contexte des objectifs de planification globale de la séquence d'extraction.

ACKNOWLEDGMENTS

I would like to thank management and staff of the David Bell Mine for the access they provided me to the mine and their data during this research project. I would particularly like to thank Ed Kardis, Francis Sherridan, and Ken McKirdy for their support and advice. I am grateful to the staff at the many other mines who provided tours, guidance and access to various mine reports to help with developing my ideas regarding mine sequencing.

I sincerely thank my supervisor, Dr. Malcolm Scoble of the Department of Mining and Metallurgical Engineering at McGill University. He provided advice and support through the never ending process of preparing this thesis and yet allowed me the freedom to develop my ever changing ideas.

I would like to acknowledge the financial support of the Teck-Corona Operating Corporation and the University Research Incentive Fund of the Ontario Ministry of Colleges and Universities.

Finally I would like to thank my wife Yvonne and children Jennifer, Karen and Andrew for putting up with me during this period. The research project was not always my first priority but too often was placed ahead of family events.

TABLE OF CONTENTS

LIST OF TABLES

LIST OF FIGURES

CHAPTER 1

INTRODUCTION

Technological developments in the past ten years have enhanced the economic competitiveness of underground mining in Canada, mainly through the application of bulk mining technology. To remain competitive, research is continuing into new mining techniques, particularly into the use of continuous mining systems and the applicability of automation and robotics.

This research began when the conversion to bulk mining systems was ongoing at many mining operations. These new systems were based on the increased drilling accuracy possible in longer and larger diameter blastholes from using the in-the-hole drill. The bulk methods include longhole, blasthole and vertical retreat stoping. Large openings are created and the placement of backfill is delayed until individual stope extraction is complete.

A series of rockbursts which began at the Campbell Mine in December 1983 had culminated with a burst killing four miners at the Falconbridge Mine on June 20, 1984. The introduction of bulk mining systems was considered by some as a possible cause of this increased incidence of rockbursting in Ontario mines. This research was undertaken to analyze and assess how the switch to bulk mining systems had affected the ore extraction strategies of hardrock underground mining operations.

These new systems are capital intensive and generally provide increased productive capacity from individual working locations. As a result, there are fewer working areas but the extraction ratio in the initial mining area increases more rapidly and interaction with backfill and pillar removal occur earlier in the mine life. This makes the mine design and planning functions more critical to the safe and economic extraction of the mineral resource.

It is certain that the future mining operations in Ontario will be required to extract ore at increasingly greater depths. The future will also see the introduction of continuous, automated and robotic mining systems, partially as a result of the problems of mining at depth. These systems will bring further changes to mining sequences. Before new mining systems are introduced their effect on the overall mine design and personnel safety should be assessed. This research would serve as an initial analysis of the possible sequences required for such systems to be safely and successfully implemented.

1.1 MINE DESIGN OBJECTIVE

All mining operations must extract and market a mineral commodity at a profit. Proven ore, as defined by the Association of Professional Engineers of Ontario, requires a closely spaced geological sampling program providing a reliable estimate of the mineral resource and that a current design exists which is both technically and economically feasible. The technical and economical feasibility of the mine design is provided by optimizing a complex set of variables through the long range plan or mining sequence.

An extensive literature review on open pit mine design optimization was recently provided by Koniaris (1). He states, " the objective of open pit design is to develop the best strategy for the depletion of an economic ore reserve" and describes the primary objective as, "the maximization of the net present value of the orebody". This requires three design stages: determining the final pit limit and creating both long and short range production plans. In discussing the practical limitations of optimizing open pit design, Koniaris states "the engineering time required to produce the optimum solution is often not justified", and argues for deriving a feasible solution under the constraints of accepted behaviour or rules of thumb.

The objective of underground mine design is also to develop a depletion strategy which maximizes the economic benefits from the ore reserve. This requires the same three design stages; definition of the minable orebody and facilities and the development of both long and short range plans, but they are more complex and the constraints more varied than for an open pit operation. The engineering effort required to produce the optimum design is increased by constraints such as ground control or ventilation requirements and often by a reduced level of information available on the geological, economic and geotechnical aspects of the orebody at depth. There is an interdependence of certain critical decisions, for example, the final design of a mine access system is dependent on the delivery of the personnel, equipment, ground support supplies or backfill demanded by the mining method.

While this thesis deals with developing the extraction sequence as part of the long range plan, the three aspects of the overall design are discussed briefly in the following sections.

1.2 THE DESIGN STAGE

The successful implementation of a mining operation begins with the design of the physical plant and openings to support the extraction plan. This is an iterative process, often seeing the design change during the mine life based on additional information or changed economics. Information required by the design team is often supplied by a separate exploration group whose priorities are to geologically define the orebody and quantify its content of economic mineralization. It should be emphasized that a mining engineer should become involved in such a project at the earliest possible time to ensure the collection of the geotechnical data required for the design process.

Designing an underground mine requires determining the eventual depth of the operation, outlining the permanent mine openings, and designing the physical plant to support the extraction. Minimizing the capital cost of these facilities is one method of maximizing the NPV of the operation. The extraction sequence, however, is directly affected by extraction facilities which may or may not initially provide access to the entire orebody. If access is limited, the sequence which maximizes the NPV may be precluded due to grade variations in the orebody or by the increased costs in a second stage of mining. The chosen productive capacity of the main mine access, generally a vertical shaft, will constrain the capacity of the mine for many years.

As an example of this, a study by Makuch (2) examines methods to maintain the productive capacity of the Campbell Red Lake Mine to depth. He recommends an entirely new hoisting facility as the present capacity can not be maintained without major modifications requiring significant production losses. The siting of this new facility will significantly affect the extraction sequence of the ore at depth, yet little information on the nature of the reserve at depth is presently available.

1.2.1 Information Availability

The major reason for limited information availability at depth is that the cost and complexity of drilling to depth limits the number of surface exploration holes which will be attempted. The cost of each wedged hole drilled on INCO's Victor deposit was reported to be approximately $1,000,000 to give three orebody intersections, so the sinking of an exploration shaft is planned. Environmental approval for an exploration shaft at Cameco's McArthur River project was justified by the impossibility of obtaining adequate information from surface drilling to produce a design which could meet the rigours of the final environmental review process. Based on limited surface drilling, a 1500 metre circular concrete shaft was sunk at the Ansil Mine to complete exploration from underground, a shaft which eventually serviced the production phase.

Even when more detailed drilling is carried out, however, information required for the design process is often not recorded or is permanently lost if only exploration objectives are considered. The minimum requirement should be that the Rock Quality Designator (RQD) of the core be recorded near and in the ore zone. While the RQD gives an approximation of the jointing of the rock mass, no recording of the joint orientation is normally attempted. If the RQD is not recorded immediately, core in the vicinity of the ore zone is normally split, obviating the possibility of a core relogging program to recover this information.

Data provided for a geotechnical study carried out by the author on a property in Eire, consisted of over 200 holes drilled in the shallow carbonate deposit without recording water levels encountered either in the overburden or the bedrock. No RQD's were recorded, requiring a complete core relogging program after ore intersections had been split for analysis. Only two holes extended more than one metre into the orebody footwall although this is where all extraction drifts would be located. Preventing water inflows from the overlying soil aquifer is considered critical to success of the mining operation, yet no holes had been grouted and so were open to the bedrock surface. While this example may be extreme, it demonstrates how information critical to the design and sequencing of an orebody can be lost.

1.2.2 Choosing a Mining Method

The initial design of an underground mining operation requires that a mining method be chosen to form the basis of the feasibility study. The detailed design usually occurs after project financing has been committed and methods are often adjusted as the project proceeds and additional information becomes available. If a major method change is required, the economics of the project can be severely altered. If ground conditions require a selective method such as cut and fill versus a lower cost bulk method for example, a significant change in operating costs can occur. If a less selective method is required, even though the cost per tonne may decrease, limitations of the hoisting or milling facilities may prevent handling a higher volume of lower grade material.

The choice of mining method restricts the long range plan by limiting the possible extraction sequences. The following factors affect the choice of mining method with each factor being evaluated based on the assumption that some form of bulk or longhole method will be chosen wherever possible.

1.2.2.1 The Physical Dimensions

The overall size of a deposit forms the basis of selecting a mining rate and justifying the capital expenditures to provide permanent mine openings. The most important physical characteristics affecting the choice of a mining method are the following:

A. Deposit Thickness

Narrow deposits generally preclude certain methods from consideration. If the orebody is regular in shape, the increased accuracy of drilling systems has considerably reduced this restriction. Longhole mining systems have now been used in thicknesses approaching 1 metre (3) (4), although the drill access sublevel spacing is reduced accordingly.

In thicker orebodies, longhole methods can be used successfully in more irregular ore or the sublevel spacing can be increased. Longitudinal stope layouts continue until the ore becomes so wide that openings cease to be self-supporting, usually at around 15 metres. Transverse layouts are then used with the stopes elongated perpendicular to the strike. Stopes are then designed at an optimal width and can accommodate irregularity in the footwall and hangingwall, but the reduced ratio of the strike length of the pillars to their transverse extension may place tight constraints on the extraction sequence.

B. Orebody Dip

The dip of the orebody determines if gravity can be used in the mining method with the approximate 50^0 angle of repose being the key parameter. Generally speaking, below that angle some adaptation of room and pillar methods will be applied. As the thickness of flat-lying orebodies increases, the geometry can be adapted to the use of longhole methods. Even though room and pillar methods have been used in ore thickness up to 30 metres, these deposits would likely be adapted to bulk methods today, as is the case at Mount Isa in Australia where a unique extraction sequence has been developed as will be described in Chapter 4.

C. Orebody Shape

The major item of importance in defining orebody shape is the regularity of the orebody boundaries. The decision to use a bulk method versus a more selective method is determined by the waste which would be included as dilution by drilling a straight hole from an overcut to an undercut horizon. Alternately, if rich offshoots could be expected at the stope boundaries, recovery of these offshoots must justify the cost of the more selective method.

One must also consider the cycle timing problems developed by mixing mining methods. An irregular stope observed at the Fraser Mine contained a high grade nickel "onionskin" complexly intertwined with a core of waste material. The entire stope was being mined as a bulk stope although the dilution was expected to be 50%. To do otherwise would require the supply of special skills and services not available in the area to service the more regular stopes and the longer mining time would place the entire level off schedule. In summary, any switch to selective methods in a mine normally using bulk methods has to be economically justified.

1.2.2.2 The Physical Parameters

The physical or geotechnical parameters include rock strength, fracturing, jointing, or large scale geological features such as faulting or bedding. The host rock characteristics are as important as those of the ore. These will determine how large an opening will be stable, how long it will remain stable, how large a load can be carried by pillars and how they are expected to shed it. Detailed information on these parameters may not be available during the initial design and flexibility should be available in the extraction sequence to allow for design adjustments as more information becomes available during mining operations.

1.2.2.3 Deposit Depth

Rock stresses increase with depth and influence the mining method which can be used and the primary extraction ratio which can be achieved. As an example, shrinkage stoping methods have not been successful at depths much in excess of 1300 metres. Mining methods and extraction sequences developed and proven successful at the upper limits of the mine may fail as the depth and stresses increase.

1.2.2.4 Grade of the Deposit

The grade of the deposit affects the selection of a mining method or extraction sequence in two ways:

1. The overall ore value determines the recovery which is economically justified. Increasing the overall recovery may increase mining costs for the entire deposit and the additional ore recovered must justify these additional costs.

2. Grade variations in a deposit may demand the use of a selective method or require additional working locations to provide grade control. These extra locations may complicate the extraction sequence.

1.3 DEFINING THE MINING SEQUENCE

The second phase of mine design is to develop a plan for the orderly and optimal extraction of the reserve. In underground mining, this long range plan involves the sequencing of mining activity throughout the overall orebody. A high degree of flexibility exists, in steeply dipping orebodies, as any area of the orebody can be accessed independently. There are, however, many constraints, ground control problems for example, which must be considered when defining the sequence.

Morrison (5), presented a general classification of mining methods which also applies to describing or classifying the overall mining sequence. He states: "It is impossible for any classification to embrace all the differences in practice flowing from the possible variations in physical conditions. However, a general classification should indicate the basic requirements for different methods. Ground control facilities are the primary consideration in the selection of a mining method and should be a distinguishing feature in any classification."

He continues: "Even when favourable initially, bad ground can result from the inept application of the most appropriate method. In all mining operations, ground conditions gradually progress towards failure and, within limits, mining methods must be adaptable to such changes. In the process, the ground control facilities of one method may gradually displace those of another. Thus, in practice methods are compromises with clear-cut demarcation lines only under ideal conditions." (5)

Morrison concentrated on stope scheduling to cause a coalescence of openings and thus induce shedding of rock stresses to the solid abutment. This control of strain energy and its associated closure, and the associated ground control problems and costs, is still considered as an important consideration in the choice of both mining methods and mining sequences.

It is not the only factor to be considered, however, as with the large capital investment required to develop a modern mining operation and within today's legislative environment, these mine extraction sequences must be formulated within the following constraints:

1. **The Management Environment.** Management objectives are often based on the availability of capital funding and the requirement to provide a return on that funding. The requirement to quickly establish and maintain a positive cash flow is often critical to company survival.

2. **The Legislative Environment.** Mines must operate within the legislative guidelines of the jurisdiction in which they are located. Maximizing extraction of the mineral resource, generation of revenues through taxation systems, ensuring the safety of the workplace, and protecting the physical environment are usual priorities of controlling legislation.

3. **The Geological Environment.** Both the geotechnical and economic parameters of the orebody and the variation in their values occurring spatially in the orebody and along its margins are critical in choosing the optimal sequence.

4. **The Technological Environment.** The technology used to extract the ore determines many aspects of how the extraction must be sequenced.

Based on these constraints, the overall objectives of the extraction sequence should be defined early in the planning process and the sequence should be rigorously implemented, bearing in mind that flexibility to adapt to changing or unexpected conditions should be built into the plan. As shown in chapters 6 and 7 of this thesis numerical modelling and instrumentation can provide important tools in planning and adapting the mine sequence.

1.4 OBJECTIVES OF THE MINE SEQUENCE

With the high development and financing costs of today's industrial climate, most managers agree that the overall objective of any long term mining plan is to maximize the net present value of the mining investment. To achieve this however, one must optimize or maximize the following individual sequencing objectives as part of that long range mining plan.

1. *To maximize the overall percentage extraction of the mineral resource.*
2. *To develop an optimum sustainable rate of extraction.*
3. *To minimize the cost per unit extracted.*
4. *To minimize the initial development time and cost.*
5. *To provide a grade control strategy.*
6. *To minimize ground control cost and problems.*

It is obvious that not all of these objectives are complementary; in fact, they may be somewhat in conflict. The development of a high level of grade control, for example, may require more initial development. To maximize the economic returns, all of these objectives must be considered by the design team and some optimal solution accomplished through the overall mine extraction sequence.

These objectives should be kept in mind as mining sequences are reviewed in Chapters 3 to 5. Each of these objectives will be discussed in more detail in Chapter 8 in the context of developing a sequencing classification system.

1.5 THE SHORT RANGE PLAN

The short range plan establishes the day to day objectives of operating personnel and generally deals with weekly or monthly time intervals. This plan covers the drilling, blasting, mucking, filling and ground support activities which are part of the extraction process. It is dependent on the long range plan as, in an underground mine, development openings must be established as part of the long range plan or the short range operations can not proceed. Delays in the short range activities, in backfilling of stopes for example, can, however, have a serious effect on the long range extraction sequence.

At some strategic time interval, usually on an annual basis, the short range plans must be adjusted to bring the extraction sequence closer to the long range plan or the long range plan itself has to be adjusted.

In conventional mining, such as shrinkage or cut and fill stoping, short range planning activities are dictated by the method. There is considerable flexibility in overall sequence planning as the number of stopes is limited only by the number of levels available and the cost of maintaining these levels. Pillar recovery is often delayed until late in the mine life.

With the advent of bulk mining methods some of the flexibility of working places was removed and a long range plan of the extraction sequence is necessary to minimize the interaction of one mining area with another. The absolute necessity of controlling this interaction to avoid dilution and expensive ground control activities placed more emphasis on mine sequencing and closer co-ordination of long and short range planning activities.

While this thesis deals mainly with the long range planning of mine extraction sequences, the case study of the David Bell Mine described in Chapter 7 shows how the stope extraction method was adapted to meet the requirements of the planned mining sequence.

1.6 JUSTIFICATION FOR RESEARCH

In the early 1980's, there was a dramatic increase in rockburst activity in Ontario mines in Elliot Lake (6), at the Macassa mine in Kirkland Lake (7), at the Campbell mine in Red Lake (8), and at the Creighton and Falconbridge mines (9) in Sudbury. The Stephenson Commission was appointed, by the Ontario Government, to investigate the occurrences and to recommend ways to mitigate the problem. In its brief to the Commission (10), the United Steelworkers Union presented the opinion that the high productivity techniques being introduced were the cause of this increased rockburst activity.

The mining industry considered that continued development of high productivity methods was essential if Ontario mines were to compete in world markets. This research program was undertaken to evaluate the new bulk mining methods, sequences, ground control techniques and monitoring requirements, and to provide a better understanding of they affected the safety and economic aspects of the mines. It concentrated on Ontario mines where over half of the hardrock mines in Canada are located, and which represent a diverse geological, geotechnical, economic and management spectrum.

This was significant, as no standard or code of practice existed for mine sequence planning and no prior research into this area had been published. It became more significant when, in 1989, the Ontario Ministry of Labour legislated that owners of a mine must prepare and maintain a mine design that should, "**describe the mining method including stope sequencing**" and that it must "**describe measures planned and used to assess potential ground instability such as instrumentation and computer modelling**" (11). While many such sequencing plans have been developed, often with the assistance of external consultants, little follow-up has occurred to evaluate their success.

A review of both traditional and contemporary extraction sequences is provided in this thesis, along with the detailed analysis of a mine extraction plan over a five year period, including an evaluation of the use of both instrumentation and numerical modelling as required by legislation. This is seen as a significant advancement of the knowledge and understanding of the design and planning process of underground mining operations.

1.7 RESEARCH OBJECTIVES

The following objectives were selected for this research:

1. To determine the main factors which have governed extraction sequencing planning, within the various mining areas of Ontario.

2. To understand and to classify the sequencing strategies used in both historical and contemporary Ontario mines and how the strategies have evolved with the change to bulk mining methods.

3. To examine the various extraction sequences in light of how they meet the objectives demanded of a successful mining operation.

4. Through a detailed case study, at an operating mine, to evaluate the application of numerical modelling in planning the ground control aspects of an extraction sequence.

5. To evaluate the effectiveness of an instrumentation program in monitoring an extraction sequence and providing the data necessary to adjust the mining plan to meet the day to day demands placed on production personnel.

6. To provide a design rationale for the advancement of future mine sequencing planning, especially in the light of rapidly evolving extraction technologies.

1.8 METHODOLOGY

The research began by first reviewing the factors which influence the outcome of mining activities within the underground hardrock mining industry of Ontario. Site visits were carried out to all mining areas in Ontario and augmented by visits to mines in Manitoba and New Brunswick. In addition, a thorough literature review was carried out on both historical and contemporary mine sequencing activities. The varied approaches to extraction sequencing and the factors causing these variations were compared and contrasted to develop an understanding of the rationale involved in sequence planning.

The second step of the research involved finding and developing a case study where the extraction sequence could be planned and evaluated through monitoring of the rock behaviour. Two initial sites were chosen for this work but the very slow extraction rates involved at both sites and changes in mine personnel led to these locations being abandoned.

The eventual case study chosen was the David Bell mine, owned and operated by Teck-Corona Operating Corporation in the Hemlo area. This area was the most significant new mining area to be developed in Canada in several years and the presence of three operations on the one orebody, as described in Chapter 5, gave an excellent opportunity to compare and contrast sequencing planning. In addition, the mine was a new operation moving to a new portion of the orebody, a portion which would go to complete extraction within a five year operating period.

The author became involved in the planning process in May of 1988 although a visit to the mine was made in 1986 as the initial cut and fill stopes on the upper levels were being excavated as described in Chapter 5. Numerical modelling required during the initial planning of the main ore zone was carried out by the author in 1988 in conjunction with the mine's rock mechanics consultant. Ongoing modelling work, as the sequence was adjusted, was carried out directly by the author in conjunction with mine personnel. Most of the numerical modelling was carried out using the MINTAB displacement discontinuity code as adapted by CANMET, although the FLAC finite difference code from Itasca (12) and the Examine3D boundary element code from the University of Toronto was also used. Regular mine visits were conducted over a five year period to discuss the results of the modelling and to plan the extensive instrumentation program.

The instrumentation program initially begun by the mine in conjunction with its rock mechanics consultant was augmented under a mine funded research contract to the author in March 1991 and matched by the University Research Incentive Fund of the Ontario Government in October 1991. Undergraduate student research assistants were employed at the mine during the summers of 1991 and 1992 to assist with the rockmass characterization and data collection. The in-situ stress measurements were carried out by Mr Peter Lausch, a Queen's University technical assistant, and funded by the research.

All production statistics during the period as well as the data collected by mine survey personnel from the installed instrumentation were supplied by the mine. Deviations from the plan necessitated by operational constraints were discussed regularly during site inspections and by telephone.

Both the David Bell Mine and the interaction of the two other mines operating on the same orebody provided an excellent opportunity to plan, observe and document the extraction sequence. Also, the plan's interaction with the short range operational and safety constraints which are part of the day to day life of the operation could be observed and evaluated.

1.9 THESIS STRUCTURE

Chapter 2 of the thesis reviews the factors which have affected mining practice and sequencing philosophy in the various mining areas of Ontario and evaluates which factors have produced differences. Chapter 3 presents an analysis of the traditional sequences developed and used in these mining areas and how they were restricted by more traditional mining methods. Chapter 4 continues with a description of the sequences developed in conjunction with more mechanized or bulk mining methods. Both chapters provides examples from outside of Ontario where they are considered to be significant.

Chapter 5 describes the mining sequences used internally at the three mining operations in the Hemlo area and how the three mines interface on extraction planning. Chapter 6 gives a summary of the geotechnical information gathered to classify the rock mass and to serve as input for numerical modelling studies. The numerical modelling carried out as part of planning the extraction sequence at the David Bell Mine is described and evaluated. Finally a discussion of the limitations of the various numerical models used is given. Chapter 7 discusses data provided by the mine's comprehensive instrumentation program and examines the practical problems encountered and how mining practice or the extraction sequence was changed to overcome them.

Chapter 8 evaluates the economic sensitivity of a project to the various sequencing objectives, providing examples from Chapters 3 through 5 of how the move to bulk sequences has affected the ability to achieve them. The chapter also presents a classification of these sequences and discusses the possible implications of new technology on the sequencing options. Finally Chapter 9 presents the conclusions arising from the research and gives recommendations for future work to better evaluate the optimal planning and sequencing of an underground mining operation.

1.10 CONTRIBUTIONS TO ORIGINAL KNOWLEDGE

There has been very little written on the subject of optimization of mine design. Zambo (13) looked at optimizing the location of extraction entrances to minimize underground development; unfortunately this often gives rise to an entrance which is poorly placed geomechanically. Some work has been carried out on optimizing stope layout in flat-lying deposits (14), generally in coal or other soft rock deposits. Yi and Strugal (15) look at optimizing the cutoff grade for underground orebodies but the work simplifies the problem by assuming that there is only one fixed and one variable cost factor involved.

The bulk of recent Canadian mine design research activity has focused in three main areas. Firstly, in the area of stope design the work of Mathews as modified by Potvin (16) has concentrated on developing a design process for sizing stope openings. Secondly Bawden (17) concentrated on designing wall support leading to a handbook on support principles and requirements. Associated with this latter project, considerable work has been carried out to develop numerical modelling tools to help estimate stress behaviour, leading to the development of the Examine3D program (18). Other models such as BEAP were developed in conjunction with CANMET. Bawden et al. (19) mentioned stope sequences as part of the overall design process, but there is no description of any attempts to classify mining sequences or their interaction with the operational problems of day to day production.

This research is the first to examine both historical and contemporary mine sequencing strategies and to evaluate them not only in a geotechnical manner but in terms of the financial and operational objectives of the mining company. It provides the first published record of an extraction cycle which led, during a five year period, to one area of a large mine successfully reaching complete extraction. The work documents the role played by numerical modelling studies, by an intensive instrumentation program, and by adaptations in mining practice in helping to successfully advance the extraction sequence.

As research continues into additional automation of the underground mining process, it is important to understand the lessons of the past and problems of the present. This research is considered to be an important first step in understanding how to best manage mine extraction strategies to ensure the optimal use of the mineral resource.

CHAPTER 2

A REVIEW OF ONTARIO MINING ACTIVITY

To determine the main factors which have affected mine sequencing practice, this research concentrated on the hardrock underground mines of Ontario since over half of the Canadian mines in this category operate in that province. While that number may change frequently, at the time this project began there were around 40 such operations. Since the intention was also to examine historical practices, there was also a wealth of published information on previous operations in Ontario.

2.1 THE MINING AREAS OF ONTARIO

In Ontario, notable geographical clusters of mining activity exist, the Sudbury area for example, with over 30 of the 40 operating mines situated in six major mining areas. These same areas have also been the location of much of the historical mining activity. While variations in mining methodology may exist within these areas, common attributes often dictate similar general approaches to mining. In the Sudbury area, while the economic minerals and the attitude of the orebodies are similar, they range from the wide deposits of the Frood Mine to the narrow tabular veins of the Falconbridge Mine, to the complex lens type deposit at Strathcona. Yet within this wide range of orebodies, the stress regime, mining methods, the management of pillar removal, and the nature of most rockbursts are similar.

The initial stage of this research examined mining methods and sequences used in the six primary mining areas shown in Figure 2.1. After a thorough literature review, site visits to each area were carried out.

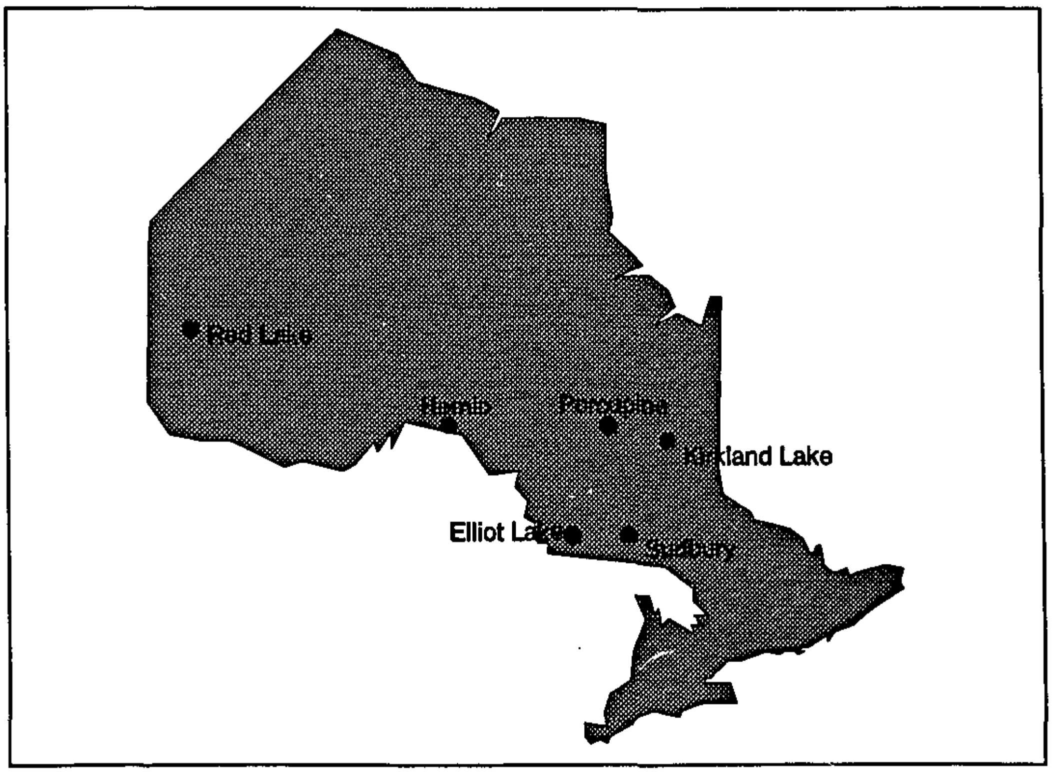

Figure 2.1 The Main Hardrock Mining Areas of Ontario

1. The Kirkland Lake Area

The Kirkland Lake area is defined as the gold mines which were clustered along the 5.5 kilometre length of the "main break" of the Kirkland Lake fault from the Macassa Mine on the west to the Toburn on the east. The Macassa operation of Lac Minerals Ltd. is the only active mine remaining.

2. The Porcupine Area

For the purpose of this research, comments are restricted to gold mining activity in the Timmins area. There has been a notable absence of rockburst activity in these mines, making an examination of the reasons for this absence important. Two mines, the Dome Mine of Placer Dome and the Pamore operation of Royal Oak Mines, continue to operate.

3. The Sudbury Area

The initial examination of the Sudbury region concentrated on the following three mines, thought to be representative examples of area mining:

(1) The Falconbridge Mine of Falconbridge Limited: a steeply dipping, narrow, tabular orebody with a long operating history. This mine was closed as this project began but much published information existed on the mine and the author had worked there in the summer of 1971.

(2) The Strathcona Mine of Falconbridge Limited which is at the northwest corner of the Sudbury basin.

(3) The Creighton Mine of INCO Ltd. which is an excellent case study of mining methods and changing technology since it began underground operations in 1904. It is also the deepest mine in the area and as such has developed novel mining methods to handle high rock stress.

In addition, site visits were later carried out to the South and Frood-Stobie mines of INCO Ltd. and to the Onaping, Lockerby and Fraser Mines of Falconbridge Limited.

4. The Elliot Lake Area

Mining in this area is confined to a tabular flat-lying orebody. While the bulk of mining in the area is relatively shallow, there has been an important and intensely researched series of rockbursting events. Until 1990, mining activity was carried out at three Rio Algom Mines and the Denison Mine.

5. The Red Lake Area

The Campbell Red Lake Mine and the Dickenson Mine carry out adjoining mining activitiesin the area. Rockbursting has been present since the 1960's and a major series of rockbursts in late 1983 at the Campbell property have resulted in important changes in the approach to mining activity in this area.

6. The Hemlo Area

The newest Ontario mining area, it is one of the most important gold mining areas in Canada with combined production of over 500,000 ounces coming from three underground mines exploiting a common orebody. While many features are common to the three operations, physical ore differences and management philosophy provide an interesting comparison of mine sequencing strategy among the operations. It is this area in general and the David Bell Mine in particular which was chosen as the prime case study for this thesis.

2.2 FACTORS FOR COMPARISON

Within each mining area a series of factors or circumstances influencing mining activity were chosen for review.

2.2.1. History

The history of a mining area is considered important as the time of mine development determines the mining methods and technology used. The mining sequence is influenced by one's ability to explore the orebody to depth and the materials handling systems available.

In the Kirkland Lake area, the Wright-Hargreaves was discovered in 1911, began production in 1921, and peaked at 1120 t per day in 1940. To the east, the Sylvanite and Toburn Mines were also brought into production.

To the west, the Lakeshore Mine's production peaked in the mid 1930's until rockbursting problems caused its production to decline. Production ceased in 1965 with the crown pillar mined in 1985 by Lac Minerals. Farther west, the Teck-Hughes Mine was developed in 1923 at the point where the orebody ceased to outcrop on surface. Still further west, drilling in 1924 by Kirkland Lake Gold Mines established the presence of ore at a depth of 550 metres. At the extreme western end of the ore zone, Macassa Mines was explored from the 750 metre level of the Kirkland Lake Mine and in 1934 a shaft was sunk to the 915 metre level. This is the only active mine in the area today. After over 20 years of development activity, the result was seven mines in operation along a 5.5 kilometre stretch of the main ore-bearing zone.

In the Porcupine area, gold was discovered in 1909 with the most notable deposits being the Dome, Hollinger and McIntyre mines. Over the years, up to 36 mines have operated in the area. While standard mining technology has been used in the area, rockbursting has not been a problem. The Dome Mine of Placer Dome Ltd. and the Pamore Division of Royal Oak Mines Ltd. are the only active mines in the area.

The first record of mineralization in the Sudbury district was in 1856 and in 1886 open pit mining began at Copper Cliff, followed by underground mining at Creighton in 1907. By 1900 all of the major INCO deposits in the immediate Sudbury area were known. In 1901 Thomas Edison outlined the Falconbridge orebody by a dip-needle survey but failed to reach it by a shaft in overburden. It was discovered after diamond drilling technology was developed by E.J. Longyear in 1916-17.

Mining development began in the Elliot Lake area in 1953 with capital expenditures of about $350,000,000 to develop a combined daily capacity of 31,300 tonnes. Development took place on 11 properties and involved the sinking of 19 shafts ranging in depth from 180 to 1160 metres. This activity is well documented by Morrison (20). With the completion of the contracts in the early 1960's and a decline in world prices for uranium, only two companies survived, Rio Algom with its Quirke, Panel and Stanleigh Mines, and Consolidated Denison. Only the Stanleigh Mine is still in operation after termination of long term Ontario Hydro contracts in 1992.

The first major find in the Red Lake area, the Howey Gold Mine, started production in 1930. Other mines such as Cochenour-Willans, Madson Red Lake, and McKenzie Red Lake, operated in the area starting in the 1930's. The Dickenson and Campbell Red Lake Mines began production in 1948 and are the only remaining operations.

The newest major mining area, the Hemlo area, was discovered in 1982 and began operation in 1985. The three operating mines are the David Bell Mine to the east and Williams Mine to the west, both owned by the Teck-Corona Operating Corporation, and the Golden Giant Mine of Hemlo Gold Limited in the central portion of the orebody. The orebody decreases in thickness from west to east and decreases in grade in the opposite direction.

Within the areas examined, with the exception of the Elliot Lake and Hemlo areas, mining had commenced by 1930. While improved diamond drilling technology would have allowed better pre-production orebody definition, other factors precluded extensive definition drilling. At Elliot Lake, for example, precise definition of the orebody would have been prohibitively expensive. In most room and pillar operations, the flexibility of the method allows the plan to deal with ore variability. In the Hemlo area, the rush to bring the mines to production resulted in minimal drilling definition, even to the point of a major production shaft being sunk through an adjoining company's orebody.

The final extraction sequence often developed from operational constraints rather than good planning. In the older mining areas the general sequence of mining from top to bottom was often based on a lack of knowledge of the orebody at depth. Final ore definition in many early mines was based on establishing a drift on the ore vein above and below the stope and driving a raise through the ore at a 50° angle. The stope was then ready for the start of production.

In terms of production technology, only the Hemlo area started out using bulk mining methods. All other areas have now adopted or adapted some bulk mining method, depending on the thickness and other parameters of the orebody. The use of shrinkage mining in the Kirkland Lake and Red Lake areas led to rockbursting problems, as discussed in Chapter 3, but technology has not directly affected ground control in the various areas. The most significant difference in mining practice among the areas is the amount of backfilling practised. In the Sudbury and Hemlo areas, backfill was used at an early stage, whereas in the other areas large expanses of open stopes were left.

2.2.2 Geology

In reviewing and comparing the geology of the various mining areas from a ground control perspective, the following points were considered important:

(1) The relationship between the host rocks and the ore.

(2) The degree of faulting which controlled the ore deposition and post-ore faulting, silicification or dike intrusion.

(3) The orebody depth, attitude, regularity in dip, strike, and thickness.

Kirkland Lake lies in a gold bearing belt which runs 250 kilometres from Swastika in Ontario to the Cadillac area in Quebec (21). Thomson describes the area as bounded on the north by mafic volcanics which to the south are interbedded with sediments and felsic volcanics. These volcanics occur in long narrow bands, have been heavily folded to a near vertical position, and have been faulted along strike both in a pre-ore and post-ore manner. From a mineralization point of view the following points are considered important:

(1) The mineralization shows a close relationship to structure. It occurs in a variety of rock formations but regardless of the rock type exhibits a direct relationship to the faulting.

(2) There has been pre-ore and post-ore faulting with post-ore movement found to occur on pre-ore faults. Pre-ore movement has created subsidiary fractures, drag folds and brecciated zones that are ore-bearing while the main fault in the area may be barren.

(3) The presence of a brittle competent rock adjacent to a fault or fold tends to be the best ore deposition location due to the increased fracturing. In some locations, rocks not normally brittle have been made so through pre-ore silicification.

The "main break" or Kirkland Lake fault is a thrust fault striking N 67^0 E with a steep southerly dip and vertical movement of up to 425 metres at the western end. Worked to a depth of 2500 metres at the Lakeshore Mine, the fault may occur as a single plane, as parallel planes a few metres apart or as fault planes up to 30 metres apart. Mining of these parallel structures and particularly of their junctions has caused many rockbursting problems.

The longest ore stoping length was on the 1000 metre level where the Teck-Hughes, Lakeshore and Wright-Hargreaves Mines shared 2000 metres of ore. The ore width averaged 2 metres but varied from a minimum of about 50 centimetres to a maximum of 30 metres at certain vein junctions.

The geology of the Porcupine area is characterized by the following:

(1) The orebodies occur as vein stockworks in fractures formed at the crests of intense folding.

(2) The various orebodies do not occur along a major structural trend but in fact are scattered along a complex fold structure.

(3) Even within individual mines such as the Hollinger or the Dome, the ore occurrences are a series of scattered pods as shown in Figure 2.1.

(4) The host rocks include mainly metamorphosed sediments which are described as schists. Dunbar (22), in describing the predominate host rock, writes: "The intensive fracturing caused by cross-folding superimposed on relatively porous and schistose rocks .... tended to create a spongy layer in which veins formed."

(5) The mineralization accompanied a hydrothermal solution which percolated into these fold induced openings. There was not the degree of silicification which occurred in Kirkland Lake.

In describing the Porcupine area in his 1942 report, Morrison (23) refers to the serpentine of the porcupine as "obviously a weak rock". He states: "In comparing such camps as Kirkland Lake and Porcupine, one can only suggest that elastic characteristics predominate in the rocks at the former as against plastic in the latter." Because of the softer highly sheared host rocks and the fact that openings of large areal extent along a continuous orebody were not present, rockbursting was not a problem.

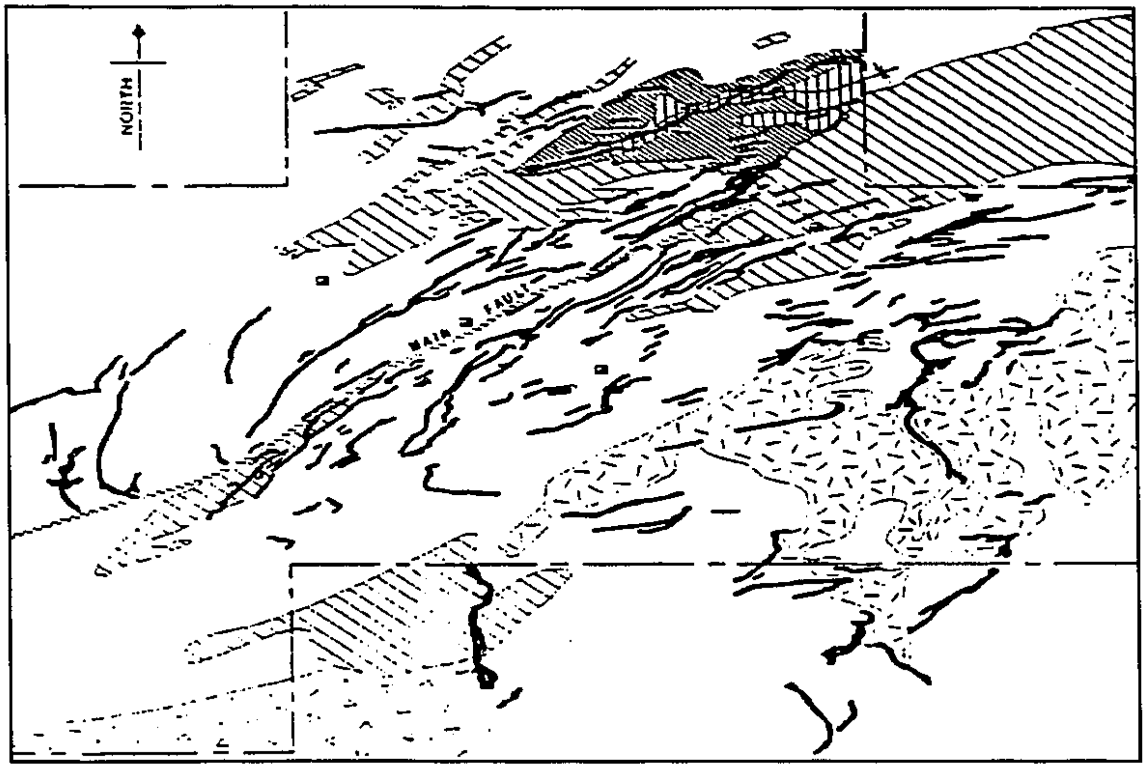

Figure 2.2 The Scattered Ore Pods of the Hollinger Mine (22)

Sudbury area deposits occur immediately adjacent to the rim of an oyster shaped norite, in quartz diorite and greenstones in three characteristic manners:

1. As disseminated ore confined mainly to the quartz diorite.

2. As massive sulphides, usually along a definite fault or breccia zone.

3. As an intricate pattern of sulphide stringers in various types of shattered and brecciated county rock.

The main structural system, the Creighton fault, runs along the south-eastern edge of the basin with horizontal displacement of up to 600 metres. Other numerous localized faults are associated with the various area orebodies.

The Falconbridge Mine ore zone was a near vertical sheet of sulphides approximately 1.6 kilometre long and over 1800 metres deep. The ore was a breccia cemented by a matrix of sulphides. The contact between the ore zone and the norite coincided generally with a shear zone known locally as the "main fault" composed of gouge material up to 1 metre in thickness. Where there was a deviation in the contact in dip or strike, tangential shears could break away from the main fault towards the north or south. Where these shears occurred in the greenstone, they formed the loci for "south wall" ore shoots which were bodies of massive sulphides lying south of the main ore zone. Where the shears broke away into the norite they resulted in localized widening of the ore.

The Creighton Mine lies on the outer rim of the norite basin at an embayment into the footwall rocks. The norite serves as the hangingwall with the footwall being metamorphosed volcanics and granite. Of interest is the presence of varying grades of ore with high grade massive sulphides occurring adjacent to faulting in the area. Disseminated ore grades away from the faults and into the quartz-diorite and the norite of the hangingwall.

The Strathcona ore reserves occur in three areas:

1. The Main Zone, 1000 metres on strike and 500 metres vertically from the 1475 to 3025 level. The dip is 60^0 to the south at the west to 30^0 at the east. Mineable widths vary from 3 to over 100 metres.

2. The Deep Zone which is approximately 150 metres in length, width and height from the 2250 to the 2750 level.

3. The Hangingwall Zone of scattered lenses parallel to the main zone and lying between the 2000 and 3150 levels.

The major feature of Elliot Lake geology is a large Z-shaped exposure of an unconformity between gently folded Huronian sediments and the highly folded and metamorphosed basement rocks. The northern portion of the "Z" is a synclinal fold plunging about 5-10 degrees to the west. The Quirke, Panel and Denison Mines operated on the north limb of the syncline where economic uranium had been outlined over an area of about 4 kilometres on strike and 8 kilometres down dip. On the south limb the Stanleigh Mine is now operating.

Mineralization occurs in a quartz pebble conglomerate reef which varies widely in dip on the individual properties. In some locations, at Denison for example, a second ore reef occurs about 45 metres above the lower reef. The ore horizon is intersected by numerous faults and dykes which can be up to 30 metres wide and persistent on strike. Important operationally, is the presence of numerous, well developed joint sets which have resulted in many wedge failures of the stope backs and in difficulty to calculate true pillar strengths.

As reported by Horwood (24), the Red Lake Basin is a huge remnant of volcanic, intrusive and sedimentary rocks, about 60 km long from east to west and up to 37 km in width, which was intruded by granitic masses. The volcanic and sedimentary formations are heavily folded. The fractures and shear zones are thought to be related to compressive stresses caused by the intrusion of several granitic masses and occur within various rock types.

Several ore zones exist at the Campbell Mine as follows:

1. The A zone is a fracture filled vein from 30 cm to 2 m wide, 450 m long and presently developed to the 920 m level. It terminates against an altered zone to the west but continues east to the Dickenson Mine.

2. The F2 zone is a continuation of the A zone to the west of the altered area. An offset of this zone to the south is known as the F zone.

3. The L zone is a series of veins formed in altered rock. The main portion, striking northeast is a replacement type vein containing varied sulphides and free gold. The zone is over 15 metres thick and 60 to 100 metres long with smaller veins perpendicular to the main zone.

4. The G zone is similar to the L zone. It is developed to 920 metres, is 10 to 15 metres wide and up to 200 metres long.

The A and F zones are narrow but of large areal extent, contained within the andesite, and have undergone fracture filling by the gold quartz solutions. The replacement L and G zones have greater thickness and reduced areal extent and are also associated with altered chloritic rock.

The Hemlo area rocks are part of an Archean age greenstone belt which hosts several gold deposits. The ore is found near the bottom of the Heron Bay group in the south limb of the Hemlo Syncline which formed during deformation and metamorphism of the original volcanic-sedimentary rocks. Important to the mine design and sequencing activity in the area is the presence of up to 5 joint sets, the highly altered rocks which may include sericite bands of up to 15 to 20 mm in thickness, and the presence of barite in the ore zones. At Golden Giant, for example, the ore reserve was published as containing up to 13% barite. These points will be covered in more detail in Chapters 5 through 7.

The main Hemlo orebody, while somewhat irregularly shaped on longitudinal section, is a continuous sheet of approximately 2500 metres strike length and extending from surface outcrop on the Williams property to near 1000 metres in depth. It strikes approximately 100^0 to 110^0 with a dip of 40^0 to 70^0 to the north. It varies in thickness from approximately 2 metres on the east to 45 metres on the west.

Rrom the examination of the geology of the Porcupine area, it can be seen that the ore occurs in a series of scattered pods which reduces the areal extent of the excavation on any pod. In addition, the schistose rocks act as a stress relief shadow. The mining sequence was not critical as the various pods were discovered and mined. No serious rockbursting has occurred in the area and, even though much of the mining was completed before numerical modelling was available, the existing operations have not become greatly involved in research in this area. Because of the differing geology and lack of rockbursting, no further examples from the area are included in this study.

The Kirkland Lake and Red Lake areas are both structurally controlled, have zones of significant areal extent, are contained in brittle rocks, and both have had post faulting and silicification of the ore zones. Both mining operations deal with a "stiff" mining environment and they have experienced significant violent ground failure.

In the Sudbury area both stratigraphic and structural controls exist. All orebodies occur along the norite greenstone contact, but the exact nature of the individual orebodies varies, depending on local faults or shear zones.

Both the Elliot Lake and Hemlo areas have strong stratigraphic controls and the mining activity is affected by strong joint patterns which can cause back instability. The nature of the orebearing material is completely different, the Elliot Lake pebble conglomerate being very brittle and the barite bearing Hemlo ore being relatively soft. Elliot Lake is also a "stiff" mine whereas the Hemlo rocks are much more compressible.

2.2.3 The Geotechnical Environment

The geotechnical environment is defined as:

(a) The physical characteristics of both ore and host rocks including their strengths and elastic moduli.

(b) The in-situ stress field, including the stress gradient with depth, the ratio of horizontal stress to vertical stress, and the orientation of these stresses to the orebody.

For the reasons mentioned in the previous section, no geotechnical parameters from the Porcupine area have been included.

At the Macassa mine, in-situ stress measurements conducted by Arjang and Vaillancourt (25) show the maximum principal stress to be in the NNE-SSW to NW direction and perpendicular to the orebody. The intermediate stress is parallel to the orebody, while the vertical stress is lower. The general relationship is given by:

Vertical Stress (MPa.)	=	0.026 MN/m x depth
Perpendicular Stress Horizontal	=	1.56 to 1.68 x vertical
Parallel Stress Horizontal	=	1.00 to 1.27 x vertical

The brittle nature of the rock in the Kirkland Lake area is demonstrated by the rock strengths of the Macassa Mine published by Cook and Bruce (8) given in Table 2.1.

TABLE 2.1

MACASSA MINE - PHYSICAL ROCK CHARACTERISTICS

ROCK TYPE	COMPRESSIVE STRENGTH (MPa.) UNIAXIAL	TRIAXIAL	ELASTIC MODULUS GPa.
PORPHYRY	227	386	69
TUFF	193	386	82
BASIC SYENITE	145	317	62 +

Rock strengths of the Elliot Lake area were examined in detail by Morrison et.al. (20). A summary of these results for the ore is given in Table 2.2. The high deviation results from the individual pebbles present in the test samples.

TABLE 2.2

ELLIOT LAKE - PHYSICAL ROCK CHARACTERISTICS

	NORTH ZONE	SOUTH ZONE
UNIAXIAL COMPRESSIVE STRENGTH	MPa.	MPa.
AVERAGE VALUE	195	160
MINIMUM VALUE	51	72
MAXIMUM VALUE	408	280
STANDARD DEVIATION	60	54

ELASTIC MODULUS 29.3 - 91.3 GPa.

POISSON'S RATIO 0.10 - 0.20, average 0.17

Stress studies show horizontal stresses to be well above the vertical stress, which is close to the gravitational forces of the overlying rocks. Stress determinations at the Stanleigh Mine, Arjang (26), are summarized as follows:

STRESS VERTICAL	= 0.0263 MN/m x depth (m)
STRESS HORIZONTAL ALONG STRIKE	= 2.0 x vertical
PERPENDICULAR TO STRIKE	= 1.4 x vertical

At the Campbell Red Lake mine, stress measurements were carried out by CANMET and the results from Arjang (27) are summarized in Table 2.3.

In addition to this work, stress measurements were carried out in different rock types along the 14 level to determine if the stress field was equivalent for all rock types or if, in fact, some formations acted as stress concentrators. The results showed equivalent stress for all rock types.

TABLE 2.3

STRESS MEASUREMENTS AT THE CAMPBELL MINE

Depth(m)	Horizontal		Vertical	Ratio	
	Normal MPa.	Parallel MPa.	MPa.	Vert.	Parallel
625(14L)	27.0	16.0	16.0	1.7	1.0
1000(22L)	54.0	24.0	26.0	2.0	0.9
1220(27L)	73.0	43.0	32.0	2.3	1.3

Rock properties from the Campbell mine were summarized by Makuch (28) from test results obtained from EX core and six inch overcore samples. The results are given in Table 2.4.

TABLE 2.4

ROCK PROPERTIES FROM THE CAMPBELL MINE

Rock Type	Sample Size	UCS MPa.	Modulus GPa.	Poisson's Ratio
Andesite	EX	33.0 to 41.0	82.4	.21
Andesite	6"	N/A	85.0	.20
Chloritic Rock	EX	17.0	60.0	.16

The geotechnical parameters are quite varied among the deposits in the Sudbury area depending on the rock types present in the hangingwall and footwall. The strike of the orebodies is quite varied as they lie around the perimeter of the basin, yet with the exception of the Creighton Mine, the maximum principal stress is roughly perpendicular to the orebody. At Creighton the maximum principal stress is parallel to the orebody although it approaches being hydrostatic at depth (29).

Table 2.5 gives the parameters used in recent Examine3D modelling by the geomechanical design group at Queen's University. The high modulus shown for the Lockerby Mine is based on the footwall granites but it must be remembered that a shear zone of several metres in thickness exists in the hangingwall of the ore zone so the true in-situ modulus is difficult to quantify.

TABLE 2.5

SUDBURY GEOTECHNICAL PARAMETERS

	Strathcona	Lockerby
Vertical Stress (MN/m x depth)	0.025	0.0198
Maximum Horizontal:Vertical	1.68	2.09
Intermediate Horizontal:Vertical	1.43	1.57
Young's Modulus	42,000 MPa	65,000 MPa
Poisson's Ratio	0.30	0.22

The initial modelling carried out by the author for the David Bell Mine was provided internally from in-situ measurements taken at the Williams Mine and at the Golden Giant Mine by Golder Associates, The normalized stress state, discussed further in Chapter 6, is as follows:

Vertical Stress (MPa.)	= 0.024 MN/m x depth
Perpendicular Stress Horizontal	= 1.8 x vertical
Parallel Stress Horizontal	= 1.15 x vertical

The rock parameters used in the modelling were as follows:

	Initial Models	Later Models
Young's Modulus	25,000 MPa	24,000 MPa
Poisson's Ratio	0.25	0.17

The initial values were based on a weighted average of the rock values assuming the presence of baritic zones as discussed further in Chapter 7. The values used in the later models are those used after on-site model calibration was carried out. Detailed parameters for the Hemlo area are given in Chapter 6. In summary, the stress regime at Hemlo is typical of the Canadian Shield and the modulus of the rock is greatly reduced by sericite zones in the hangingwall and baritic zones in the ore material.

From this data it can be seen that the general order of magnitude of stresses are as expected in all mining areas. The rock properties of the Kirkland Lake, Elliot Lake and Red Lake areas, as dictated mainly by the elastic modulus, are such that they can be expected to fail violently. As such, mining operations in this area are dealing with "stiff" rocks where stresses have to be managed on a very local scale. The Hemlo and Sudbury areas have lower modulus rocks and a different stress management procedure can be expected to be successful in these areas as will be shown by examples in later chapters.

2.2.4 Orebody Economics

Ore value determines the pre-development exploration and development which can be justified. It determines the mining method used and the degree of extraction which is attempted. Variations with time as markets or grade fluctuate affect the consistency of the mining approach.

The gold mining operations went through the very lean post-war period when gold prices in Canada were fixed at $35.00 per ounce ($1.09 per gram) but production costs increased dramatically. During this period, only the higher grade operations survived by minimizing production costs.

For the gold operations in the Timmins area, grades varied dramatically in various ore zones. The Hollinger operation had certain very high grade ore zones. The Dome Mine continued operations in high grade areas but avoided lower grade material. Low grade ore around the shaft collars, for example, is now being mined by open pit methods with a much larger open pit operation now in the final design stages. The Pamore division continues as a low grade operation, but struggled through low gold prices. The spotty nature of the ore zones, as described above, allowed the selective mining of higher grade zones without serious effect on mine sequences.

In Kirkland Lake, the Lakeshore Mine continued operations until the high cost of production shut them down in 1965. The grade of the Lakeshore crown pillar mined in the 1980's by Macassa was reported to be around 17.2 grams per tonne. With the exception of the Macassa operation, all other mines in the area were shut down but some exploratory work was carried out in the late 1980's to see if the material left behind could justify reopening the operations.

Ore grade at Macassa has been remarkably close to 14.3 grams per tonne for almost 60 years of operation. During periods of low gold prices, lower grade areas were not mined thus decreasing the overall extraction and reducing the rockbursting potential. The very irregular stope outlines produced by the close attention to grade in every cut, however, certainly increased rockburst potential. In fact, the first rockburst reported by Cook (8) was in a small tear-drop pillar left behind in a shrinkage stope, presumably due to the grade.

Campbell Red Lake Mines Limited's production to the end of 1990 was 10.7 million tonnes of ore containing over 224 million grams of gold; in other words, the production grade has averaged just over 20.9 grams per tonne during the life of the operation. Estimated ore reserves as of 1991 were reported at 4.9 million tonnes grading 16 grams of gold per tonne. The decision to mine at the orebody average was a management decision as the narrow vein A and F2 zones were above the orebody average and were mined at a rate to maintain the average.

When we later examine the extraction sequence followed in these two high grade zones, it must be remembered that the grade would certainly have justified any attempts to ensure total extraction. It must also be remembered, however, that at a time of low gold prices the introduction of more expensive techniques involving fill into these zones would have increased the overall mining cost and consequently the overall mine cut-off grade.

Sudbury ore grades vary from 1.5% to 3.5% nickel with around 1% copper. By-product minerals include cobalt, gold, silver, platinum and palladium. While there is considerable grade variation, it must be remembered that the large proportion of the free world's production which came from the Sudbury mines for many years led to price stability. There is little evidence that the variations in grade caused much of a variation in extraction sequences. The main effect was that because of the disseminated ore in the hangingwall, the cut-off was varied with price or mining costs. Examples of this are given in discussions of the Strathcona and Onaping mines in Chapter 4.

Elliot Lake experienced a boom period when uranium prices soared in the early 1950's, then experienced controlled growth and stability until the recent expiration of long term supply contracts. With ore grades of 0.10% to 0.20% uranium, in order to remain economically viable, high productivity primary extraction was necessary. No exacting mining sequence was present at most operations except the maximization of extraction as the operations proceeded to mine down dip from the access point. Design control concentrated on pillar design which saw rib pillar alignment generally parallel to the dip. Exceptions to this were the square pillars on the Denison property and the elongated pillars at Quirke which were oriented along strike, leading to regional collapse as documented by Hedley (6). Regional pillar stability was built into the Stanleigh Mine design but the overall extraction and costs have suffered.

The geological grade of the Hemlo orebody, as provided in annual reports, varies from 5 grams per tonne gold at the Williams property on the west, through 10 grams per tonne on the Golden Giant property, to about 25 grams per tonne at David Bell. In fact, certain mining periods have averaged from a low of approximately 3.5 grams per tonne at Williams to a high of over 30 grams per tonne at David Bell. All of this ore justifies extraction at today's prices and the mining sequences are discussed in detail in Chapter 5.

2.3 CONCLUSION

Ontario has had a rich history of mining since the turn of the century and mining remains an important contributor to the province's economy. Much of the activity has been centred in six major mining areas scattered from the eastern to the western extremities of the province. This activity also represents a major portion of the mining activity of Canada.

A comparison of the factors influencing mining shows that the evolution of mining technology, while concentrated in the Sudbury area, has rapidly spread to all areas of the province. The greatest technological improvements have been in ground support, backfill placement, mechanization, the level of knowledge and understanding of rock reactions to openings, and in new bulk mining methods which have improved the level of productivity in the industry.

In the geotechnical area, much of the stress regime is predictable within the accuracy required for initial design by using data compiled from numerous measurements over the past fifteen years. One major geotechnical factor causing differences in the mine extraction sequences in the various areas is the brittleness of the rocks, generally provided through measurement of the elastic modulus. The presence of sheared material in the Porcupine area has greatly reduced the importance of sequencing to the extraction plan so it was excluded from further study. The brittle rocks of the Kirkland Lake and Red Lake areas have caused mining practices to differ from those of the Sudbury area.

In looking at the economics of the areas, all but the Elliot Lake area have ore values to justify the costs to effect a high level of extraction where the orebody is continuous. The lower value ore and the flat-lying orientation of the Elliot Lake area greatly restricted what could be achieved in optimizing the extraction sequence and the area was excluded from further examination. The subsequent research focused on the remaining four Ontario mining areas and the development of a case study of the newest, the Hemlo area.

CHAPTER 3

TRADITIONAL MINE EXTRACTION SEQUENCES

One objective of this research was to review and classify both historical and current mine sequencing practices and to understand how they have been affected by changes in mining methods. Separating historical sequences from those used today is difficult since, in some instances, today's sequences are similar. Separating the sequences based on narrow vein or bulk methods is complicated by the fact that longhole or bulk methods are being adapted to more narrow ore. The idea of separating on the basis of stiff versus yielding sequences was considered, but again a clear cut demarcation is difficult.

It was decided to use the term "traditional" to separate past practices from the contemporary sequences described in Chapter 4. There is an overlap in time as mines continue to use established mining methods; two mines in Canada continued to use square set stoping until the early 1990's. These traditional methods tend to be based on narrow vein mining of higher modulus ore, although the Sudbury area can be considered an exception. Changes in technology have decreased the development time to advance tracked headings, using hand-held drills, versus trackless headings using a drill jumbo. There was increased sequencing flexibility available from the historical practice of providing lateral ore transport on each level versus the present norm of reducing the number of full service levels for sublevels where ore removal may or may not occur. This is significant because it leads to more dependence on ore passes located in the vicinity of the ore blocks.

In May 1992, the author visited phosphate mines which have been in operation since the turn of the century on the Red Sea coast in Egypt. An unsuccessful attempt had been made to operate a Dosco roadheader in a mine where materials handling is still by hand-pushed rail cars, pointing out the difficulty of changing one aspect of mining practice independently of others.

It is equally difficult to evaluate or classify mine extraction sequences totally in isolation of this overall provision of services. Under this constraint, the terms traditional and contemporary seem appropriate to describe the evolution of overall mining practice in hardrock mines during the past 60 years.

3.1 LONGITUDINAL SEQUENCING PRACTICES

In examining traditional narrow vein practices it is difficult to separate "in stope" sequencing practices from "extraction" sequencing practices. Much of the planning was carried out with little advanced information on the orebody. The traditional approach was to progress downwards from the top of the mine with development preceding production by a few levels. Ore was often defined by in-ore level development above and below a stope and a raise connecting the levels. This raise was often driven at an oblique angle to achieve an angle of 50^0 on "apparent dip", allowing it to be driven as an open raise and avoiding the requirement to drive a separate two compartment timbered raise. The oblique angle between these raises and the advancing stope cut created the ideal geometry to initiate rockbursting.

The term "narrow vein" is a relative one and as such has many interpretations. In choosing a mining method, the thickness of the orebody is critical. In more narrow ore the choice has traditionally been shrinkage mining. As the ore widened, cut and fill was the predominant method, with the degree of mechanization increasing in wider ore. Recently, ore with regular outlines has been successfully mined by longhole methods using closely spaced sublevels down to widths approaching one metre . Longitudinal extraction sequences, however, remain in effect until the ore becomes too wide to be mined by longitudinal stoping layouts. This critical distance, generally around 10 metres, is the width at which the ore is not easily supported over it's width, at which point transverse mining sequences are used.

The first major Canadian publication on mine sequencing was by Morrison in 1942 (23). He had returned from the rockburst prone Kolar Gold Field in India and was asked to study the problem in Ontario mines. The major problems at that time were in Kirkland Lake and in Sudbury where over 120 "heavy" rockbursts had occurred between 1934 and 1940. To overcome the problem, he recommended a "coalescence" of openings to form a dome. His "Dome Theory" became the classic sequence adopted by the Lakeshore Mine, several Sudbury operations, the Campbell Red Lake Mine, and is reflected today in the sequence adopted at the Golden Giant operation in the Hemlo area.

MacMillan and Ferguson (30) introduced some principles of stope sequencing for ground control in longitudinal stopes. Figure 3.1 represents a 5 m thick orebody 275 m high by 225 m on strike. They suggested dividing the strike length into five 45 m cut and fill stopes along strike with a production capacity of about 1800 tonnes per month each, thus requiring about 10 years to mine the orebody. The objectives of the suggested approaches were to minimize horizontal sill pillars and reduce level maintenance.

The "flat back" method shown in Figure 3.1(a) has the advantage of allowing all stopes to begin at the same time. The flat front of advance, however, has often proven unstable. In addition, the scheduling of blasting and filling would be very difficult as individual stopes approached each other.

The staggered sequence shown in Figure 3.1(b) reduces the interference but delays the second and third stopes until the first has advanced several cuts,

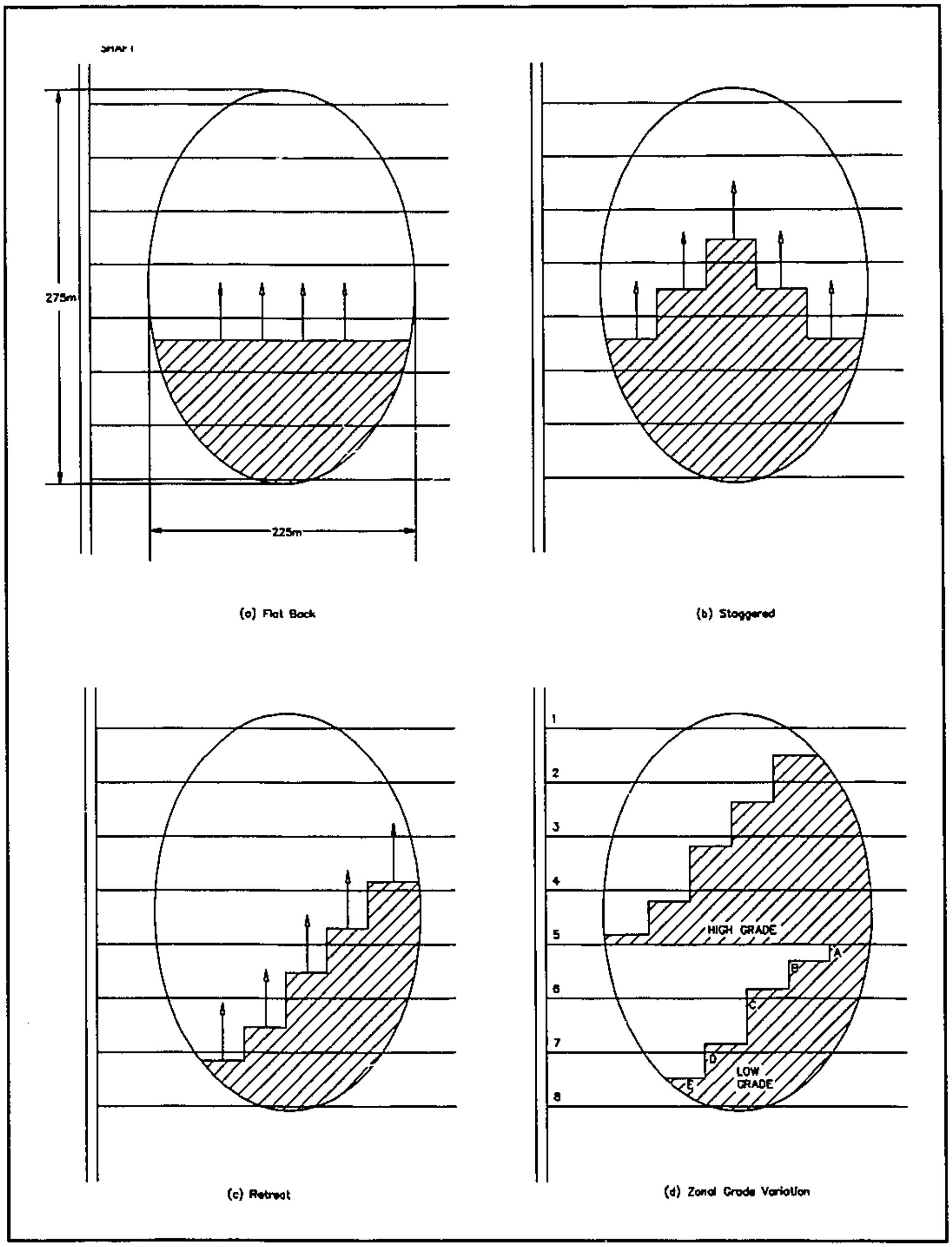

Figure 3.1 Alternate Stope Extraction Sequences (30)

similarly delaying the final two stopes. Each level would have to be maintained through the mined out areas to the orebody extremities or an off-ore haulage established. The retreat sequence shown in Figure 3.1(c) eliminates this problem but now delays the stopes on an individual basis. The sequence shown in Figure 3.1(d) accounts for a zonal variation in grade and increases the productive capacity. It will not double, however, as more expensive and less productive methods are now required to deal with a horizontal sill pillar.

The general rules listed by MacMillan and Ferguson were as follows:

1. Establish as few pillars as possible. Commence mining on as few levels as possible within the limitations of maximizing the production rate consistent with safety, irregularity in the orebody, available shaft depth, and/or a financial or time limitations.

2. Take advantage of barren or low grade material to establish permanent pillars, which should be oriented according to stress direction.

3. Attempt to maintain a retreat sequence to reduce level maintenance and avoid bad ground conditions.

4. Simulate the position of the stope faces throughout the extraction.

5. Schedule pillar extraction as soon as possible.

6. Correlate geological structures with stope and overall geometry.

These rules and the preceding discussion emphasize the critical interaction between stope sequencing, productive capacity, grade control and overall mining costs. The flexibility and increased productive capacity afforded by bulk methods makes the first rule more achievable. The larger openings caused by the delayed fill cycle have reduced flexibility thus making rules 3 and 5 more critical. The increased use of numerical modelling helps in achieving rule 4, with the use of the modified Matthew's analysis helping to achieve rule 6.

Examples from four Ontario mining operations will now be presented to better illustrate historical narrow vein practices. The first three are from mines with very brittle rocks where stress relief generally occurs violently through rockbursting. The final example comes from a mine with more forgiving rocks where undercut and fill mining was used successfully to mine failed sill pillars.

3.1.1 The Lakeshore Mine

The rockbursting problem at the Lakeshore Mine was described in 1942 by Morrison (23) and in 1946 by Robson (31). Morrison and Robson recognized three factors which are favourable to rockbursting: depth (high stress), a high percentage of extraction on longitudinal section and brittle rocks. All three of these factors were present in Kirkland Lake, particularly at the Lakeshore Mine which operated from 1914 to 1965 and achieved a depth of 2500 metres. The brittle nature of the rock was discussed in Chapter 2 and demonstrated by the rock strengths from the neighbouring Macassa Mine in Table 2.1. The effect of depth on stress as measured at Macassa by Arjang (25) was also outlined in Chapter 2 and showed the maximum principal stress to be perpendicular to the orebody.

The extent of extraction achieved early in the Lakeshore operation and adjacent mines is shown in Figure 3.2. Morrison noted a lack of rockbursting to the east where waste zones act as barrier pillars separating mining activities from the extensive excavations.

The first rockbursting at Lakeshore was in May 1932 when a floor pillar burst in a drift on the 1200 foot (365 m) level. In 1934 bursting assumed "noteworthy proportions" according to Robson. Three types of rockbursting and the actions taken to prevent it are summarized from Robson as follows:

(a) A series of floor pillars from the 2200 to 2575 horizons.

> "Mining down to the 1600 level was by shrinkage stoping and, although attempted, backfilling was difficult and incomplete. Below that, mining was converted to horizontal cut and fill using unconsolidated rock fill. Mining above the 1800 and 2000 levels was completed without incident but above the 2200, 2325, 2450 and 2575 levels bursting started when within 12 metres of the level. These series of long horizontal sill pillars were then divided into smaller pillars by driving raises and mined by a series of short rills. This gave rise to a large number of isolated pillars which were being reduced in size and eventually severe bursting occurred. Floor pillars eventually burst over a distance of 75 metres along strike."

These sill problems are similar to problems encountered in many mining operations where flat upward-advancing cut and fill mining has been employed.

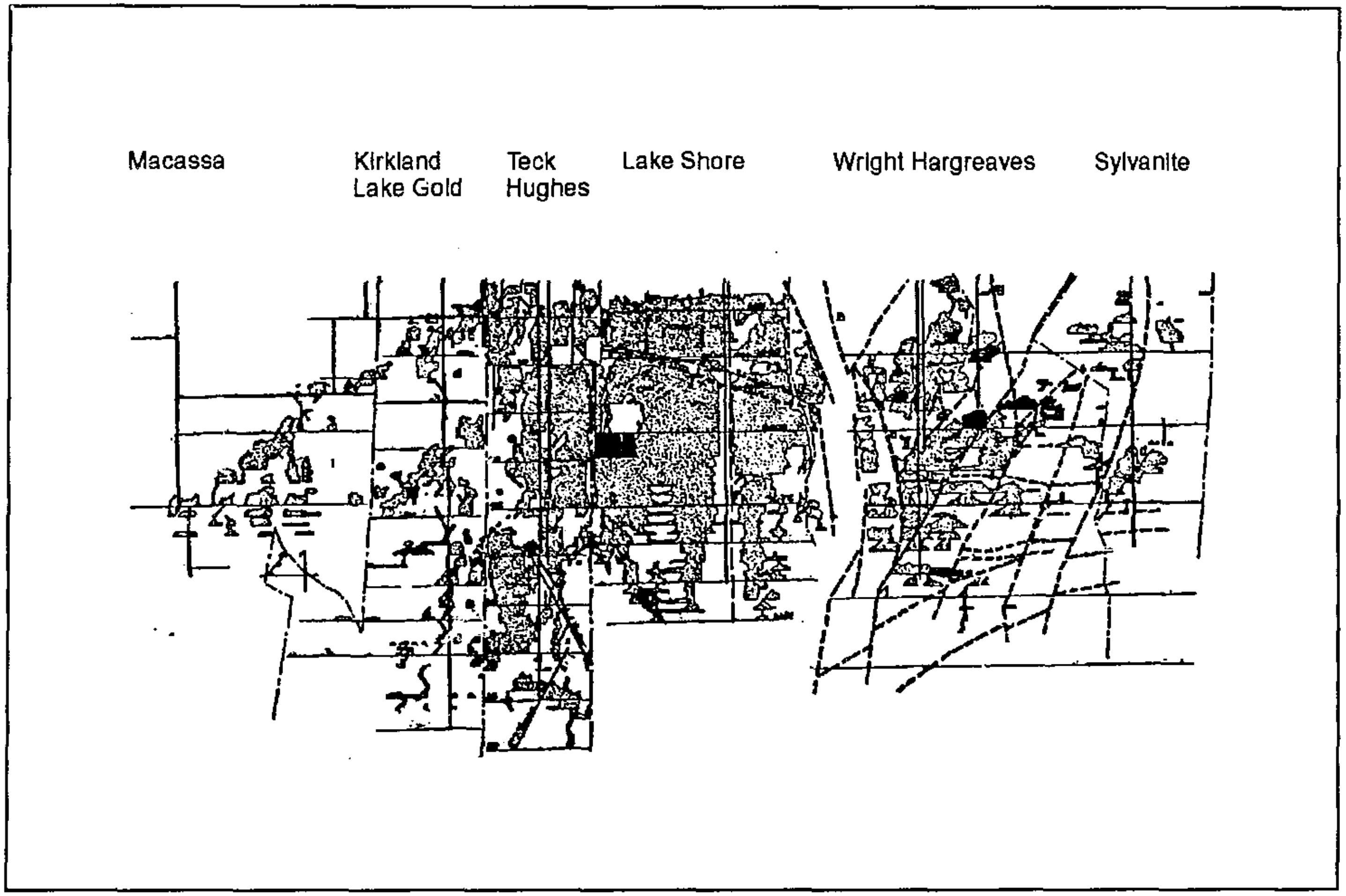

Figure 3.2 A Longitudinal Section of the Kirkland Lake Mining Area, from Morrison (23)

(b) Parallel and branching veins from the 2700 to the 3825 foot (825 to 1165 m) level.

"In the southeast part of the mine two parallel vein structures, the No. 1 vein to the south and the Middle vein to the north, separated by from 5 to 21 metres, ran from the 1600 level to the 3700 level at which point they merged, as shown in Figure 3.3. When mining these structures simultaneously serious bursting occurred. Some relief was accomplished by mining the No. 1 or hangingwall vein in advance of the main vein. In one section on 3450 level 23 bursts occurred and all but six were major bursts. At that point the No. 1 vein had not been mined but the level drifts had been established; in two bursts 18 m of drift were completely closed and damage occurred over 180 m."

As can be seen, the relief achieved by mining the No. 1 vein first was provided by stress shadowing but the severe problems occurred in the Middle vein while mining in the abutment of the large area opened up on No. 1 vein. The use of stress shadowing to prevent stress related bursts has more recently been tried at Campbell Red Lake Mine and the effects will be interesting when the overlapping C zone ore will be mined at the David Bell Mine.

(c) Footwall side of ore zone 3075 level to 4325 level.

"As mentioned in (a), the horizontal square-set mining used above the 2575 level resulted in sill pillar bursts. Below 2575 a steep square set rill system was employed with a planned stoping sequence from raises 125 and 365 metres west of the main cross-cuts. As mining approached the 3825 level it was anticipated that the vertical pillar of decreasing size

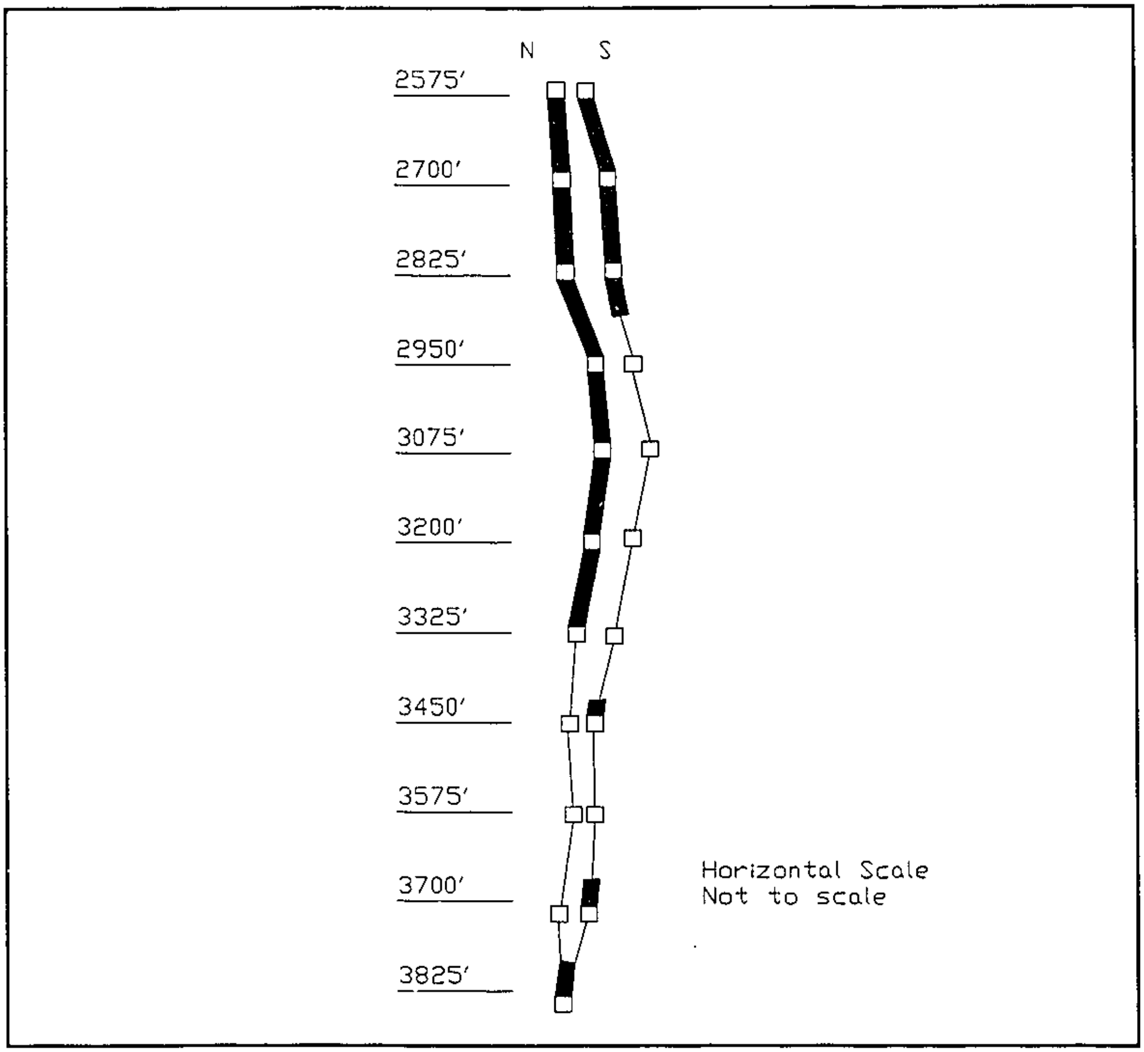

Figure 3.3 A Cross Section of the Lakeshore Mine (31)

was going to become a problem. Below that level, therefore, the sequence was reduced to one stoping area from a raise mid-point in that ore shoot. Because there was a branching of veins below 3825 level, a 250 foot pillar was left from 3450 to 3700 level. Even this pillar did not prove to be adequate as severe bursting occurred from 3825 level to 3950 level with subsequent extension upwards to 3075 level and downwards to 4325 level."

These steep rill or cascading sequences are shown in Figure 3.4. At depth this later single raise sequence was carried out without any major problems. The fact that this reduction in working sites was required at depth to control rockbursting necessitated a reduction in production tonnage.

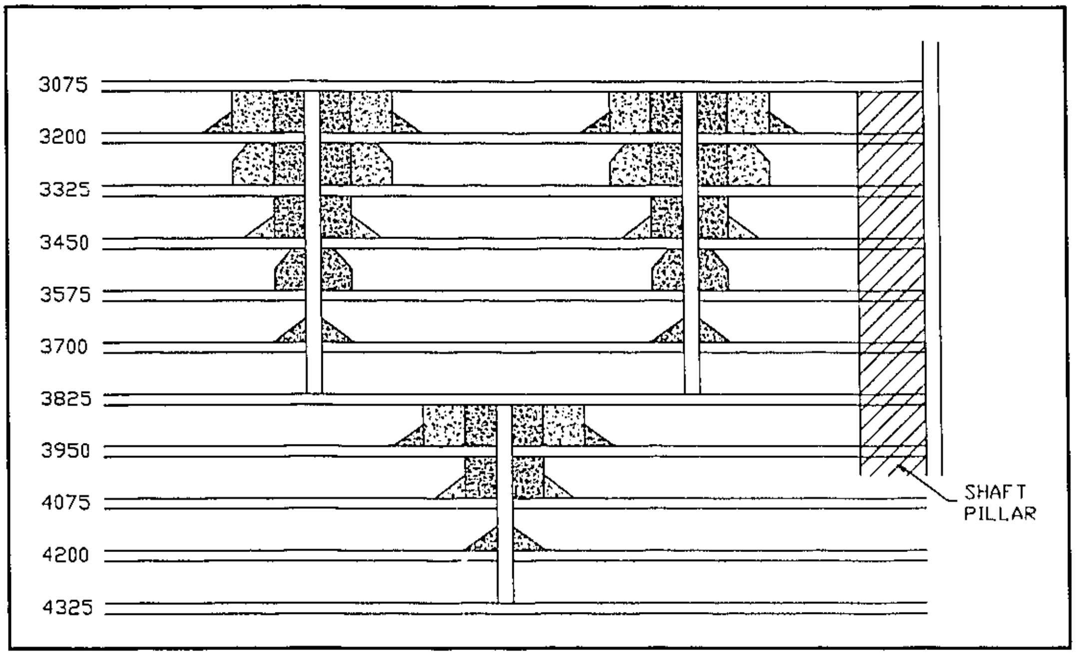

Figure 3.4 The Steep Rill Mining System at the Lakeshore Mine (31)

The initial daily production at Lakeshore exceeded 6000 tons but was reduced to about 3000 tons during the last 25 years of its operating life. While other factors may have contributed to this reduction, the reduced number of working areas was a major factor.

In 1936 Weldon published a set of rules to control rockbursting, stemming from the Lakeshore experience (32), summarized as follows:

1. "Bursting often occurs at the time of, or closely following, blasting.

2. Complete extraction of the ore is desirable, and the leaving of small pillars, whether in ore or waste, is to be avoided so that a gradual and uniform subsidence of the hangingwall may take place.

3. Horizontal pillars are a menace and must be avoided, or removed as quickly as possible.

4. Mining out a remnant from two directions is not good practice, regardless of the form of the triangle established.

5. Steep rills avoid the formation of horizontal remnants and hence are least susceptible to bursts.

6. Long rills, extending from level to level, and advancing in the direction of unmined ground, are less susceptible to bursts then those mining out a remnant.

7. The method of mining vertical cuts in short sections is useful in particularly heavy ground, but is higher in costs.

8. Bursts at either side do not transfer through the shaft pillar, nor in most cases do they occur close to it.

9. No bursts have occurred in the vertical pillars formed by the Teck-Hughes boundary and the diabase dyke."

Many of today's ground control practices, for example, the usefulness of mining to an abutment or creating a regional boundary pillar, can be recognized in these principles. It is interesting that we have had to re-establish these principles subsequent to the high incidence of rockbursting which occurred in the early 1980's in Ontario mines.

In summary, rockbursting occurred at the Lakeshore Mine at relatively shallow depth. Pillars of strong, high elastic modulus rocks caused large stress concentrations in pillars near openings of large areal extent. This was controlled at depth by a mine sequence which reduced the available stoping areas and the mine output. Even at that time, suggested research into these problems included seismic monitoring systems, closure monitoring and attempting to monitor the frequency of microseismic events for rockburst prediction.

3.1.2 The Falconbridge Mine

Development of the Falconbridge Mine began in 1928. The following schedule of shaft deepening activity demonstrates the traditional sequence of mining from the top down with development just preceding production.

Number 1 Shaft

Completed to	1000 ft (305 m) level	1929
Deepened to	2100 ft (640 m) level	1937
Deepened to	2800 ft (850 m) level	1945

Number 5 Shaft

Completed to	1400 ft (425 m) level	1934
Deepened to	2450 ft (745 m) level	1938
Deepened to	3150 ft (960 m) level	1942
Deepened to	4347 ft (1325 m) level	1953

Number 7 Shaft

Collared on 2450 ft (725 m) level 1948

Completed to 4025 ft (1225 m)level 1950

Number 9 Shaft

Collared on 3850 ft (1175 m) level in 1956. It was completed below the 6200 ft (1890 m) level, the deepest development of the mine.

As described in Chapter 2, the main ore zone is a near vertical sheet of sulphides approximately 1600 m long and over 1800 m deep. The north wall is norite and the south wall metamorphosed greenstones. Faulting has been very important in controlling the mining sequence at the mine. The controlling feature of the orebody, and the contact between the ore zone and the norite is a shear zone known as the "main fault". It is composed of gouge material up to 1 m thick and in places is present on both sides of the ore breccia.

Three major cross faults, the 14 East and 78 East vertical faults and the No. 1 Flat Fault, intersect and displace the main fault and the ore as shown in Figure 3.5. The No. 1 Flat Fault has a northwesterly strike and dips 45^{0} to the northeast. Apart from localized control of sheared ground in stoping areas, the main ground problems at Falconbridge were associated with this fault. With the development of number 9 shaft, mining activities were carried out below the fault causing the ore surrounding it to be left essentially as a horizontal pillar. The author worked at the mine as a student during the summer of 1971

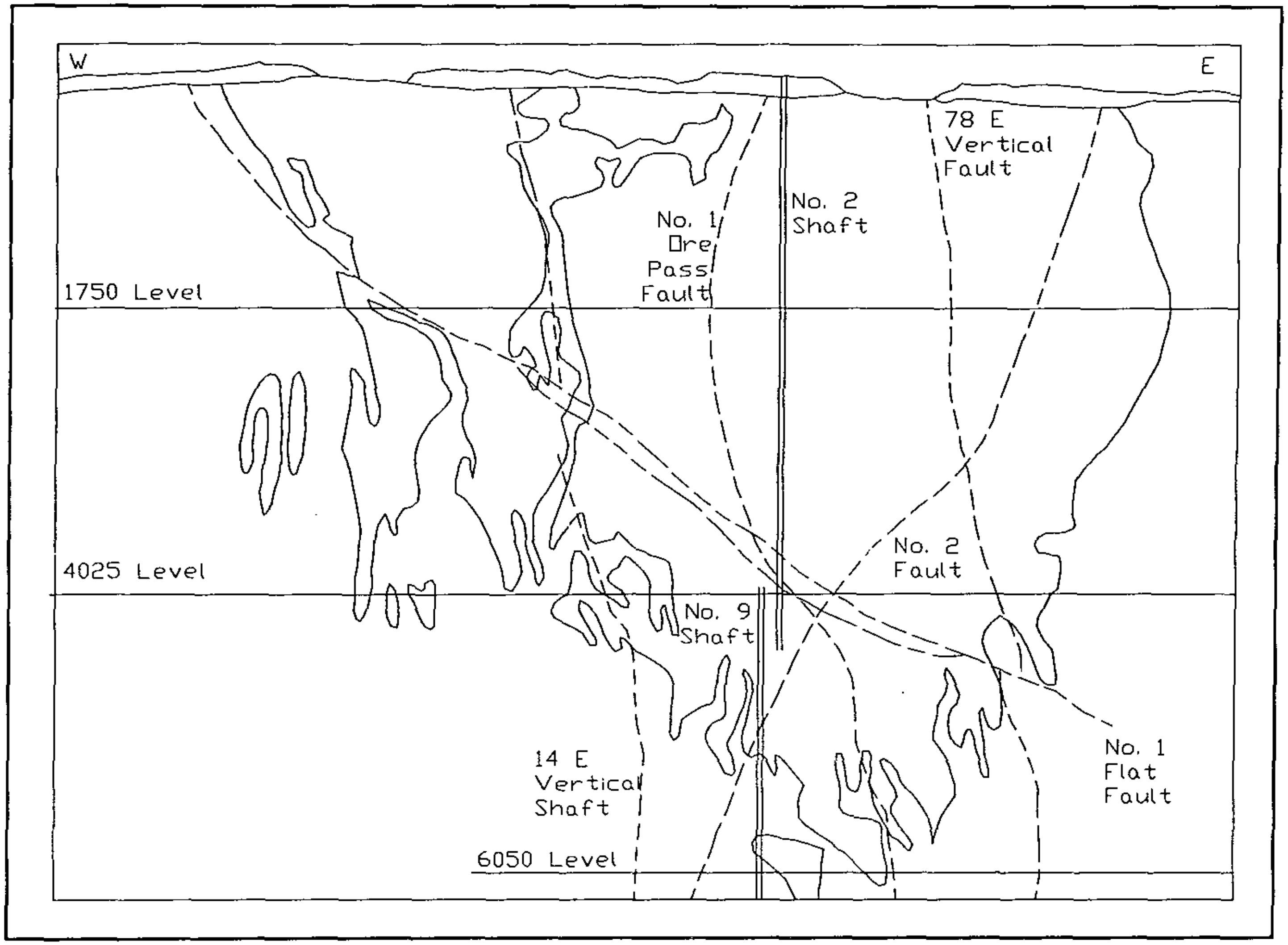

Figure 3.5 A Longitudinal Section Showing Faults at The Falconbridge Mine (9)

and recalls a longitudinal section in which the remaining ore reserves, which were coloured red, were shown dipping at approximately 45^0 along the fault.

Cut and fill mining accounted for 90% of production by the early 1970's. Sequencing involved the use of primary and secondary stopes of approximately equal longitudinal length, 60 to 80 metres. Undercut and fill, developed at the Falconbridge Mine to replace square set techniques in sill pillar recovery or secondary stopes in failed ground, was described by Tims (33). Figure 3.6 shows the interfacing of overhand and underhand cut and fill beneath the No. 1 Flat Fault. This mixing of methods of differing productivity demonstrates how geological structure can affect mine sequences. A similar situation at the Fraser Mine on the western edge of the basin was handled much more easily by bulk methods as described in Chapter 4.

Subsequent mining retreated towards the number 5 and number 9 shaft pillars where both the number 1 and number 2 ore pass faults cross the No. 1 Flat Fault. This is a classic example of an ever shrinking pillar in a large areal extent of longitudinal mining. A series of major rockbursts on June 20, 1984, associated with movement along these faults, caused the death of 4 miners and the eventual mine closure as described by Hedley, et al (9).

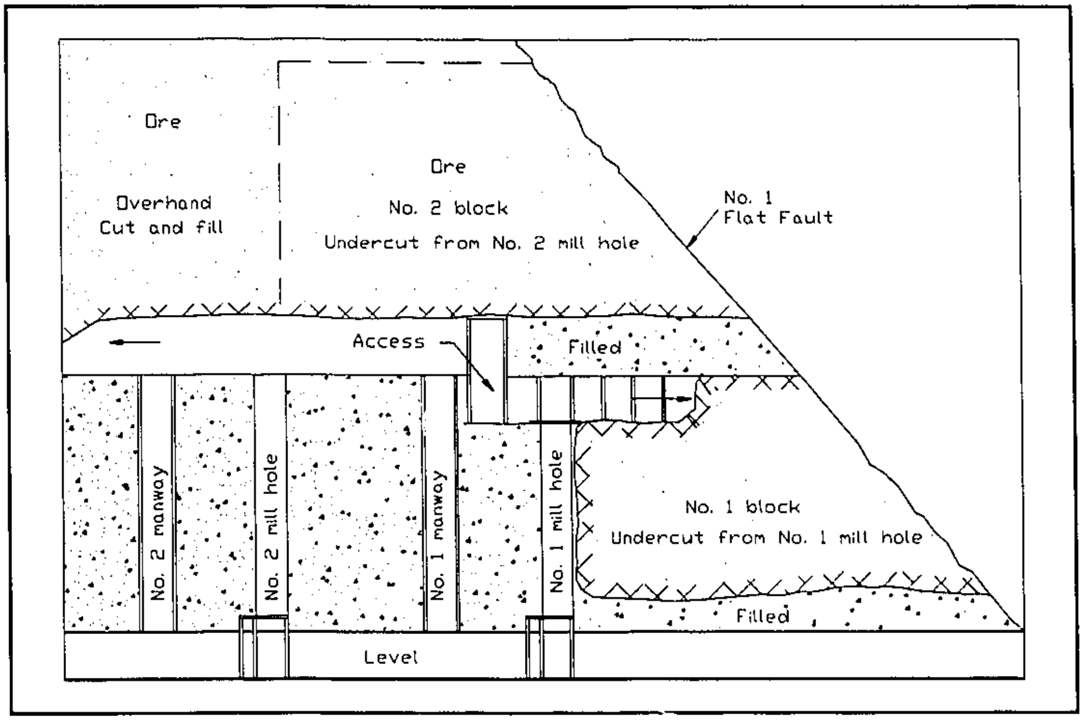

Figure 3.6 Underhand and Overhand Methods used below a Flat Fault (33)

3.1.3 The Campbell Red Lake Mine

Campbell Red Lake Mines Limited began production in 1949 at 350 tons per day with gradual expansion to reach 1170 tons per day in 1986. Mining was entirely by flatback boxhole shrinkage stoping until 1961, with stopes mined to sill pillars. With the advent of rockbursting on the Dickenson property, R.G.K. Morrison was consulted and his report (34) in August 1961 gave several recommendations including the introduction of cut and fill mining which accounted for 30% of production through the 1970's and increased to 90% by 1986. In the late 1980's, longhole methods were introduced, even in narrow vein mining areas.

As described in Chapter 2, several zones exist at the Campbell Mine but sequencing and rockbursting problems occur in two main zones, the A zone and the F zone. These two zones are thin tabular gold-quartz veins contained in andesite whereas other zones are contained in altered rocks.

3.1.3.1 The F Zone Mining Sequence

The initial flatback boxhole shrinkage mining provides an excellent example of a stiff mining sequence. In the F zone, shown in Figure 3.7, only 17% of the ore was left as pillars in the initial mining and no backfilling was carried out. This zone extends from surface to the 15th level, a depth of approximately 675 m and tapers from a strike length of about 450 m at the top to about 150 m at the bottom. The standard operating practice was to establish chutes on 8 m centres (35), in a sill of 4 to 6 m, thus creating the boxhole pillars. This high extraction rate had been practised down to the 12th level with no ground control problems.

When problems began, the sill elevation was increased to 8 to 9 m in stopes started in 1981 after a pillar burst on the 11th level. Such adjustments in mining layout should be part of overall sequence planning as mining depth increases, and should not be initiated only after problems have developed. In early 1982, there was a deterioration in pillars on the 14th level and attempts to destress them have been described by Neumann et. al. (36). It is interesting that high stresses developed in a pillar so near to the abutment of the mining area, although it was suggested that this may have been due to a fault acting as a stress concentrator.

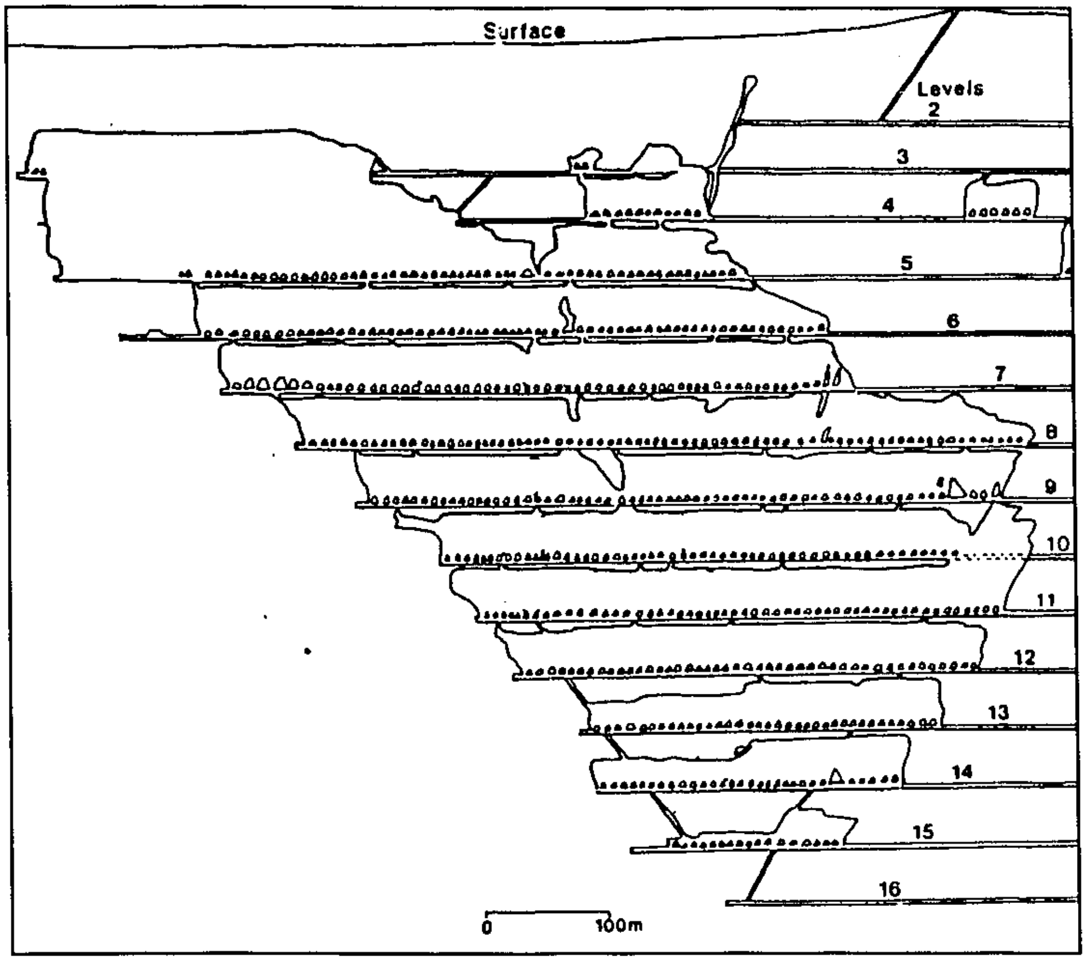

Figure 3.7 A Longitudinal Section of the F Zone at the Campbell Red Lake Mine

After a September 1982 destress blast, several boxhole pillars burst on the 9th, 10th and 11th levels and mining was stopped. Attempts to rehabilitate the 7th level were abandoned in December 1982 and all mining in the zone stopped for 10 months. In September and October 1983 attempts to re-enter the 7th and 12th levels were accompanied by additional bursting. A major sequencing of bursting which began on December 30th, 1983 and ended on January 10th, 1984 has been described by Hedley et al.(8). During this time, 1300 seismic events were recorded, 22 of them of sufficient magnitude to be recorded 180 km away in Pinawa, Manitoba. The largest had a magnitude of 3.3 M_L with 13 others over 2.0 M_L.

This provides an excellent example, over a zone of significant areal extent, of potential energy being stored in pillars containing less than 20% of the original longitudinal area. Also significant is that the remaining material is all contained in small pillars which in themselves must have blast damage surrounding the effective solid core. The strategy of increasing the standard pillar size as the depth increased began only after mining had reached the 12th level and trouble had already started. The potential energy was rapidly released as individual pillars shed their load to adjoining pillars which in turn failed. The process stopped only after sufficient load had been shed to the abutment to allow the higher level pillars to remain intact.

One alternative strategy would be to remove the pillars from the mined out stopes above in a timely fashion as suggested by Morrison (34). This would bring more load onto active mining areas and increase the size requirements of the pillars with depth more rapidly. Another alternative would be to leave behind pillars of adequate geometry to carry the regional load and which could then possibly be mined at a later date subsequent to being destressed.

3.1.3.2 The A Zone Mining Sequence

The A zone at Campbell is separated by a 12 metre thick boundary pillar from the adjoining Dickenson property, where, above the 300 metre level, mining has been carried out by shrinkage with no backfilling. The A zone varies from 20 cm to 1 metre in width, dips 80° to the south and has a strike length of 400 metres on Campbell. The first recorded rockburst occurred in 1965 with all bursting prior to 1970 confined to boxhole pillars or small sills.

Above the 10th level, mining was by shrinkage methods as in the F zone. The 4 metre sill pillars which were left were intended to be removed on retreat from the "party wall", or boundary pillar. In early 1989, while attempting to remove one of these pillars below the 11th level, a miner was fatally injured by a sudden and violent rockburst.

In 1962, the first cut and fill stope was started on the 11th level, again against the boundary. When that stope was undergoing the removal of the 6-9 m sill pillar, bursting occurred. Stopes above the 17th to 20th levels were established in an inclined "stair step" retreat sequence from the Dickenson

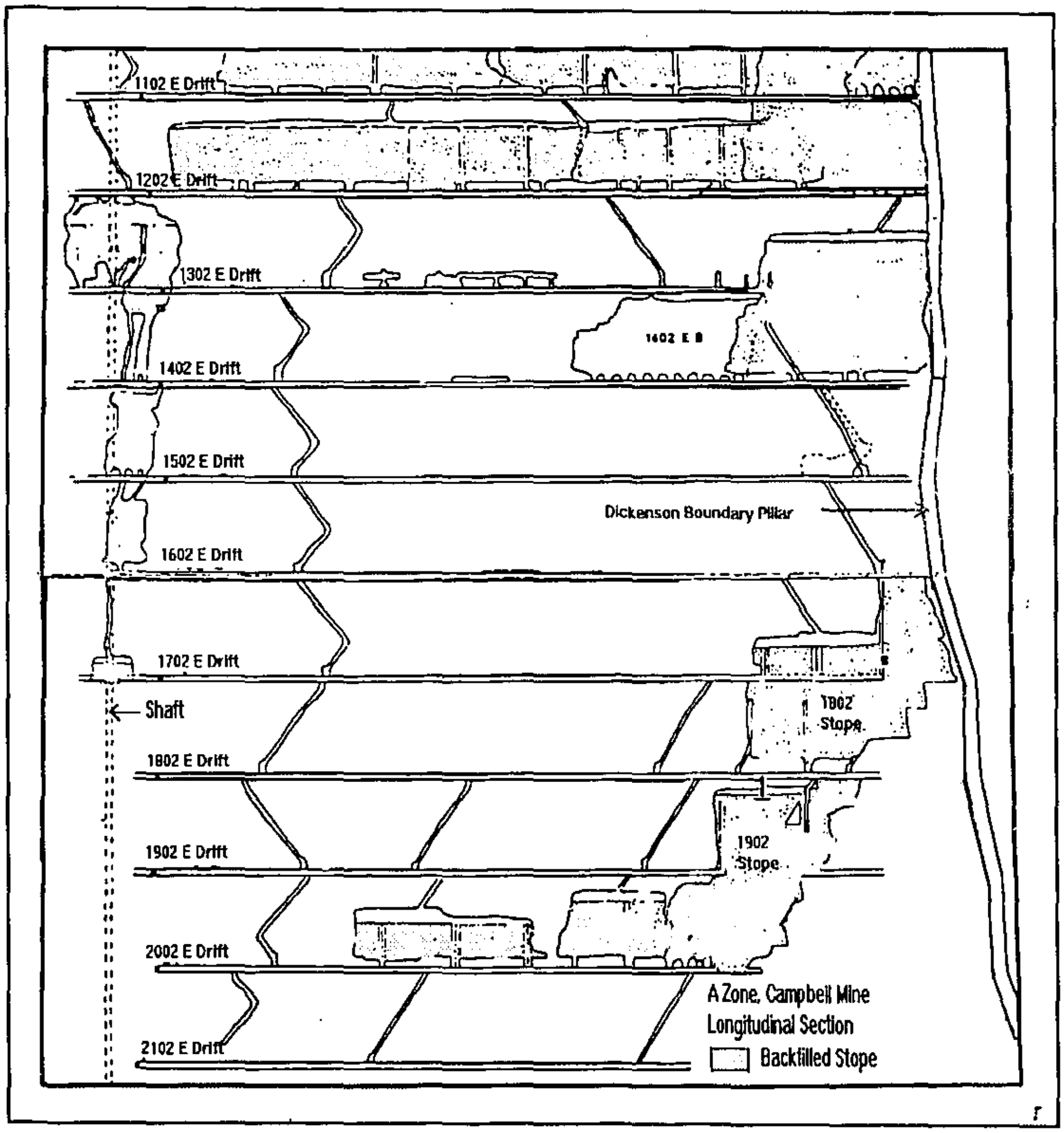

Figure 3.8 A Longitudinal Section of the A Zone at the Campbell Red Lake Mine (35)

boundary and were begun as 40 metre stopes to reduce the length of the sill pillar created. Both of these recommendations came from Morrison and are shown in Figure 3.8. As described by Neumann et. al. (36), these stopes were later extended to 80 metres because of the low productivity and very slow mining rate caused by their length. Neumann describes the destress program for the crown pillar of the 1902 stope and the sill pillar of the 1802 stope initiated after miners reported "working ground" and microseismic activity was reported. This allowed the crown pillar to be mined without incident.

The recent A zone cut and fill mining provided low productive capacity. Individual stopes generally had only single crews assigned, allowing 16 hours between shifts for the ground stress to re-adjust when the stope was empty. Initially, the adoption of longhole methods were resisted for two reasons: it was felt that dilution would be excessive in such narrow ore and that possible high grade stringers which occasionally occurred would not be recovered. O'Flaherty et. al. (37) describe the conversion of areas of the zone to longhole methods and measures taken to overcome initial dilution problems. An increase of 25% in productivity and a decrease of 25% in the mining costs are quoted, even after the development work required to make the conversion. The initial trials of the method were necessary both from a safety point of view and because the sequence demanded that the ore in the narrow ore zones be depleted at a rate equivalent to the other mine reserves.

3.1.3.3 Summary

Examination of mine sequences, methods and the ground control problems, as described in the mine report to the Stephenson Commission (35), confirms that:

1. That all bursts have been confined to the A, F and F2 zones which are narrow fracture-filled quartzite ore zones in the andesite.

2. That the bulk of the bursts have occurred in boxhole and sill pillars in unfilled shrinkage areas.

3. That the other bursts occurred in cut and fill sill pillars away from the Dickenson boundary pillar.

While the effects of introducing the longhole mining method have yet to be evaluated, the problem of rockbursting was initially controlled by the following methods:

1. Increased use of cut and fill mining.

2. Using timbered backs in shrinkage stopes to eliminate boxhole pillars.

3. Mining sill pillars immediately in cut and fill mining.

4. Extensive use of ground support systems.

5. Pillar destressing wherever possible.

6. Microseismic monitoring.

While many of these solutions are not directly related to mine sequencing, these examples from the Campbell Red Lake Mine contribute to our understanding of how to deal with brittle rocks while attempting to achieve a very high degree of extraction. It demonstrates a top-down sequence which achieved high extraction but which was limited by the increase of stress with depth. It also demonstrates how a move to introduce a more vertical aspect to the stope geometry was limited by productivity restrictions. Finally it provides an example of the successful use of destress blasting to eliminate areas of problem stress accumulation.

3.1.4 The Macassa Mine

Traditional Kirkland Lake area mining has been described in the discussion of the Lakeshore Mine. The Macassa Mine has operated at the western end of the seven mines along the "main break" for almost 60 years. The mine progressed to the west and down the orebody plunge via two shafts and two interior winzes to a depth approaching 2000 metres. While the mine contains brittle rocks and high stress levels, the extraction ratio on the various levels was reduced by a discontinuous ore zone. Arjang and Nemcsok (38) have estimated, however, that 400 bursts occurred during a 50 year period with only 10% considered heavy. Mining was by shrinkage stoping and cut and fill stoping using unconsolidated rock fill.

The high stress and brittleness of the rock in the area are demonstrated by the problems encountered in sinking the number 3 shaft to a depth of over 2200 metres (39). The shaft, a conventional four compartment shaft, has subsequently caused ground control problems. In addition, in order to minimize development costs at depth, the shaft was located so as to penetrate the orebody at the 1600 m elevation, necessitating a 60 by 180 m shaft pillar.

Important to the discussion of mine sequencing is the difficulty in removing sill pillars in cut and fill stopes and the introduction of new stope sequences to overcome this problem in the future.

The first method attempted to extract the sill pillars by removing vertical slices retreating from the end of the sill. This method, as outlined by Hanson et. al. (40) and shown in Figure 3.9, is similar to the approach described by Robson. The method necessitated the building of fill fences for every cut to contain the unconsolidated rockfill so was a very slow and costly approach.

The second method was to destress the sill as described by Hanson et. al. (40) and shown in Figure 3.10. The destress blast, which was detonated before a holiday weekend to allow rock reaction time, was heavily instrumented with extensometers and monitored by a portable microseismic system. The author inspected the blast during loading and the very high stress level was demonstrated by the pear shaped destress holes. After blasting, an exploration hole was drilled through the sill to determine the condition of the rock. The measured closure, the microseismic activity, and the observed rock condition

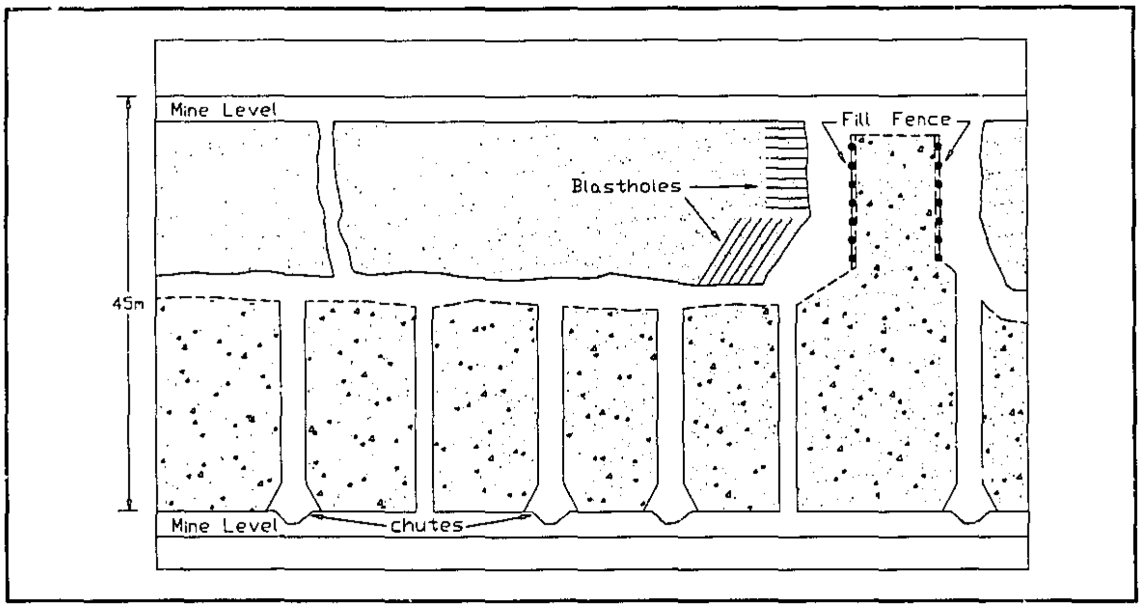

Figure 3.9 A Rill System of Sill Pillar Recovery at the Macassa Mine (40)

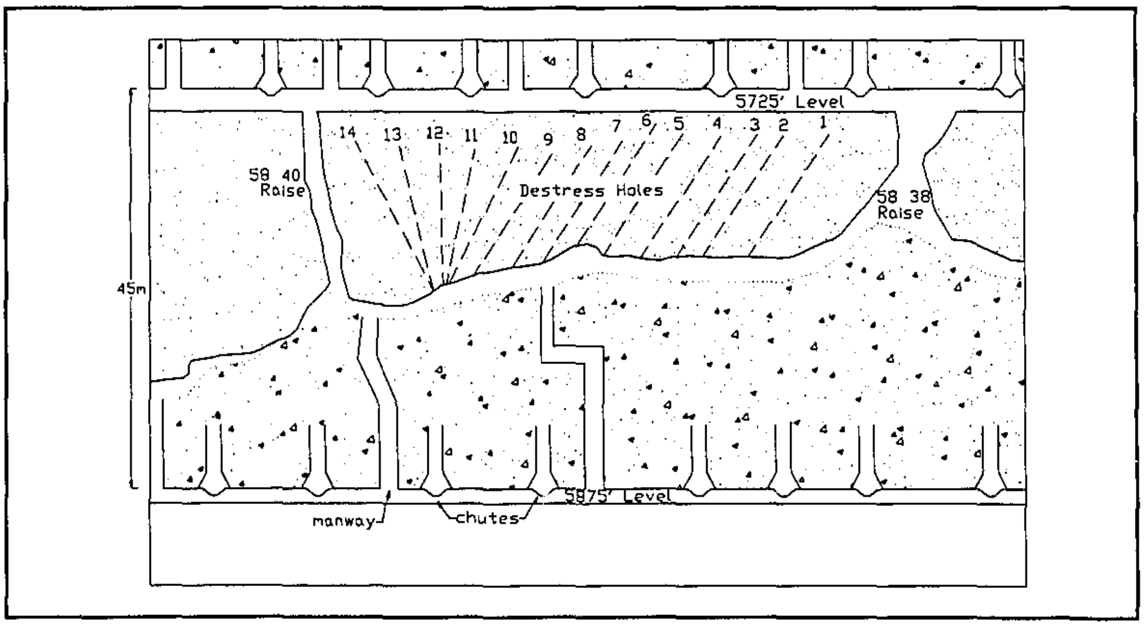

Figure 3.10 A Destress Blast in a Sill Pillar at the Macassa Mine (40)

suggested that the blast had not been successful, partially due to the difficulty in loading the holes. The sill pillar later burst, as did some 50 metres of the level beyond the stope which was completely filled with rubble (41).

The completion of number 3 shaft allowed the use of cement slurry (42). Numerical modelling suggested that the use of cemented rockfill would reduce closure, and showed that high quality fill was necessary to stabilize the shaft pillar. Mine personnel designed a new stoping sequence to eliminate sill pillars through the use of a rill mining system, again similar to the Lakeshore Mine, whereby consolidated rockfill would be introduced through a raise in the centre of the rill. The stope would then advance outwards in both directions, as described by Quesnel and de Ruiter (42), and illustrated in Figure 3.11.

When pillars burst, the damage above the sill destroyed the timber holding the unconsolidated fill in the stope above and made drift rehabilitation very difficult. To overcome this problem, a method under trial uses a 5 metre trench excavated above the sill and filled with consolidated rockfill to act as a buffer against this damage. The sill created above the conventional overhand cut and fill stoping will be removed by a rill with fill placed through a series of vertical culverts left in the already placed fill as shown in Figure 3.12. This fill will have to be placed by slushers with subsequent addition of the cementing agent meaning fill quality could be inconsistent and tight filling will be

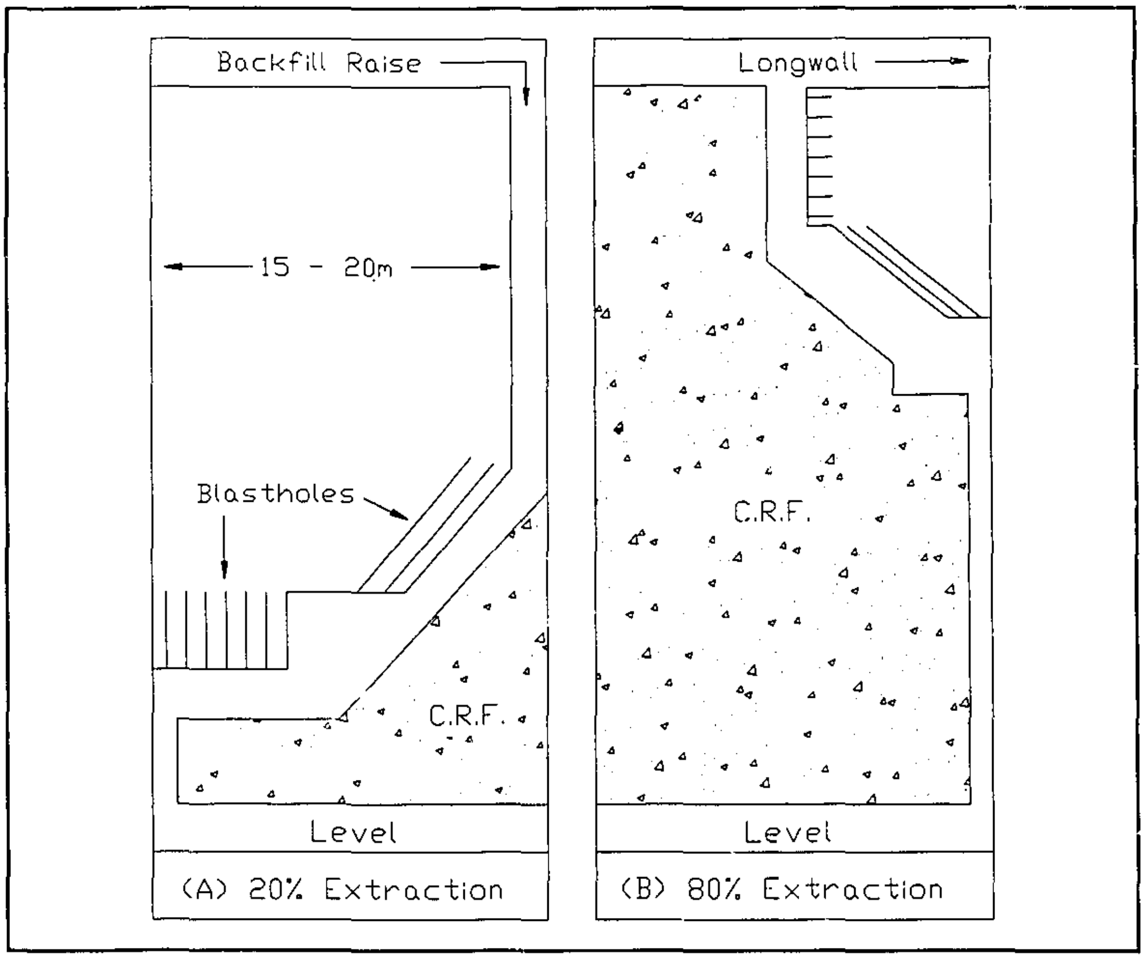

Figure 3.11 The Rill Mining System Developed for the Macassa Mine (42)

impossible for subsequent cuts. Also the miners will be required to work beneath this fill.

While the use of consolidated rockfill has reduced the incidence of rockbursting, mining problems persist in extracting such deep ore in the brittle and highly stressed ground.

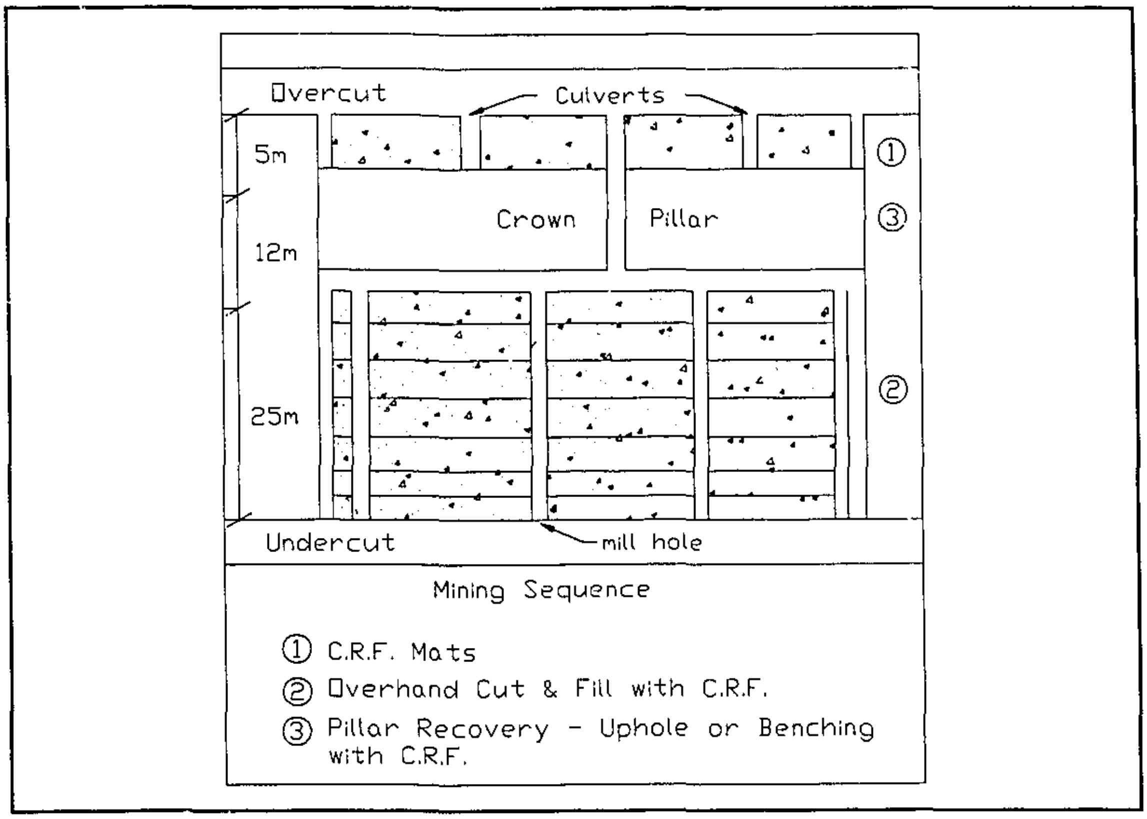

Figure 3.12 A Longitudinal Section of the Trial Mining Method at the Macassa Mine (42)

3.2 TRANSVERSE SEQUENCING PRACTICES

As outlined above, when the orebody becomes too wide to be self supporting in a single cut, the stopes are oriented perpendicular to the strike as transverse stopes. Within Ontario, this situation really only existed in the Sudbury area. In a 1975 paper Oliver (43) gives a summary of the evolution of the traditional sequencing strategies developed in the Sudbury area.

With the advent of transverse cut and fill stopes at the Frood Mine in the

1930's, stopes within a stoping block were 13.7 m wide and rib pillars 10.7 m wide. The primary extraction rate within these blocks was 64%. The overall sequence consisted of creating a series of these blocks with a strike length of 125-150 m. Barrier rib pillars, 20 m wide, separated these blocks and major sill pillars 15 m thick, were left every second level to control subsidence as shown in Figure 3.13. The blocks were sequenced from the ends towards the middle of the orebody.

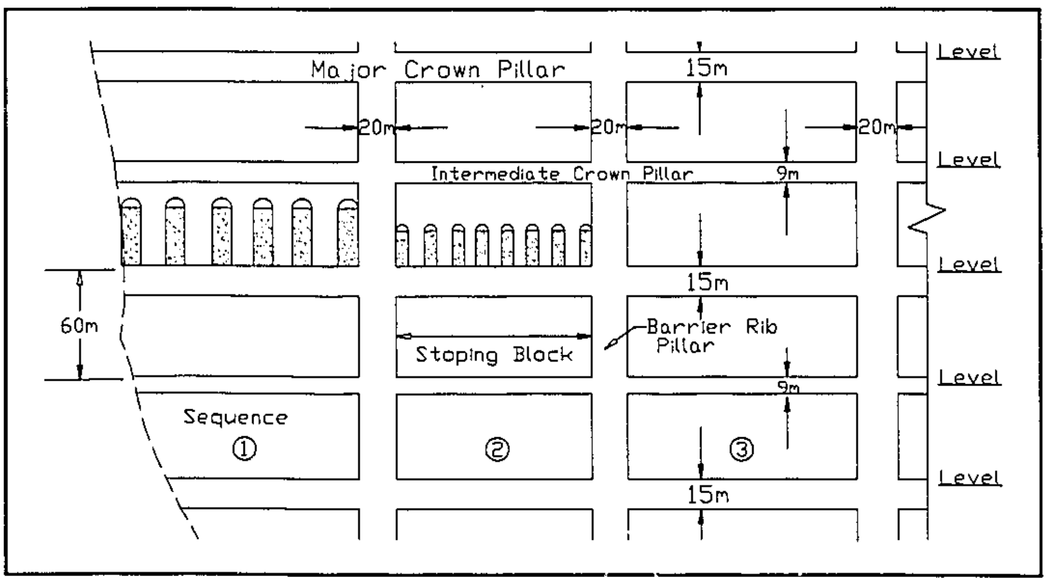

Figure 3.13 An Historical Longitudinal Layout from the Frood Mine (43)

With difficulty in controlling the back, cut and fill was converted to square-set mining. As block mining approached within 12.2 m of the level, ground control problems developed so the stopes were reduced to 9.3 m (5 sets + overbreak) and the pillars to 5.8 m (4 sets - overbreak). This relationship developed through trial and error, allowed mining to proceed without problem.

As mining progressed, the barrier, rib and sill pillars caused rockbursting problems. Based on the Morrison Report in 1942, the doming theory was adopted and the barrier pillars were eliminated, as shown in Figure 3.14. Below the 4000 level at Creighton Mine the geometry had to be changed to 5.9 m stopes (3 sets + overbreak) and pillars of 4.1 m (3 sets - overbreak). This reduced primary extraction from 62% to 59% but ensured a yielding pillar.

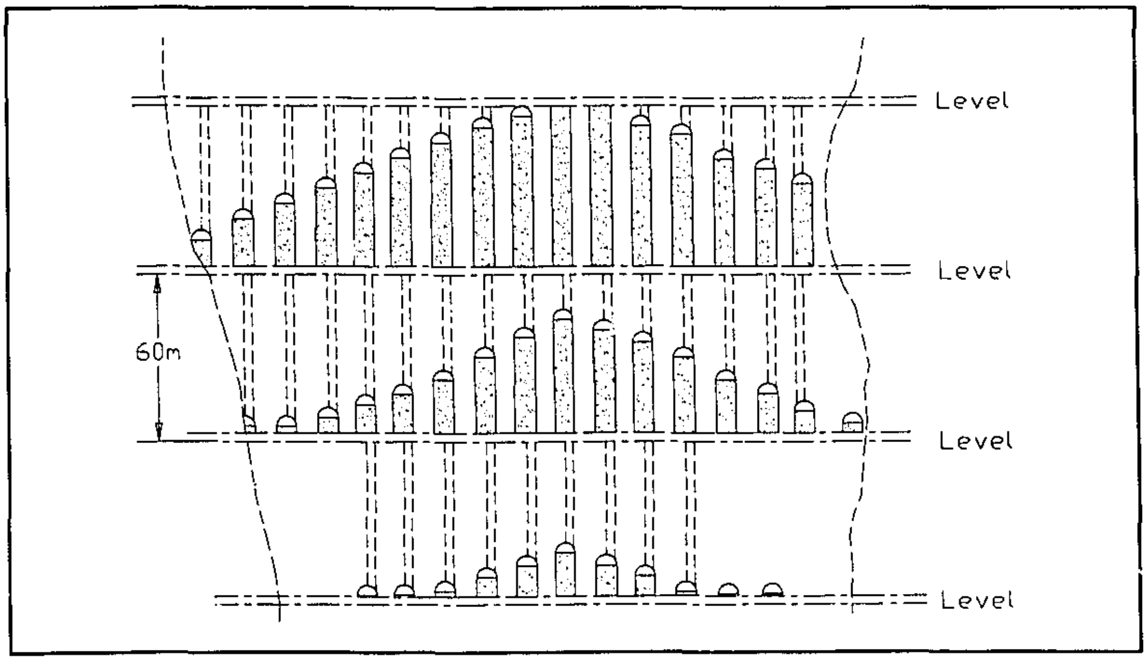

Figure 3.14 A Conventional Transverse Mining Layout Using a Domed Sequence (43)

With the advent of hydraulic fill in the early 1950's, of rockbolts in the late 1950's, and cemented hydraulic fill in the 1960's, the mining method reverted to cut and fill with undercut and fill for sill recovery. The mid-1960's saw the introduction of mechanized cut and fill and the sequences were changed (Figure 3.15) to allow a series of stopes to advance together in order to take advantage of the productivity possible with mobile equipment.

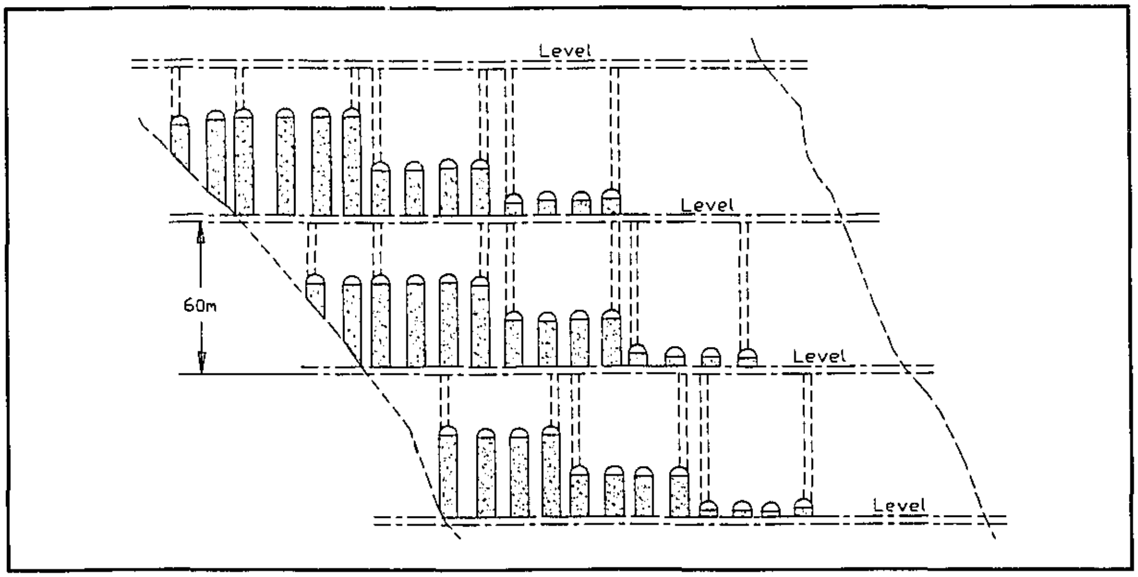

Figure 3.15 A Mechanized Transverse Mining Sequence (43)

The recovery of the crown pillars and the secondary stopes in these sequences was originally accomplished using square set mining but was converted to underhand cut and fill mining methods in the 1960's. The pillars designed at 5.8 metres to equal four sets minus overbreak were specifically designed for square set mining but were convertible to undercut and fill through double pass mining. It was to recover these pillars at Levack that Vertical Crater Retreat (VCR) mining methods were developed. By leaving a skin of ore along each wall, these pillars were successfully recovered from within unconsolidated hydraulic fill.

The philosophy in the Sudbury area was to eliminate stiff pillars whenever possible. The use of rib pillars which were narrow enough to yield in a non-violent manner and a sequence of mining which allowed this became the standard procedure. This subsequently gave rise to post-pillar mining, to the single pass mechanized cut and fill stopes using recoverable rib pillars such as used at Creighton Mine above the 7000 level, and to the one-pass system now designed for the Creighton deep area as discussed in Chapter 4.

To achieve this goal of eliminating stiff pillars, an agreement was reached between INCO and Falconbridge to mine the boundary pillars completely. At the Fecunis and Strathcona mines of Falconbridge which connected with the Inco Levack and Coleman mines respectively, whichever company first mines to the boundary supplies the other with complete plans of stoping in the area. One company can then mine to the fill of the other and any other facilities, for example, ventilation or drainage control, are created to allow this.

3.3 A DISCUSSION OF THE PROBLEMS

To better understand these conventional approaches to sequencing, it is necessary to examine what factors are common, where they differ, and what the constraints are in developing alternate approaches. The first point that should be noted is that transverse sequences using conventional stoping methods will not often be used in the future. Where ore thickness permits, bulk methods will be chosen unless some exceptional reason such as highly oxidizing ore precludes their use. The alternate sequencing strategies using bulk methods are discussed in Chapter 4.

In narrow vein mining, conventional mining methods such as shrinkage and cut and fill stoping will continue to be used:

1. Where the ore boundaries are so irregular that drilling longholes will cause either too much dilution or too great an ore loss.

2. Where the host material is so heavily sheared that the walls of the stope will not stand more than the one lift opening height created using breasting or the two lift opening created when using uppers.

3. Where the grade of the ore is so variable that some selectivity needs to be achieved by leaving material in place.

Using these methods guarantees that the slender flat-lying pillars will continue to be created. Because of the suggestion by Morrison that a more vertical aspect to the sequence be achieved, the cut and fill stopes on the 19 and 20 levels at the Campbell Mine were started 40 m in length but were subsequently converted to 80 m because of low productivity. The fill cycle and the advancing of raises and millholes represent such large time constraints in a cut and fill stope that any shortening of the stope increases the proportion of time that the stope is stopped to carry out these tasks.

This geometry gives rise to a crown pillar that often causes problems when the stope advances to within 10 metres of the level above. In stopes having 60 to 80 metres strike length, the ratio of longitudinal to dip extension of this pillar is 6 or 8 to one. If one considers that there is some blast damage along the top of the sill from the level or undercut of the stope above, this ratio is increased even further. The centre portion of this pillar is isolated from the longitudinal abutment with the areas above and below mined out. With the maximum principal stress usually perpendicular to the ore with some up-dip component, this pillar orientation is incorrect. A modelling study of these pillars would be interesting although it would demand three dimensional models in order to properly apply the stresses.

The rill system using unconsolidated fill used at Macassa to recover crown pillars was a move to introduce the vertical component to the sequence to reduce the rockburst potential of these flat sills. Using unconsolidated fill meant fill fences were required if the fill was to stand up above the angle of repose. The system using consolidated fill as shown in Figure 3.11 uses the consolidation of the fill to provide a stable slope to mine against. It is felt, however, that a great deal of potential exists for failure of this slope as blasted material moves down the fill face, causing a dilution problem.

The standard operating procedure has been to leave secondary stopes with a length equivalent to the primary stopes. These stopes in a "stiff" mine were extracted using a similar mining method although the increased ground pressures often necessitated tight fill and breasting rather than using uppers. In relaxed or failed secondaries, undercut and fill methods were used so again productivity would have required an equivalent length secondary stope.

Developing new methods to recover the secondaries would allow this geometry to be changed. In "stiff" mines, for example, larger pillars could be established to ensure regional stability. The design of the Stanleigh Mine in Elliot Lake offers an interesting example of this, if one can imagine the layout in longitudinal section rather than in plan in that flat-lying orebody.

The Stanleigh Mine is operated by Rio Algom Limited on the south limb of the Quirke Syncline at an approximate depth of 1500 m. Before reactivating the mine in 1984, the planning group looked at the best design, considering the increased depth of the operation, to ensure regional stability while still maximizing extraction.

In order to ensure stability, it was considered necessary to design rib pillars with a safety factor of 1.5. One alternative was to use rib pillars with a factor of safety of 1.2 in conjunction with regional load bearing pillars. This increased the overall extraction ratio and improved regional stability. From a practical point of view, stoping blocks of about 300 metres square were considered convenient and the system of mining stopes in groups of three was planned. This meant these larger pillars would be placed nine stopes apart. After detailed design by Golder and Associates (44), the final design comprises nine stopes of 20 m width with 3 m pillars between and 23 m barrier pillars, as shown in Figure 3.16.

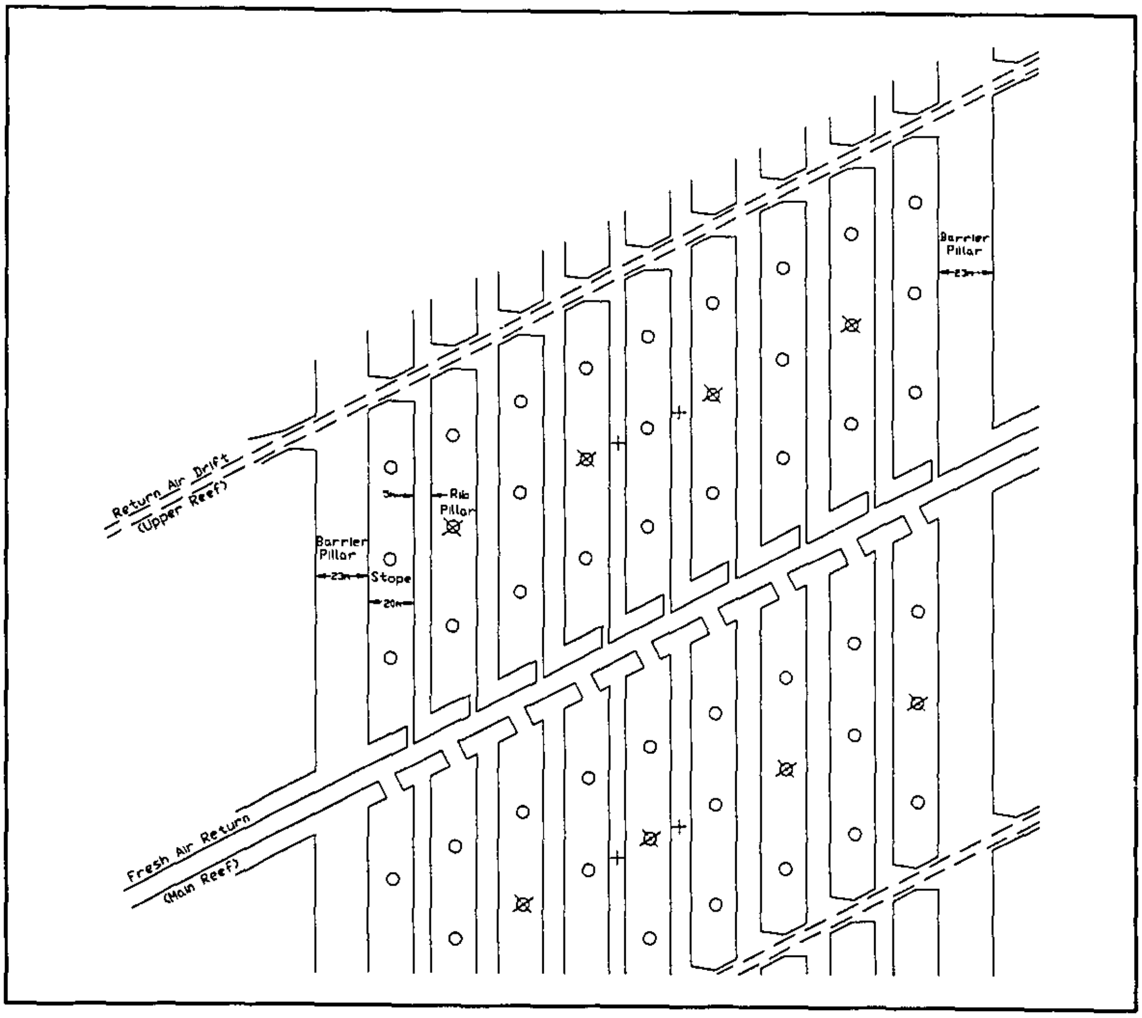

Figure 3.16 A Plan of the Stanleigh Mine Design, from Golder Associates (44)

Such pillars, interspersed within the shrinkage stopes as described at the Campbell Mine, might have provided the load bearing capacity to allow recovery of all the boxhole pillars and crown sills without major problems. If recovery of these pillars was then economically justified, methods could be developed to do so. They could be destressed in a concentrated manner from a remote drift so that rehabilitation of extensive lengths of on-ore drifts was not necessary. Even if longhole methods were used to recover them, the dilution or ore losses would only be incurred on this portion of the orebody.

In mines where a yielding sequence was desired, these secondary pillars could be designed to yield or caused to yield by destressing from the ends of the primary stopes as they advanced upwards. The pillars could then be recovered by longhole methods or even from a raise-bored hole using a drill stage as was successful at Namew Lake (45).

3.4 CONCLUSION

The traditional sequences of narrow vein shrinkage or cut and fill mining were carried out without much regard for optimization. Orebody definition was not carried out very far in advance of the development. The rate of advancing the development was slow but so was the rate of depleting a stope. With two to six cuts per year depending on the stope width or degree of mechanization, stopes could have a life of up to ten years. Many levels were placed in operation at the same time and secondary stopes were mined after completion of the primary stopes with the method dependent on the ground conditions.

In reviewing traditional mining sequences, two philosophies can be recognized depending on the brittleness of the rock mass. In mining areas with high modulus or brittle rocks three sequencing approaches were identified:

1. A stiff mining sequence whereby closure is prevented during the entire extraction process can be adopted. An example of this is provided by the boxhole shrinkage mining at the Campbell Red Lake Mine F zone. As long as adequate pillar size is maintained to support the regional stress mining can be carried out in a flexible and economical manner. If failures occur, the damage to access drifts can be difficult to repair. The situation could be improved by providing larger load bearing pillars as shown at the Stanleigh Mine. The recovery of these pillars can be delayed until late in the mining cycle. Any decision to recover them must be based on a separate economic study.

2. The stope sequencing strategy is based on destressing pillars as mining progresses. This approach has been demonstrated from the A zone at Campbell Red Lake Mine and from the Macassa Mine. The destress blasting can be slow and expensive and the transferred stress can cause damage elsewhere. Methods are being tried at Macassa to reduce rockbursting damage to access drifts.

3. A mine sequence can be adopted which eliminates pillars. This is the

approach taken at depth at the Lakeshore Mine. The use of this more vertical sequence was tried in the A zone at Campbell Red Lake and is reflected in the sill pillar removal methods tried at Macassa. Introduction of this sequence can restrict the availability of working areas.

In mines with lower modulus rocks, the creation of yielding pillars has been important to the overall mining sequence. The reduction in pillar size to ensure that no stress accumulates has been the practice in the Sudbury area. This latter practice is more in line with the sequences used in conjunction with bulk mining methods as described in the following chapter.

CHAPTER 4

CONTEMPORARY MINE SEQUENCES

The mine sequences discussed in this chapter were made possible by the introduction of new mining methods and equipment. Mechanized equipment allowed the development of new sequences in transverse mining stopes and eventually post-pillar cut and fill and ramp cut and fill mining. The advent of large diameter in-the-hole drills and precision machined drill steel allowed the increased drilling accuracy necessary for blasthole mining from widely soaced sublevels. Finally, the advent of remote controlled equipment allowed flexible drawpoint designs and removed the requirement to condition the walls or backs of large open stopes after blasting. Higher productivity achievable in individual stopes requires fewer working places for the same longitudinal mine area.

With wider orebodies, the aspect ratio of the pillars is more dependent on the thickness of the orebody. In wider ore, a mining sequence is less likely to be dependent on a "stiff" mine philosophy for two reasons:

1. The thicker pillars are less likely to be comprised of a homogeneous rock type based on the natural variability which occurs in a rock mass.

2. Pillar stability is more likely to be affected by structure; with a given joint frequency, more joints are going to be present and the likelihood of intersecting joint sets being present is greater.

This can be equated to the variability of the modulus of a rock mass estimated from a laboratory sample compared to its true in-situ modulus. The reduced modulus of this increased thickness will be important in managing stress during the ongoing extraction sequence.

As the ore becomes thicker, as discussed in Chapter 1, the details of the orebody outline become less critical as the boundary dilution or the ore loss incurred is proportionately less important. Some aspects of grade control will be sacrificed to a reduced mining cost.

Finally, considerations other than just rock mechanics give impetus to more rapid pillar extraction. Mechanized equipment demands larger openings which are more expensive to develop and maintain. The equipment also requires that more service facilities, garages for example, be available; this can lead to inefficiency if they are too widely separated. All levels must be ventilated if equipment is to move to widely dispersed mining areas, thus greatly increasing the overall air supply requirement.

In short, there is a concentration of mining activity which will likely see an earlier interaction between secondary stopes and the backfill of the primary stopes.

4.1 MECHANISED CUT AND FILL SEQUENCES

As described in Chapter 3, modified transverse mining sequences took advantage of the productivity available from mechanized equipment. There was little benefit in having expensive mucking equipment sitting idle in a stope during the drilling or filling cycles. As openings were created between the individual stopes, the remaining pillars were reduced in size. What evolved was a one pass mining sequence, the discussion of which is kept separate from the one pass bulk mining sequences described later in the chapter. The first of these sequences became known as post-pillar cut and fill mining.

4.1.1 Post-Pillar Cut and Fill Mining

The initial development of post-pillar cut and fill mining occurred at the Strathcona Mine of Falconbridge Ltd. The development of this mining method evolved from a novel approach to maximizing total ore recovery. The original planning for Strathcona was to mine transverse stopes 13.4 metres wide where the ore was in excess of 15.2 metres thick, separated by 5.5 metre pillars. This was a conventional Sudbury layout where the pillars were to be mined by undercut and fill methods.

The hangingwall was a gradational one which, through applying a cutoff grade to individual stopes, would have seen the lower cost primary stopes mined farther into the hangingwall than the higher cost secondary stopes. Because of production restrictions and because pillar recovery was going to be difficult in the relatively flat-lying ore, it was decided to implement the post-pillar mining method. The variation from tradition was that the ore pillars were, by design, left behind. The additional mining of the hangingwall of the secondary stopes, justified by using lower cost methods, compensated for the 15% of the ore tied up in pillars; in fact the total recovery of metal approached 91%. This method proved to be very successful as mechanized "triplets" of 15.2 metre wide stopes up to 122 metres long and joined through the rib pillar created openings not previously considered possible.

One of the major problems involved in sequencing post-pillar cut and fill stopes is the virtual impossibility of bringing one stope up beneath another. This could possibly be accomplished through the use of undercut and fill stoping methods or developing an in-fill drill access above the sill to remove it by longhole methods. Both of these solutions became impossible at Falconbridge mines subsequent to a corporate decision never to work beneath fill made following the quadruple fatality in an undercut and fill stope at the Falconbridge Mine.

To avoid this sill problem below the 4000 level at the Onaping Mine of Falconbridge Ltd., mine personnel used a vertical sill wall as shown in Figure

4.1. The limited extent of the orebody, bounded by a gradational hangingwall with a flat dip of approximately 40^0, required development on several levels to maintain production. The top stope had experienced ground control problems when approaching within about 12 metres of the mined out areas above. Sill recovery had been attempted by drilling uppers from the stope on a retreat basis, leaving a skin to support the fill, and remotely mucking the ore. Due to the shallow dip, however, only about 50% of the ore was recovered and

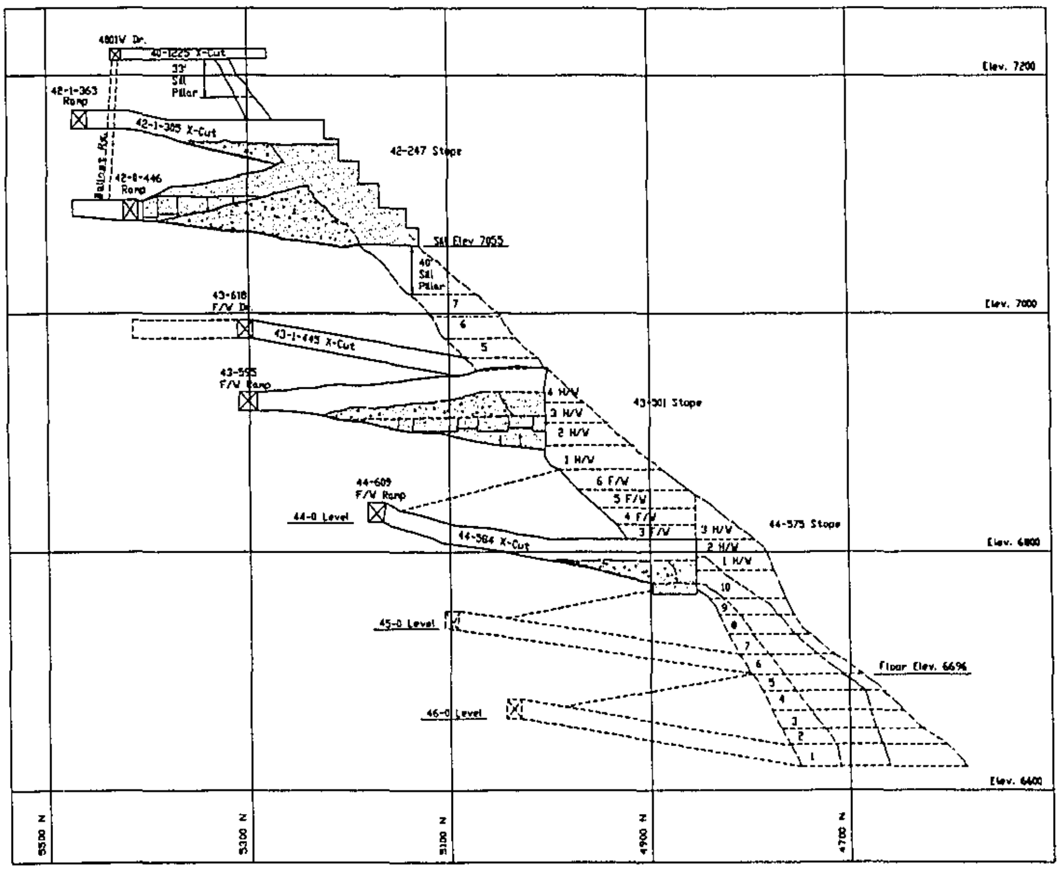

Figure 4.1 A Vertical Pillar at the Onaping Mine

significant rehabilitation of the openings was required between blasts. The possibility of using more expensive mining methods was precluded due to the low nickel prices at the time.

There are several problems associated with this use of vertical sills. Productivity is reduced in the small working area available in the initial and final phases of the stope. Stope access from the footwall becomes impossible when the stopes begin to overlap, thus necessitating development at the ends or in the hangingwall of the orebody. The ground control implications of the stress load being placed on the wedge shaped pillar just before the fill wall in the overlying stope is exposed have never been evaluated. There may also be stability problems for the fill wall when it is exposed but this will be dependent on what regional support is provided by the pillars which are imbedded in the fill. The fill placed along the exposed wall has to be entirely consolidated to at least a 20:1 ratio, or a thin skin of material left to prevent unravelling of the layered fill material.

The economic impact of this approach became obvious when the price of nickel suddenly jumped to record high prices in 1989. Not only was 15% of the ore still being lost in pillars, but it was impossible to rapidly extend the stopes into the now economic ore on the gradational hangingwall and still maintain the integrity of the vertical wall.

4.1.2 Ramp Cut and Fill Mining

Ramp cut and fill mining developed to exploit thick orebodies which had a somewhat more shallow dip. The method provides high productivity as equipment access is available at all times. The Trout Lake operation of Hudson's Bay Mining Ltd. claimed to be one of the most productive mining operations in the world with productivity of 4 tonnes per man-hour. Also, because access was provided by a spiral ramp to the 450 metre level, a stope could be started at any elevation. It is interesting to note, however, that the mine has switched to a bulk method for the deeper ore, partially because of a fatality caused by a fall of ground in a cut and fill stope in 1991.

As ramp cut and fill mining is development intensive, the orebodies mined by this method tend to be relatively thick and are generally mined in one pass. As in post-pillar cut and fill mining, sequencing of stopes is hampered by the difficulty of interfacing stopes along a common vertical wall and the virtual impossibility of bringing one stope up beneath another. Also, because of the large volumes of waste generated in the ramp and sublevel development, the backfill used tends to be unconsolidated rockfill.

The Brunswick Mining and Smelting operation at Bathurst, New Brunswick was the largest ramp cut and fill mining operation in Canada until its recent conversion to blasthole stoping. The Brunswick orebody was silled out on three main levels and was mined in one pass with waste pillars left for support. Unconsolidated rockfill was used exclusively with waste material from older open pits being used initially as fill material and a waste pit put into operation when this material was exhausted. In looking at the longitudinal section of the mine shown in Figure 4.2, it can be seen that little thought was put into removal of the sills created by choosing this mining sequence. There is presently about 18 million tonnes of ore tied up in three sill pillars within the last three cuts of the unconsolidated rockfill overlying the sills.

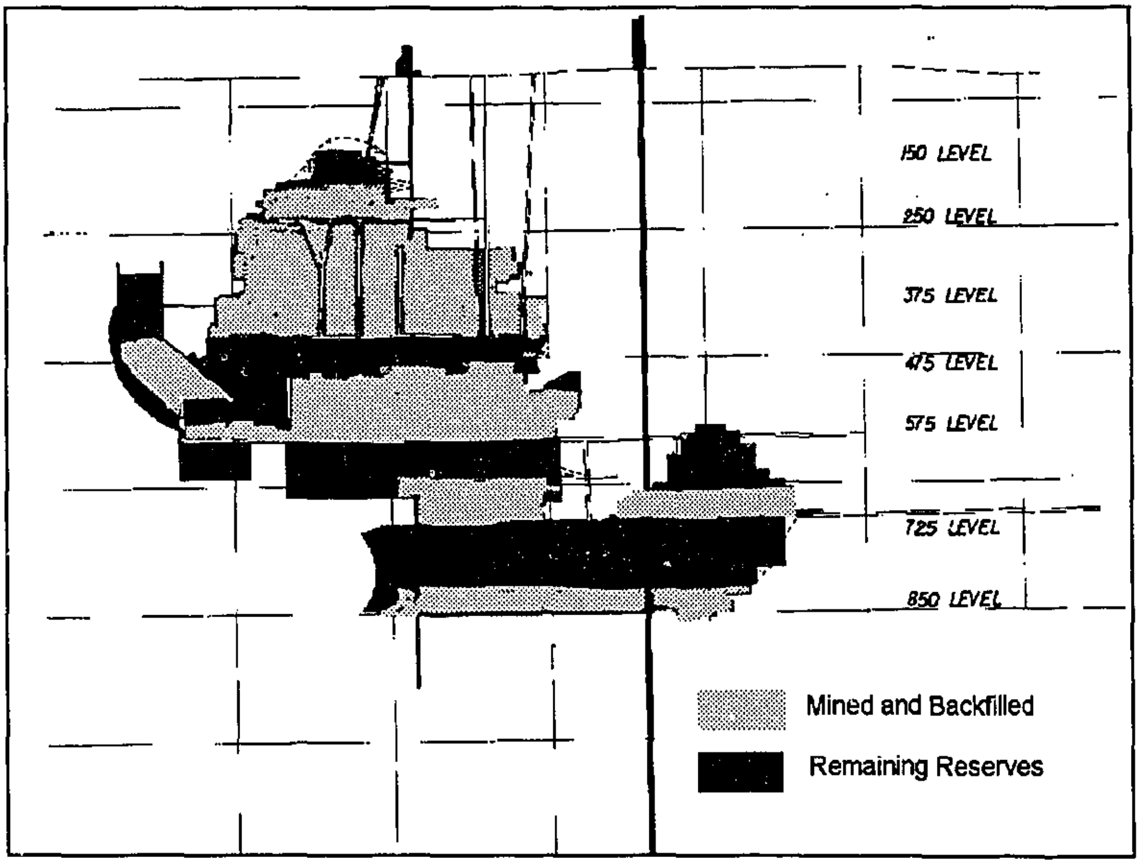

Figure 4.2 A Longitudinal Section of Brunswick Mine

This mine has now been converted to the blasthole stoping method below the 1000 level. The main reasons for this conversion, given that the initial financial studies predicted mining costs to be virtually identical for both methods (46), were quoted as follows:

1. The fear that, as the depth and overall extraction level of the mine increased, serious ground control problems would develop in the large flat sill pillars which would decrease the productivity and increase the costs of the existing method.

2. The fact that the waste pillars had been carried up the approximately 70^0 dip as opposed to being oriented vertically. This was accomplished by using specialized push plate dump trucks to ensure that the unconsolidated waste fill was packed tightly around the pillars and tightly to the back. It was also felt that this practice would not be possible at depth.

3. The fact that only one more major ore lift remained in the mine and it was judged possible to retain the productive rate of the mine for a longer period because of the higher productivity per longitudinal area available from a bulk mining method.

Brunswick Mining provides an excellent example of how mining methods and economics have affected the mining sequence. When the decision to sill out on three main mining elevations was made, the mine was experiencing economic problems, mainly as a result of poor metallurgical recoveries. The development of the presently widely used bulk open stope mining methods had not taken place. The use of hydraulic fill was impossible because of the high sulphur content of the ore and the very fine grind required for liberation.

Any decision to recover the 18 million tonne reserve, now tied up in sill pillars, must be based on an independent economic study. If one considers the present value of the increased cost of placing cemented rockfill and of disposing of the development waste in an alternate manner for the past 20 years, however, the decision taken at that time was likely the optimal economic one. It is not until the threat of additional ground control problems places the existing operating system in jeopardy that the increased cost of improving the fill placement methods can be justified.

4.2 THE TERTIARY MINING SEQUENCE (1-5-9)

The first of the bulk sequences to be discussed can be seen as an evolution of sequencing strategies at Falconbridge Ltd.'s mines, as described by MacMillan and Ferguson (30). They describe a sequence initially being considered for the Onaping deep ore. As shown in Figure 4.3, they describe the orebody as being 24.4 metres wide so that, to ensure adequate wall support, they require a pillar that is 9.1 metres wide. They show primary extraction of only 25% of the ore with another 25% on secondary extraction. This initial extraction was planned using overhand stoping methods with the 50% of the ore to be mined by tertiary extraction using underhand methods.

Support for the hangingwall comes from 75% of the ore during the

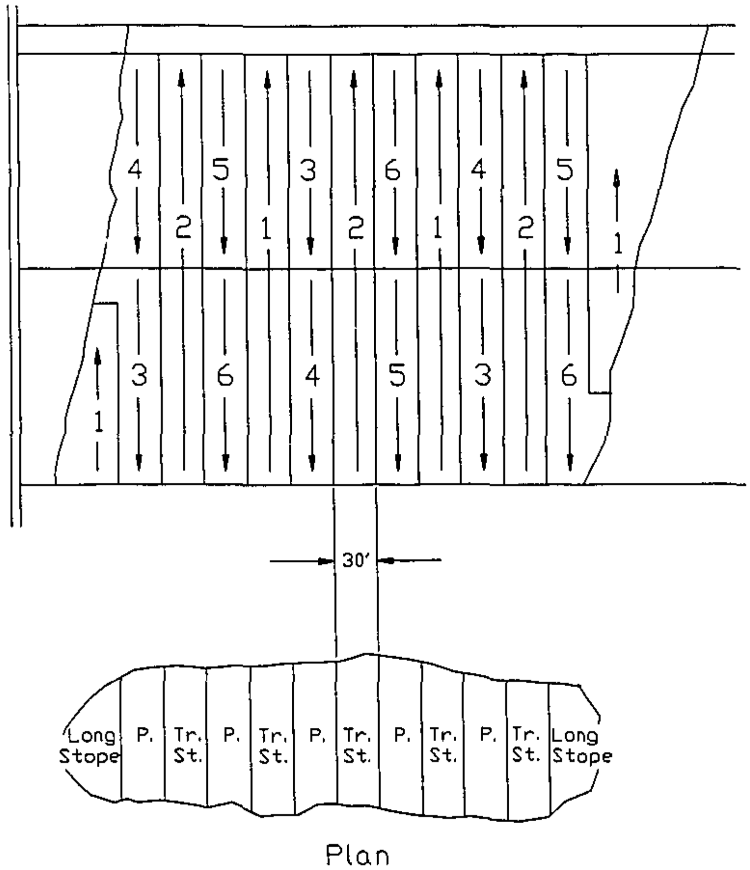

Figure 4.3 A Mining Sequence for Wide Ore (30)

primary extraction and from 50% of the ore plus consolidated fill during secondary extraction. It was thought that, during the remaining extraction, the consolidated fill would prevent pillar crushing beyond the point where underhand techniques could not be used.

They point out the importance of strictly following the sequence during the entire extraction process, thus limiting the number of operating stopes at any one time. They express concerns about the pillars bursting rather than failing gradually, the high cost of secondary extraction, and the increased costs of cement addition to the backfill of the primary stopes.

From these initial ideas the mine sequence used in the bulk mining stopes of the Fraser and Lockerby mining operations of Falconbridge Ltd. has evolved. This has been described by Potvin (16) as the 1-5-9 sequence. In effect every fourth stope is considered a primary and is extracted with 75% of the ore still intact. The third and seventh stopes are taken as secondary stopes in the sequence, still with no fill interface but with only 50% of the ore now intact. In a bulk mining situation, the restriction on production areas is offset by the higher productivity of the individual stopes and because tertiary production can begin before either the primary or secondary extraction has reached the top of the ore zone. While consolidated fill is still required in the two initial extraction phases, there is no requirement to consolidate the fill during the 50% tertiary extraction so the cost and productivity of this phase are greatly improved.

Potvin's diagrams as shown in Figure 4.4 are based on the Fraser Mine. The Fraser operation extracted several ore lenses which were not of major areal extent; this sequence was ideally suited to provide early production from each area as it was developed. Each stope was mined transversely across the lenses which could be in excess of 40 metres thick to an irregular gradational hangingwall.

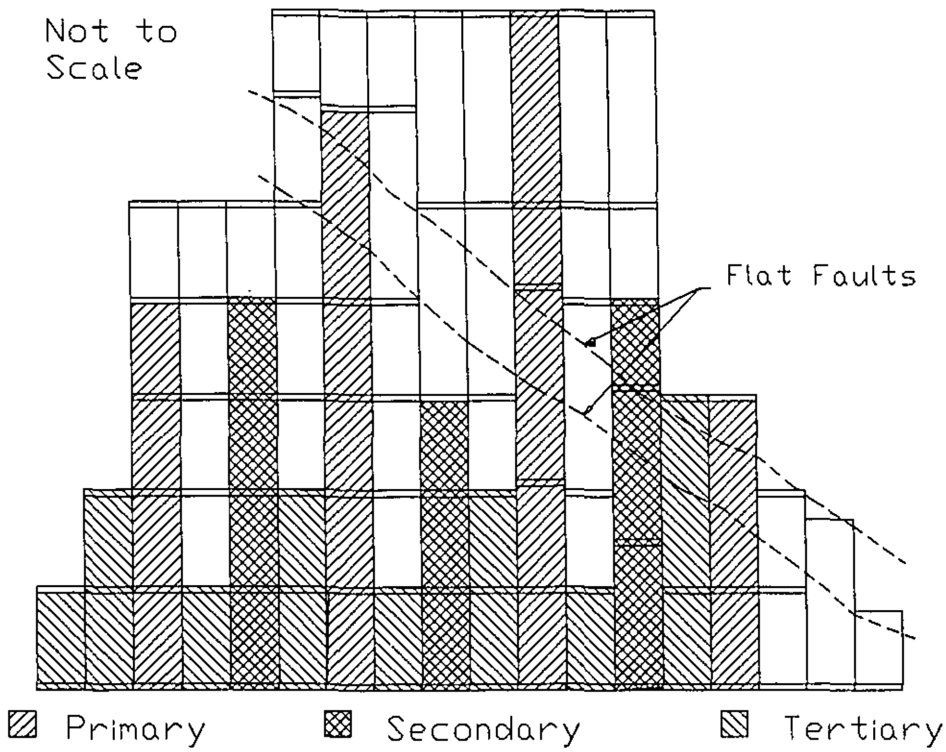

Figure 4.4 The 1-5-9 Sequence as Described by Potvin (16)

Of major interest in Potvin's drawing, although not discussed in his thesis, is the presence of the flat faults. A similar fault at the Falconbridge Mine was described in Chapter 3. When the fault was encountered at Fraser, the level development which would normally have occurred in the faulted area was eliminated and the stopes were double lifted through the difficult ground. This is a major increase in productivity and eliminates the additional support required for personnel to enter the area. The major penalty would be extra dilution from drill hole deviation or from the increased fill wall height. However, operating personnel reported no major problems in achieving this extraction.

This extraction sequence was also used at the Lockerby mine when all cut and fill mining was converted to bulk mining methods from the sill pillar left above the 3600 level cut and fill stopes up to a sill pillar left above the 2600 level. Another similar sequence was carried out above this sill pillar. The bulk stopes have now been successfully extracted and removal of the sill pillars is being attempted.

The importance of strictly following the extraction plan is demonstrated by the Lockerby sequence, as shown in Figure 4.5, at the stage when all primary and secondary stopes have been removed and only the tertiary stopes remain. It can be seen that one tertiary stope sequence, the 154 stopes, is beginning to lag behind the removal of the others. As this pillar of stopes immediately paralleled the main orepass in the footwall of the mine, stress transmitted through this tertiary stope caused failure of the orepass (47), necessitating the eventual establishment of a new orepass to bypass this level of the mine. This was considered unusual as these tertiary pillars would have been expected to fail and destress before their removal.

One would reasonably expect difficulty in developing the undercuts for these tertiary stope columns. In fact, with the use of extensive cablebolting, the ground above the undercuts was quite stable. Since these tertiary stopes are literally suspended in backfill, there was considerable pressure placed on the

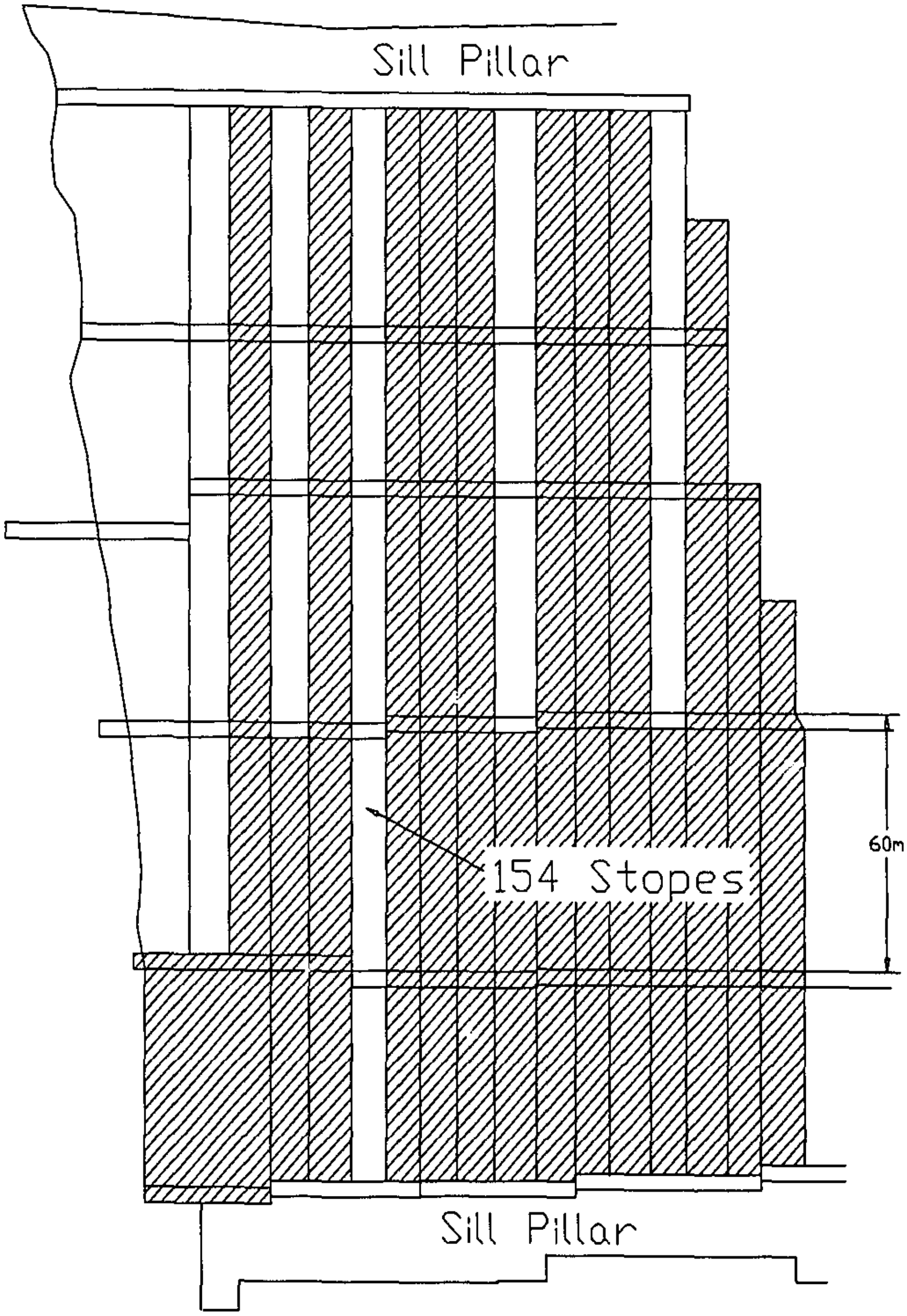

Figure 4.5 The Tertiary Mining Sequence from the Lockerby Mine

backfill itself. The backfill was kept from unravelling through the extensive use of splitsets in the backfill walls. These splitsets were installed in undersized holes drilled using auger steel into the consolidated 12:1 backfill. In observing these splitsets beneath the 154 stope just above the 36 sill, the load on them had developed to the point of the heads beginning to buckle. Movement in the fill mass had seemingly bent the ends of the splitsets to secure them and the squeezing of the fill mass had produced the load.

In evaluating this mining sequence, the above experiences indicate some advantages and dangers. The main advantage is that high productivity can be obtained early in the mining cycle. If primaries lead secondaries by two levels and secondaries lead tertiaries by two levels, the removal of all the primary stopes on four levels and the secondaries on two levels is possible before any fill faces have to be exposed. Stopes will then be available on five working levels to allow high productive capacity.

No ground control problems should develop until the secondary removal has advanced above a level, at which point the condition of the tertiaries will depend on the longitudinal extent of the ore zone, the depth of mining and the rock failure mechanism. If the retreat is to an overlying abutment or ore zone limit, no violent failures should develop if the tertiaries are not allowed to lag behind the secondaries.

The Fraser example also points out the possibility of avoiding problem ground near major structural features by eliminating the required entry of personnel. The height of a lift would be limited only by the stability of the high rock face, the quality of the backfill, ore irregularity, and drilling accuracy.

This sequence could be improved by adopting a "domed" removal sequence through starting primaries in the central part of the ore zone ahead of those toward the side ore limits. This might delay the development of productive capacity but in a wider ore zone this delay would be minimal.

The main problem lies in having to establish and develop the bottom of the ore zone to eliminate sill pillars, even though the bottom of the ore zone may not be yet defined. Other problems can develop, however, if the sequence is not closely followed. If the tertiaries are allowed to lag well behind the secondaries and failure is not progressive, violent failure becomes a possibility. If gradual failure takes place, ground support requirements in order to establish undercuts and overcuts could become considerable. Also, when the bulk of mine production comes from tertiary removal, backfill dilution may seriously threaten the profitability of the reserves unless the ore value is very high. The dilution quoted for Lockerby has ranged as high as 60%.

4.3 SINGLE PASS MINING SEQUENCES

The key consideration in classifying a bulk mining sequence as a single pass sequence is the virtual elimination of secondary pillars. Pillars may be used early in the sequence to establish productive capacity, but this is at a time when ground control problems would be expected to be minimal. After that, all stopes are mined in contact with the backfill of the previously mined stope. This lack of initial productive capacity and the requirement to place good quality consolidated fill in all stopes are characteristics of the sequencing strategy.

Two Ontario examples of this sequence will now be discussed. The "domed" sequence implemented at the Golden Giant Mine, which is also considered a one pass sequence, will be discussed in Chapter 5.

4.3.1 The Kidd Creek Sequence

As described by McKay and Duke (48), mining at Kidd Creek began as an open pit to the 220 metre level. Underground mining began in 1978 from the Number 1 Mine which was developed to the 790 metre level, and has continued to the Number 2 Mine which was developed to the 1400 metre horizon. Presently, ore below that horizon is mined and transported up to the existing services using Kiruna electric trucks, with development of the Number 3 Mine nearing completion.

In the Number 1 Mine, sublevel blasthole stoping was used with stopes 18 to 24 m wide, 30 m long, and 90 m high. Pillars between stopes were 24 m wide and sill pillars 30 m thick were left between mining horizons. The concept of total removal was practised; in fact, the first lift beneath the pit was removed without fill making the pit over 300 metres deep. The walls of the pit are slowly caving into this unfilled portion. The use of fill was only begun at the point where the eventual caving of the walls to the angle of repose would affect the stability of the headframes serving the underground operations. The fill used is cemented rockfill utilizing the waste rock from the pit.

As planning for the Number 2 Mine began, mine personnel felt that, because of the increased depth, cut and fill mining would be the only adequate method (48). Both the ramp cut and fill and Avoca mining methods were considered. One level was developed to allow ramp access to mechanized cut and fill stopes and an Avoca stope was subsequently tested on the 2800 level. After observing the success of blasthole mining as used at the Lockerby mine, a trial stope was tried at Kidd with such success that the method was adopted.

It was decided to make the individual stopes 15 m wide by 30 m long by 60 m high. The one pass system as practised at Kidd Creek uses a triple lift of these 60 metre stopes with the ore silled out every 180 vertical metres to increase working levels, as shown in Figure 4.6. The first stope on a level is mined adjacent to the hangingwall and after it is filled and the fill allowed to cure, the next stope in the same panel is mined to the footwall. Adjacent stopes are then mined along the strike of the orebody.

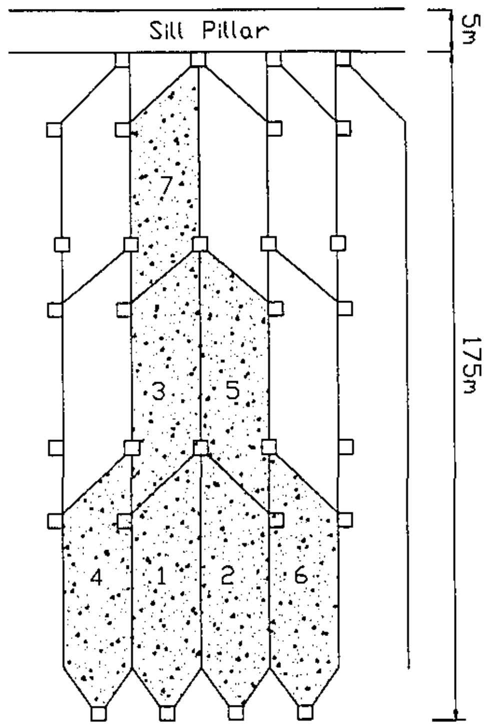

Figure 4.6 A Longitudinal Section of the Extraction Sequence at Kidd Creek Mine (48)

The delay required for the fill to cure is overcome, as shown in Figure 4.7, by taking a second stope before completing the slot. The sequence is established by stope 3. If additional productive capacity is required, a two core extraction can be established as shown in Figure 4.8. The extraction sequence of the three le.els follows a domed sequence with the sill pillar eventually removed by drilling upholes in a radial pattern from the overcut and dropping the material through an adjacent unfilled stope as shown in Figure 4.9.

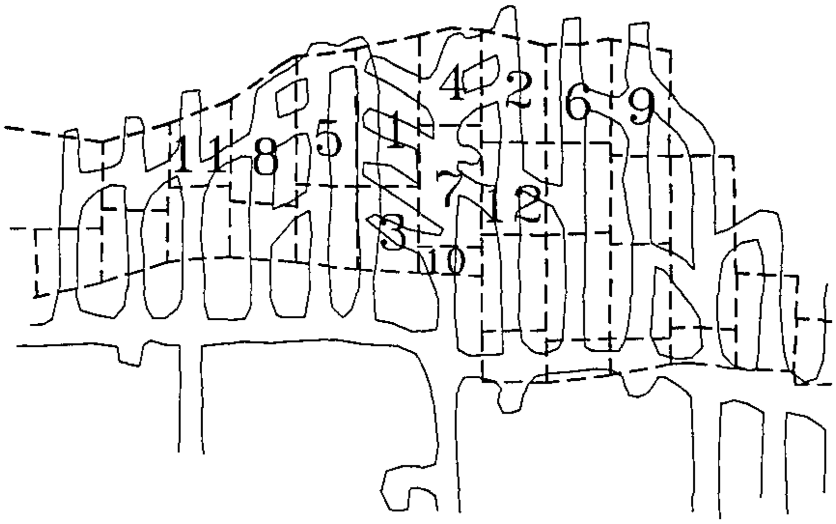

Figure 4.7 A Plan View of the Single Pass Mining Sequence at Kidd Creek Mine (48)

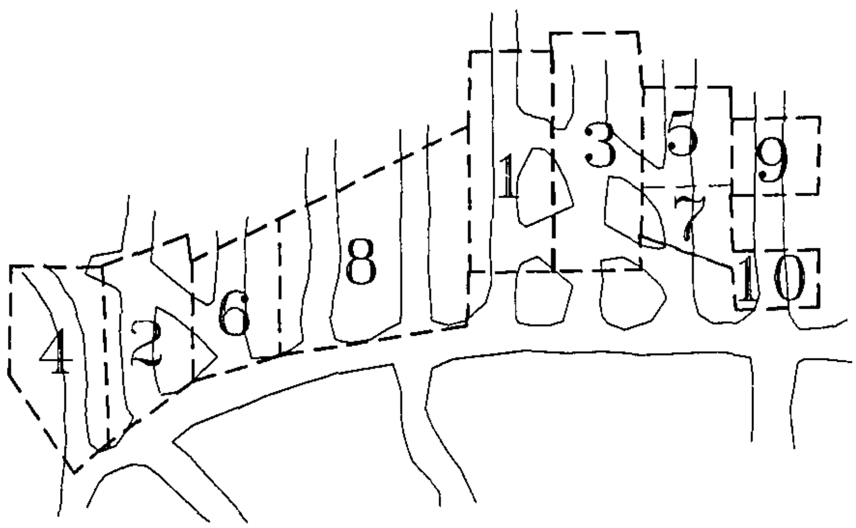

Figure 4.8 A Two Tier Extraction Sequence at Kidd Creek Mine (48)

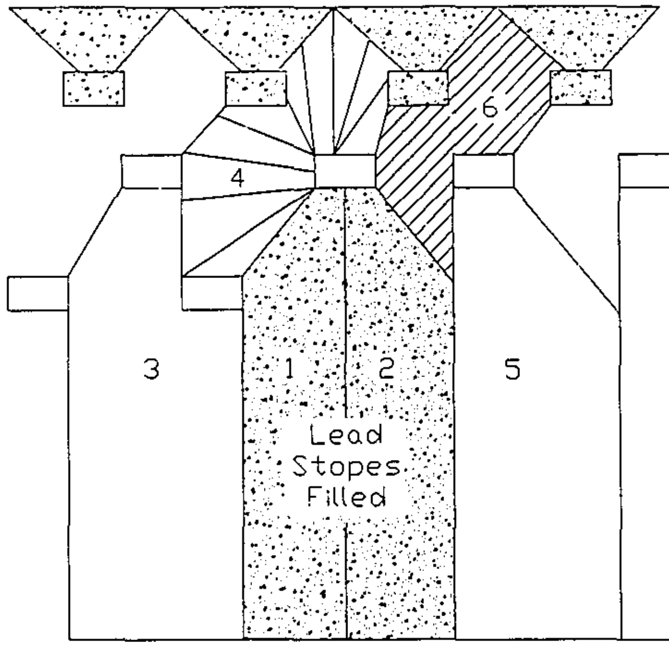

Figure 4.9 Sill Pillar Removal at Kidd Creek Mine (48)

This one pass system allowing 100% extraction has been practised at Kidd Creek with considerable success. Mining has now gone below the 1400 metre level and a minimum of ground control difficulty has been experienced. It should also be pointed out that sequencing has been diligently followed at Kidd Creek and backfill quality control has been a model for other operations.

4.3.2 The Creighton Mine Sequence

The Creighton Mine began as an open pit operation in 1900 and moved underground in 1907. In 1969, the number 9 shaft serving the present operation was sunk from surface to the 7137 foot level to become, at that time, the deepest continuous mine shaft in the western hemisphere. Ore mineralization is known to extend to the 10,000 foot level.

During its 85 years of uninterrupted operation, the Creighton Mine has seen more development and adaptation of mining methods than any Canadian operation. With the development of large diameter underground drills in the mid-1970's, an almost complete conversion to vertical retreat mining has occurred. Important to this discussion is the single pass mining sequence developed for mining from the existing 7000 level to the 10,000 foot elevation, which began with experimentation on the 6800 level.

The first experiment on the 6800 level involved the elimination of the upper sill drift normally developed in ore and its replacement by a drill drift excavated in the backfill of the level above. This drift was excavated in backfill by an Alpine continuous miner, as shown in Figure 4.10, to avoid the problems of drifting in highly stressed ground. This concept has subsequently been abandoned for the approach of excavating the drill horizon immediately below the backfill in the destressed rock at the edge of the underlying abutment.

Two main reasons have been given for abandoning the approach of excavating in backfill. The first was the fear of having personnel work in a backfill drift without complete surrounding support. While Bernold screen and

Armco arches were tried for support, the final decision was to utilize timber sets. The installation of these sets delayed the excavation considerably.

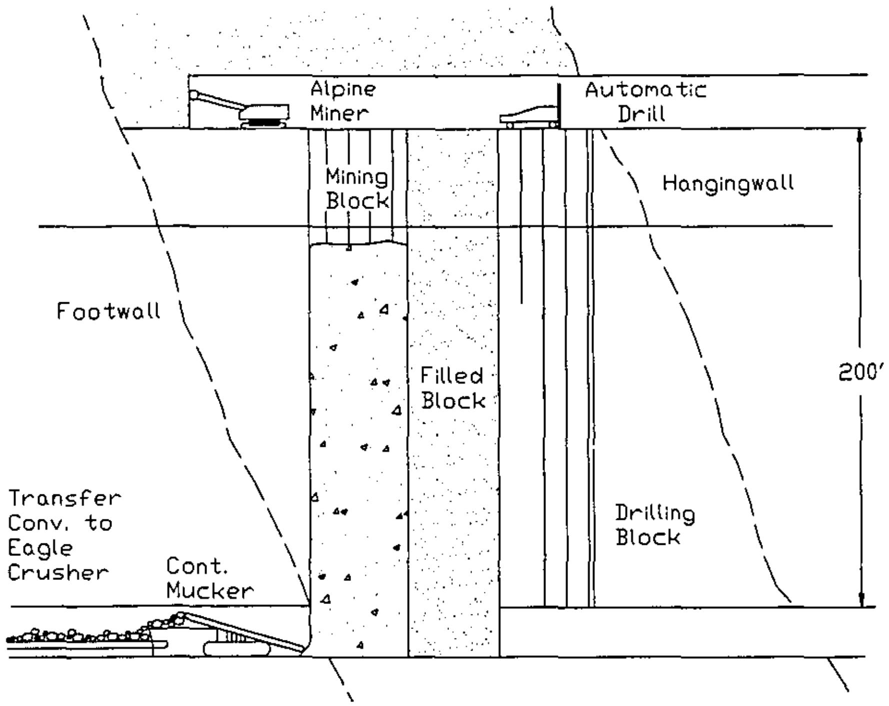

Figure 4.10 Excavating the Drili Access Using an Alpine Miner

Secondly, all material below the 6600 level has to be elevated by conveyors before hoisting so it was difficult to dispose of the excavated tailings.

A second experiment saw the creation of a destress slot across the orebody above the 6800 level by mining an initial line of stopes as shown in Figure 4.11. This was done to divert the maximum principal stress, which at Creighton is parallel to the orebody.

The success of these experiments gave rise to the method presently used between the 7000 and 7200 levels and planned for successive levels. A 12.2

m square panel of ore is first mined in the centre of the orebody. After filling, mining radiates outwards from this slot, panel by panel, as seen in Figure 4.12. There are difficult scheduling and layout problems inherent in this system, including the requirement of both hangingwall and footwall access and difficulty excavating beneath the ore wedge left at the bottom of each stoping block.

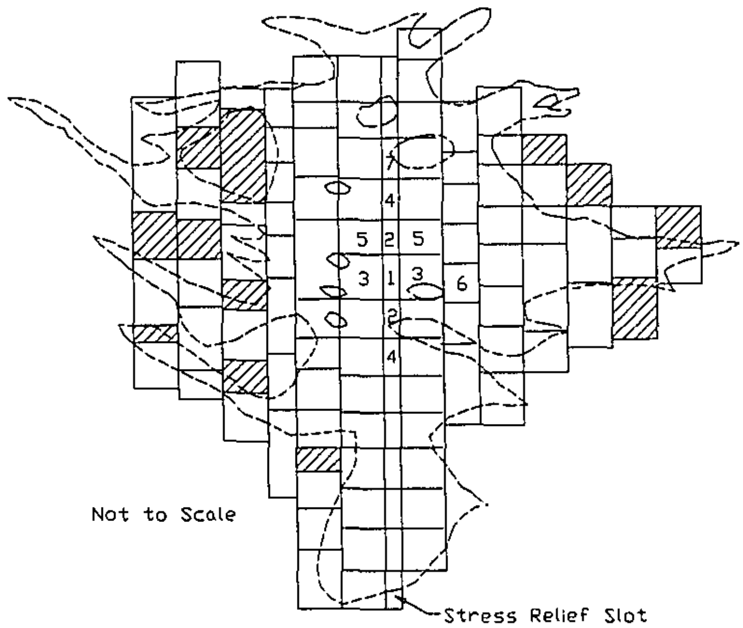

Figure 4.11 Creating a Destress Slot at the Creighton Mine

These problems may reduce the production level of the mine as the pillars remaining above the 7000 level, which now augment the production, are mined out. Succeeding levels can only begin production after the diamond shaped slot of the level above has been established entirely across the orebody.

The approach, however, is seen as the only alternative to controlling the

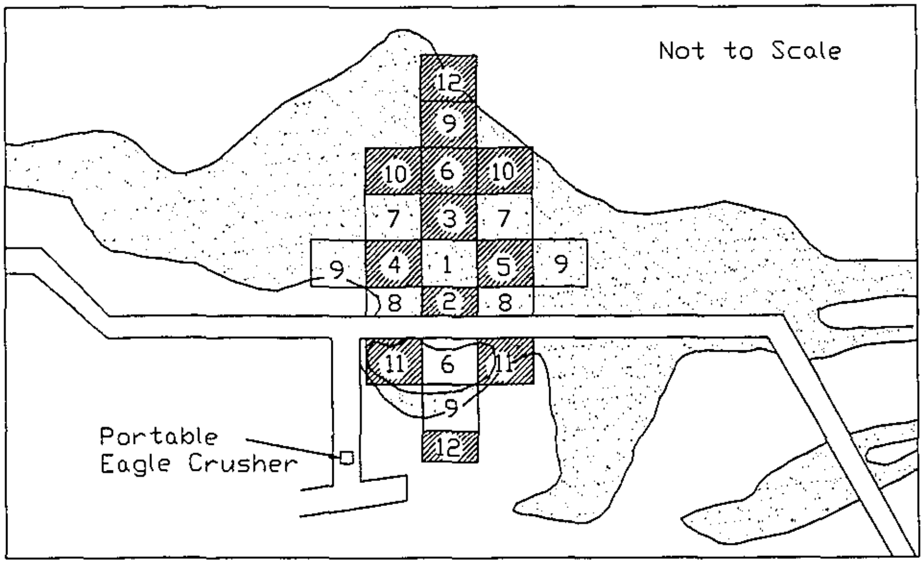

Figure 4.12 The Mining Sequence for Deep Ore at Creighton Mine

immense stress at such depth and demonstrates an impressive adaptation of mining methods and technology to overcome stress related problems.

4.4 ALTERNATE MINING SEQUENCES

To this point, a tertiary mining sequence applied in longitudinal section and single pass sequences applied in plan have been discussed. Several mines use a primary-secondary sequence in longitudinal section and this, as used at two Hemlo area mines, will be discussed in Chapter 5. This section describes a possible tertiary sequence which could be applied in plan and discusses the possible adaptation of other sequences to a top down approach.

4.4.1 The Mount Isa Sequence

The Mount Isa Mine in Australia is a high tonnage base metal operation producing from a thick, flat-lying tabular deposit. The "eggcrate" sequence developed at that operation (16) is shown in Figure 4.13.

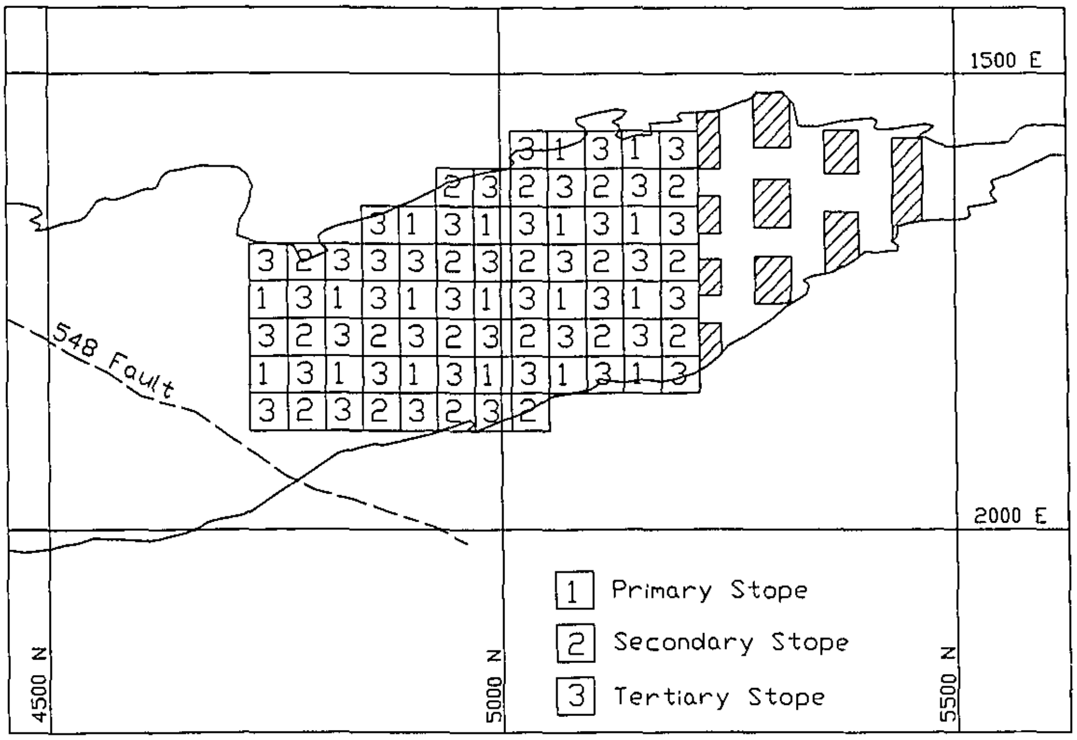

Figure 4.13 The Eggcrate Sequence at Mount Isa Mines (After Alexander and Fabjanczyk 1981)

It is a tertiary sequence using square stope outlines similar to the sequence developed for the deep ore at Creighton. Mount Isa's sequence is different in that larger productive capacity is possible because the availability of working places is not limited by the curing time of the fill. This is more critical in a flat-lying orebody where different working levels cannot be developed. To start mining at various points along the orebody would create the ground control problems of the ever shrinking pillar between working areas.

The sequence at Mount Isa is implemented using access for drilling and filling operations in the hangingwall, and developing drawpoints in the footwall. The system could be adapted to a thick, steeply-dipping deposit if drilling and filling access were developed in the backfill of the level above and drawpoints were created in the abutment of the ore beneath. Production from the level beneath could then begin when the upper level had reached the tertiary extraction stage at one point across the orebody section.

4.4.2 Top Down Sequences

From a development standpoint, a sequence beginning at the top of the orebody and proceeding downwards provides the earliest access to ore. However, the majority of the bulk longitudinal sequences used today progress upwards from some main access level. The progression may be to the top of the orebody or to some previously created mining sequence. In this latter case, a major sill pillar is created which requires special methods or a higher level of ground support to complete its extraction. The situation often develops when the mine's ultimate depth is not determined and additional ore is discovered at depth or early production is developed at the upper levels of the mine.

The single pass sequences previously discussed in this chapter are in reality "top down" sequences, the Creighton Mine sequence progressing by a single lift and the Kidd Creek sequence advancing by three lifts and a removable sill pillar. All of the bulk mining sequences discussed, the 1-5-9 sequence for example, could progress from the top down if access for the drilling and filling operations were created in the backfill of the overlying stopes. Such sequences would be somewhat less productive than the standard practice as the cure time of the fill in the previous lift becomes a factor and the access drifts could not be established in advance. These sequences should be considered for second stage extraction when mining additional ore below a previously established upwards advancing sequence.

Excavations in previously poured hydraulic fill were tried at Creighton Mine as described above. A drift being developed in hydraulic fill at the Denison Mine led to a fatality (49), when a skim of highly consolidated fill left between the opening and the original back subsequently failed onto the scooptram operator excavating the material. Openings have been successfully excavated in cemented rockfill at the Kidd Creek Mine using a Dosco roadheader as described by Wittchen et. al. (50). An extensive research program was carried out by the author (51) to assess the stability of openings in hydraulic fill and to evaluate the potential use of such openings in mine sequencing.

This research program involved both physical and numerical modelling of the stability of openings in hydraulic fill as described by Pelley and Mitchell (52). Physical modelling was carried out on cemented tailings material from Falconbridge's Strathcona Mine using the centrifuge installed in the Department of Civil Engineering at Queen's University. These tests showed the openings to be inherently stable; even when the fill was undermined the backfill failed to a stable arch as shown in Figure 4.14. Numerical modelling was carried out using the FLAC finite difference code from Itasca (12). The results also suggested such openings to be stable and, when undermined, the fill also failed to a stable arch as shown in Figure 4.15.

The presence of large pieces which fail into the open stopes before backfilling and disposal of the excavated material were the main operational problems to be expected when creating openings in backfill. Excavation systems could be developed to create such openings even in unconsolidated backfill or graded unconsolidated rockfill (51). As described by Wittchen et. al. (50), pods of unconsolidated rockfill were encountered at Kidd Creek because of the segregation which occurs when filling a bulk stope. One would expect this problem to be lessened in hydraulic fill, especially when high cement ratios are generally used in the lower levels of the stope to form a stable drawpoint plug. In summary, the use of openings in backfill could allow the development of sequences to remove ore at the bottom of mines in a downward retreating

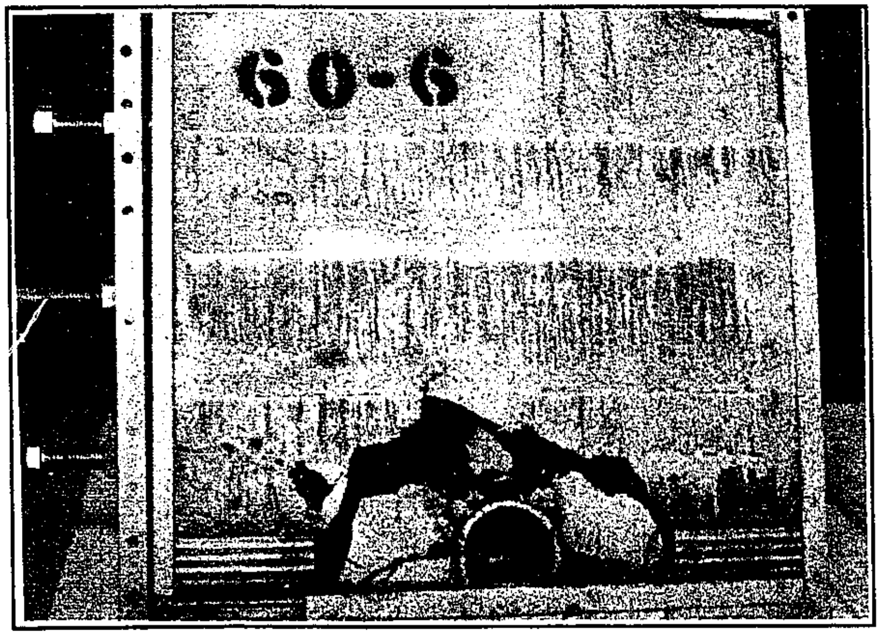

Figure 4.14 The Creation of a Stable Arch in Undermined Fill (52)

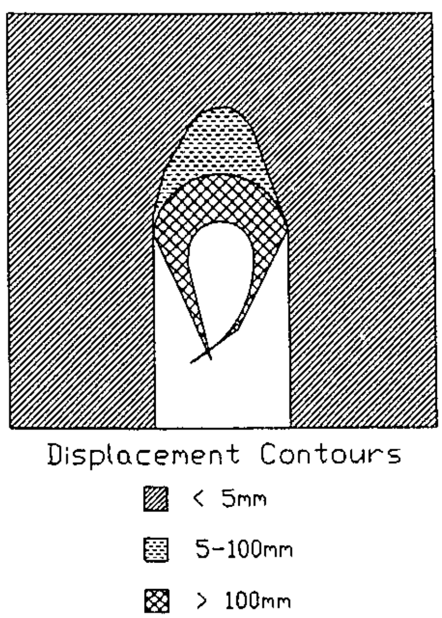

Figure 4.15 The Formation of a Stable Arch in Caving Backfill (52)

manner. This would place the initial high extraction areas at a mid-mine elevation and allow retreat to occur towards both the upper and lower abutments. The somewhat increased costs of the downward retreating sequence would be compensated for by reduced early development at depth and by eliminating the high cost sill removal.

4.5 CONCLUSION

This chapter has described the development of mechanized and bulk mining methods which have led to significant changes in mining practice over the past 10 years. Section 3.2 demonstrated how the requirement to get more productivity from mechanized equipment required a change in a traditional transverse mining sequence in the Sudbury area. The further development of post-pillar and ramp cut and fill mining saw impressive increases in productivity but little emphasis on sequencing, leading to problems in recovering sill pillars.

The move to bulk mining techniques with delayed fill allowed the use of more flexible mining sequences which increased productive capacity and reduced both costs and ground control problems. The 1-5-9 sequence allowed the entire strike distance of a deposit to be placed into production after primary stopes had advanced a maximum of five levels. The traditional sequence would have required that primary and secondary cut and fill stopes reach the top of the ore zone before the underhand cut and fill stopes began to advance downward. Only one half of the backfill needs to be consolidated in the bulk sequence as opposed to the entire backfill mass in the earlier sequence.

Single pass mining sequences provide better management of stress at depth. Important in these sequences is the fact that the vertical dimension of the individual stopes is at least equal to or exceeds the maximum horizontal dimensional dimension of the stopes. This allows more material to be broken in individual blasts of the drill, blast, muck, fill cycle and has all of that material report by gravity to a single drawpoint. This has also allowed increases in productivity. This geometry also simplifies the delayed placement of backfill. Important in ground control, is that this geometry has eliminated the creation of the large horizontal sills of previous sequences which proved difficult to recover and which often gave rise to rockbursts.

These sequences, however, demand more adherence to the long range plan as they lead to localized high extraction rates early in the life of an operation. Ground control problems and dilution early in the life of an operation can more seriously affect the economic viability of the project than the loss of reserves or high cost mining late in the mine's life. Grade control has been made more difficult, especially in lower tonnage mines where a large proportion of the daily production may originate from only a few individual stopes.

The early development of high extraction rates also increases the requirement to monitor ground behaviour so that methods can be modified for extraction sequences in other areas of the mine. The benefit of such programs will be demonstrated in the case study of the Hemlo area mines as presented in the following chapters. Continued research into alternatives such as the creation of openings in backfill is required to allow future sequences to be as flexible as possible.

CHAPTER 5

A CASE STUDY OF THE HEMLO MINING AREA

The Hemlo mining area is located near the northeast corner of Lake Superior, approximately 40 kilometres east of Marathon, Ontario, along Highway 17 which serves as the Trans-Canada Highway in Ontario. The junction with Highway 614 which serves as the access to Manitouwadge is approximately one kilometre to the east of the mine.

The area includes three mines along one orebody as can be seen in the longitudinal section provided in Figure 5.1. The David Bell Mine lies on the eastern extension of the orebody with the Golden Giant Mine of Hemlo Gold Mines being immediately adjacent to the west and the Williams Mine on the western edge of the orebody. Both the David Bell and the Williams operations are owned jointly by Teck Corporation and International Corona, the latter now merged with Homestake Mining Corporation.

This chapter begins a detailed look at mine sequencing planning by describing the difficulties created by operating three adjacent mines on one variable grade orebody. It describes the overall sequencing at the Williams, Golden Giant and David Bell Mines. The planning of extraction sequencing at the David Bell Mine and the interface with operational problems is then described in more detail in Chapters 6 and 7.

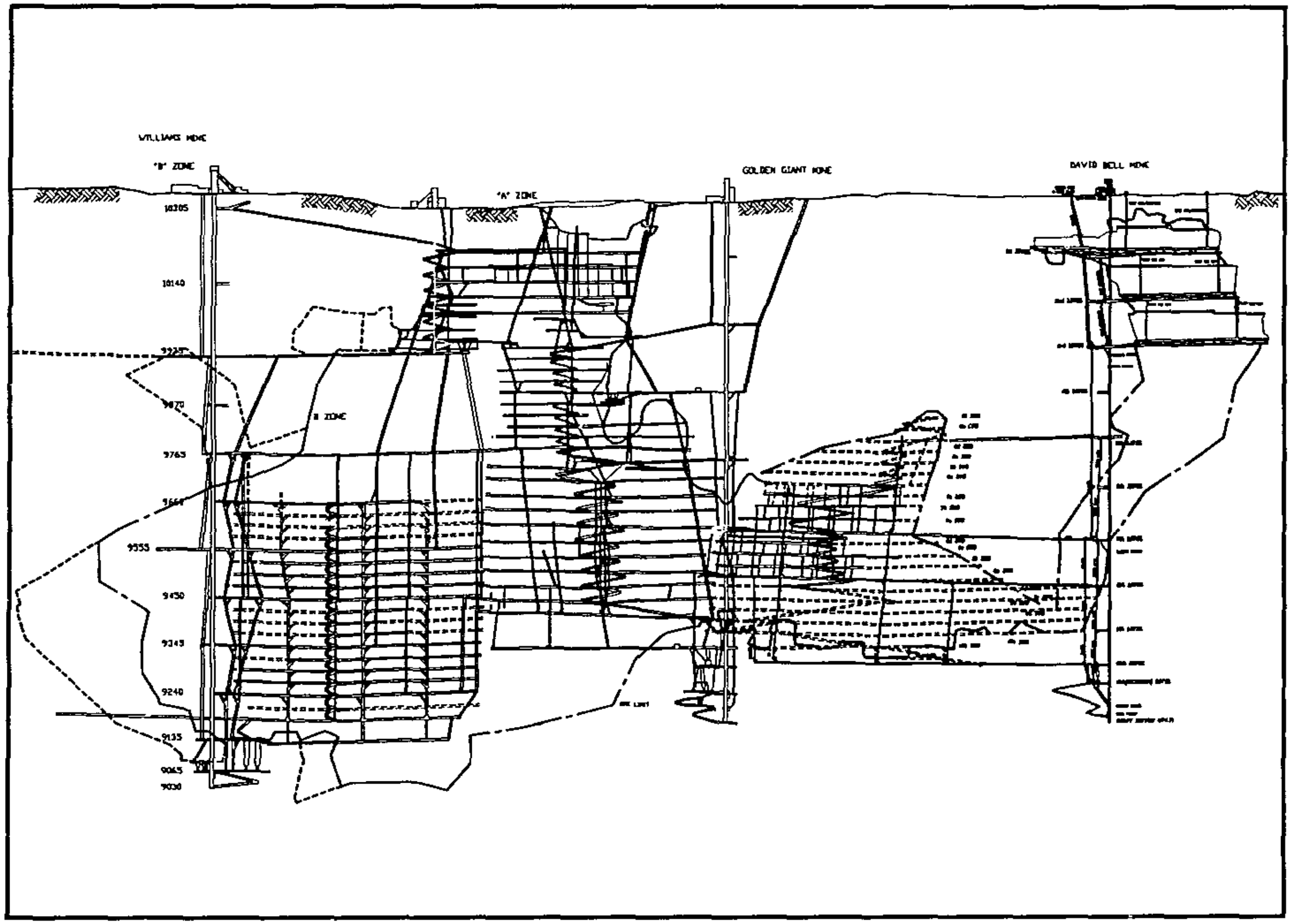

Figure 5.1 A Longitudinal Section of the Hemlo Mines - February, 1992

5.1 OVERALL OREBODY OPTIMIZATION

The claim staking rush which occurred in the Hemlo area, superimposed on some previously existing claims, saw three major property owners initially emerge with a share of the orebody. It has been argued that the way to optimize an orebody is to form one operating company and for the various owners to share expenses and profits. While this did not occur at Hemlo, even if it had the wide differences in tonnage and grade along the orebody would have made the optimization of an extraction sequence a challenge.

The boundaries between the three operations is the vertical projection of an irregular surface property outline projected onto a dipping and plunging orebody. The three operations began as subsidiaries of three companies which had very different priorities. The David Bell operation, 50% owned by the then junior mining corporation, International Corona, had as a main objective the early establishment of cash flow. That objective was aided by the near surface narrow vein zone developed as the first three mining levels.

Noranda acquired the Golden Giant property by agreeing with its owners that it would bring it to production within two years or incur significant penalties. The Williams mine was initially involved in litigation over ownership between the developer, Lac Minerals Limited, and the Teck-Corona Operating

Corporation who eventually prevailed. It had the benefit of the only surface outcrop of the deposit which allowed an open pit operation as underground development proceeded. In the early stages of the underground operation, certain claims not part of the disputed Williams' claims were preferentially mined as the profit from this ore was not subject to the dispute.

With three companies having different objectives, two of which were litigating ownership, mining an orebody with great variance in grade and thickness, there is little wonder that one operating corporation to exploit the deposit was not formed. Without becoming involved in the discussion of the potential benefits of one operation, it makes an interesting academic exercise to theorise on how such an operation would handle the optimization process.

The entire orebody at Hemlo had a combined reserve of about 62.5 million tonnes grading 6.5 grams of gold per tonne. That grade varied by one order of magnitude from the western to the eastern extremities of the orebody. In the same way, the thickness varied by one order of magnitude from east to west. Using Taylor's equation to estimate the optimum production level from such an orebody, the mine would have produced about 12,500 tonnes per day for a period of 20 years.

To produce ore at the reserve grade would likely have seen the operation begin on what is now the Golden Giant property. With this portion of the orebody being thinner than on the Williams property, it would have been difficult to achieve the desired production level without at least two major stoping areas. The open pit operation would have assisted with the early production but the reduced grade in the pit would have demanded that a higher grade be produced from underground to maintain the average deposit grade.

As the mine advanced east and west to the ends of the orebody, a large variation in grade between the areas would have developed. Scheduling from the two ends of the mine would now become critical to avoid wide grade fluctuations in the mill. Even on an hourly basis the gold recovery might fluctuate greatly as the grade varied. In fact, the best approach would likely be two separate mill circuits, with the high grade being sent to a scavenging circuit before combining with the low grade ore for final treatment. If that were the case, two hoisting shafts would be required so as not to combine the product before the mill.

In other words, even if the Hemlo orebody were to be operated as one mine, because of the wide variation in thickness and grade, there would likely have been two very separate operations. Obviously there would have been a savings in capital investment and operating economy of scale, but from a sequencing point of view, certain of the problems which presently exist would still have required solutions. In fact, many existing problems are more a result of the rushed development than the fact that three operations were created.

5.2 SEQUENCING BETWEEN ADJACENT MINES

The existance of adjoining operations is not unusual, the same situation has been described in Kirkland Lake and Sudbury for example. With three adjacent mines operating along a high value orebody, however, agreements had to be reached to ensure one operation did not seriously impact on the safety or profitability of the other. The main cooperative agreements reached in the Hemlo area are described in the following sections.

5.2.1 The Golden Giant Shaft Pillar

Because only Lac Minerals had access to the footwall of the orebody, all other shafts had to be collared on the hangingwall of the orebody. The David Bell shaft had a substantial waste zone between the eastern and western ends of the orebody so no problems were encountered. The Golden Giant shaft was to be sunk on the quarter claim optioned from the Teck-Corona Operating Corporation through what was thought to be a waste window. A small change in the dip of the orebody, however, caused the shaft to be sunk through the widest and highest grade ore on the David Bell property. This necessitated a shaft pillar agreement under which Golden Giant was forced to buy the gold tied up in the pillar and to trade off reserves at depth, an exchange not yet fully developed as exploration at depth is ongoing.

More importantly, by virtue of the grade and thickness, the David Bell Mine chose to first mine the ore immediately adjacent to the shaft pillar. While that was accomplished with no major problems developing, it necessitated a major shaft instrumentation program at Golden Giant and some anxious moments. David Bell has also closely monitored blast vibration in order to avoid any potential liability.

5.2.2 The Williams - Golden Giant Crown Pillar

By virtue of the property configuration, Lac Minerals' open pit operation was directly up-dip from the top of the Golden Giant operation. This necessitated a crown pillar between the two operations. The boundary pillar, as now demanded by Ontario legislation, is a 60 metre horizontal zone on each side of the boundary as shown in Figure 5.2. All mining within this area must be carried out under the close scrutiny of the Ministry of Labour. Due to the projection of this zone on a dipping orebody, the area within the boundary pillar represents a major ore tonnage. In order to transpose this into an acceptable geometry, an agreement was reached between the two parties to establish a 40 metre crown/sill pillar as shown.

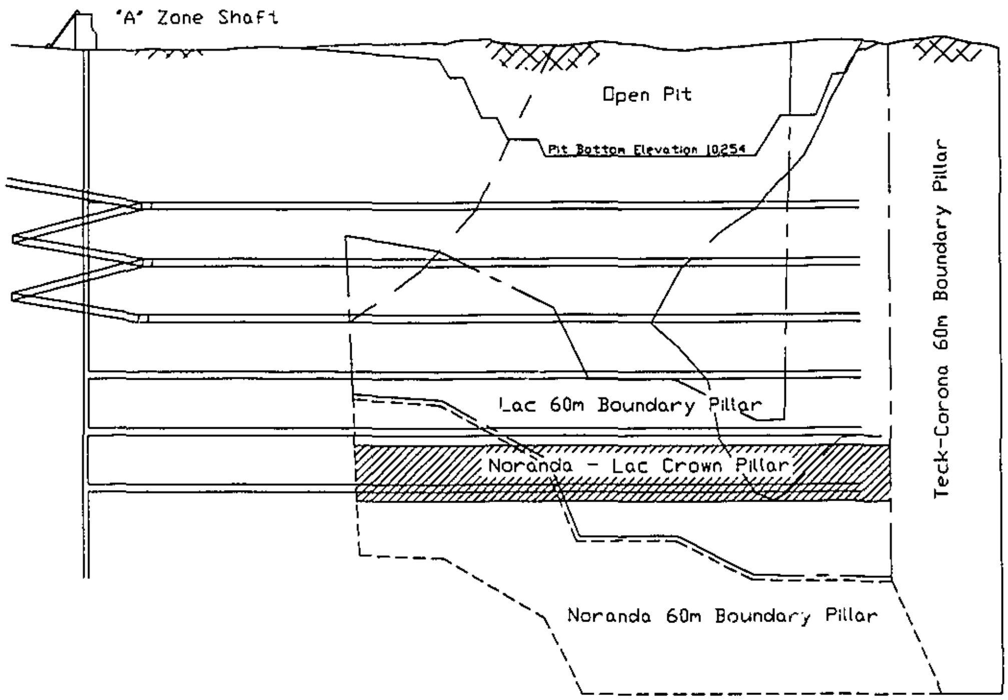

Figure 5.2 The Golden Giant - Williams Boundary Pillar

This represents an exchange of mineral inventory of equivalent value with the material tied up by each party shown in Figure 5.3. This pillar is left in place until such time as good mining practice dictates its removal; sound mining practices must be carried out above and below this pillar to ensure that maximum extraction can be realized. Both parties must exchange mining plans and schedules and allow inspection as required.

The agreement (53) includes, as an appendix, a memo from Golder Associates which attests to the fact that:

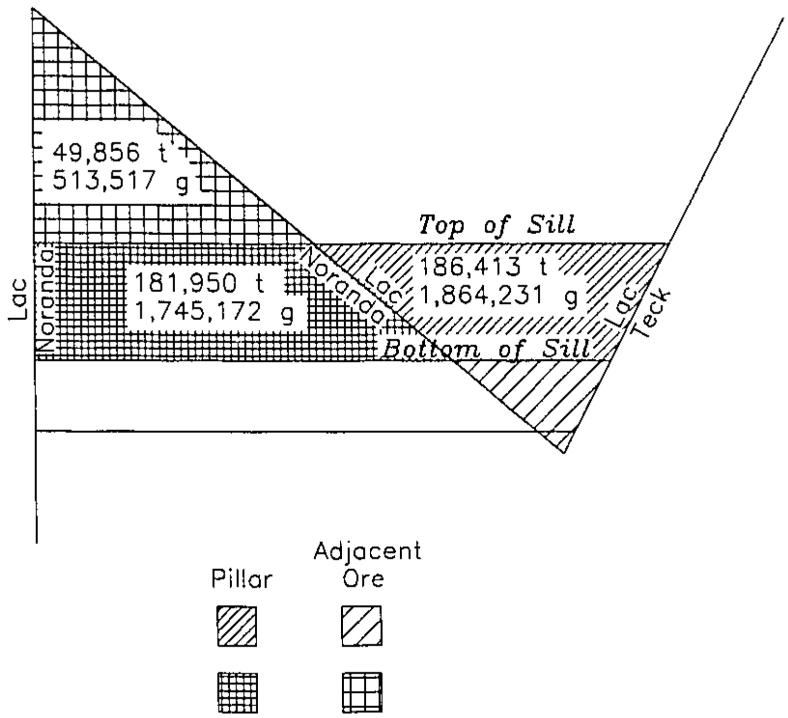

Figure 5.3 Inventory of the Golden Giant Crown Pillar

(a) "the stresses imposed on a proposed 40 m pillar will be quite small, even with total extraction of the orebody above and below the pillar."

(b) "A pillar of this geometry and proposed dimensions is expected to be satisfactory not only in terms of overall stability, but also in not resulting in small scale shear or dilational movements which in turn may result in significant water flows from Lac Minerals' property into Noranda's property."

As described by Bawden et. al. (19), because of delays in filling the upper stopes in the first mining block and poor rock support, about 8 metres of the sill pillar left below this crown pillar caved above the initial stopes on the 4900 level before being stabilized by cablebolting.

5.2.3 Orebody Wide Sequencing

Orebody wide sequencing became important to avoid creating an ever shrinking pillar as a result of activities on separate properties. All of these sequences have been numerically modelled by Golder Associates using the NFold displacement discontinuity code (54). Figure 5.4 illustrates the model for the Williams - Golden Giant interface as predicted for the end of 1991 and Figure 5.5 shows the Golden Giant - David Bell interface (55) for the end of 1990. While the timing and some detail of these sequences have been changed, the overall plan remains valid.

Along the Williams boundary, the diabase dyke is a natural feature and the initial upper level mining at Golden Giant extended to the dyke onto the Williams property. In the intermediate elevations, the bulk of the material up to the dyke will be mined by the Williams operation with one small area above the 4400 level of Golden Giant being mined by that operation. Below that level, Williams will mine to its boundary first and the remaining material between the boundary and the dyke will be mined by Golden Giant.

Along the David Bell boundary, all initial mining will be done by that operation.

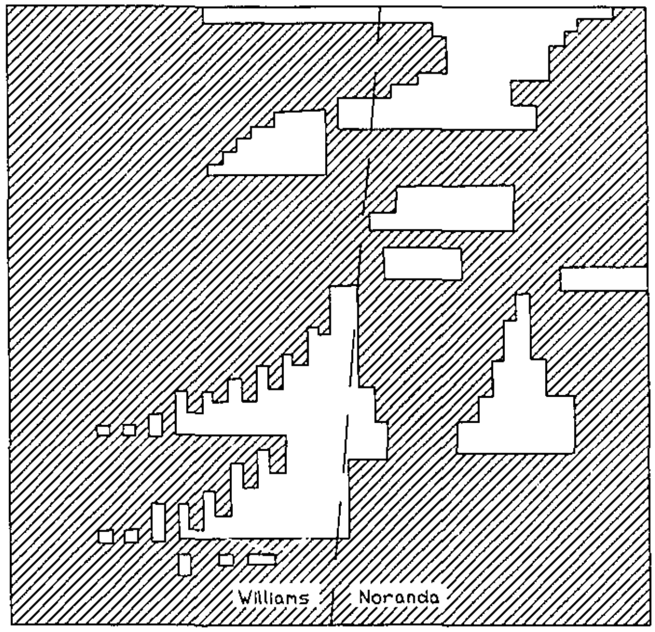

Figure 5.4 The Williams - Golden Giant Boundary Sequence

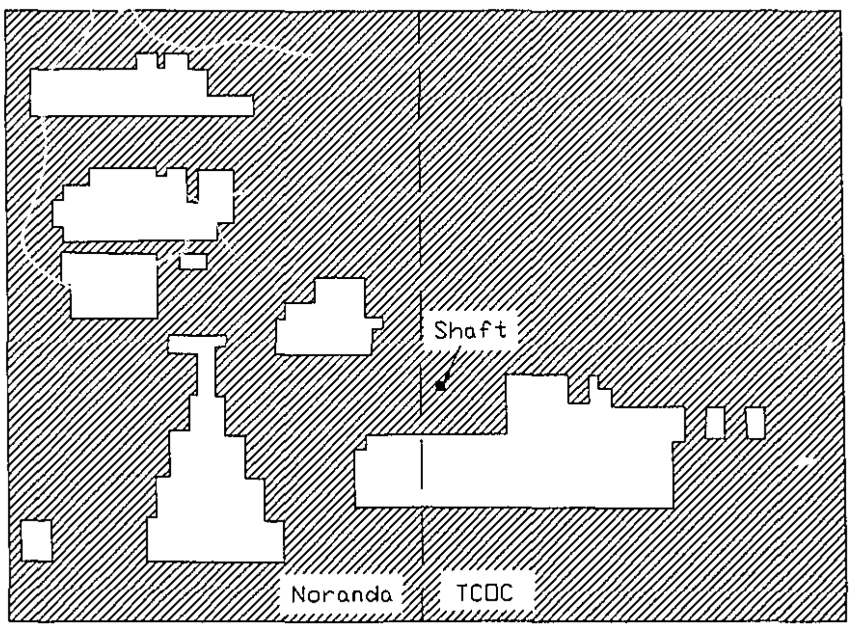

Figure 5.5 The Golden Giant - David Bell Boundary Sequence

5.3 SEQUENCING AT THE GOLDEN GIANT MINE

Totally apart from any interaction with its neighbours, there has been an interesting evolution of sequencing strategies at the Golden Giant operation. As described in 1985 by Hurley (56) and as outlined in Figure 5.6, the mining of block one blasthole stopes was to commence with the one lift above the 4900 level. The stopes and pillars were to be 25 m on strike, the full orebody thickness of approximately 25 m, and 100 m high. Drilling was to be carried out from 25 m sublevels but with the entire height extracted before filling. A second sequence, block two, above the 4750 level, was to use 25 m primaries with a secondary and tertiary stope extraction sequence to remove the 50 m pillar. There was to be a sill pillar between this lift and the lift above. Successive lifts in block three, above the 4400 level, were to progress without using sill pillars and with 33 m sublevels. The objective of this block sequence was to commence production immediately and as described by Pieterse (57) they were successful; 9.6 million grams of gold were produced before the Golden Giant and David Bell shafts were completed to their ultimate depth.

5.3.1 The Mining of Block One

As described by Bawden et al, (19) and Bawden (17), serious ground control problems had already developed during mining of block one, the 100 metre high series of 9 stopes above the 4900 level as shown in Figure 5.7.

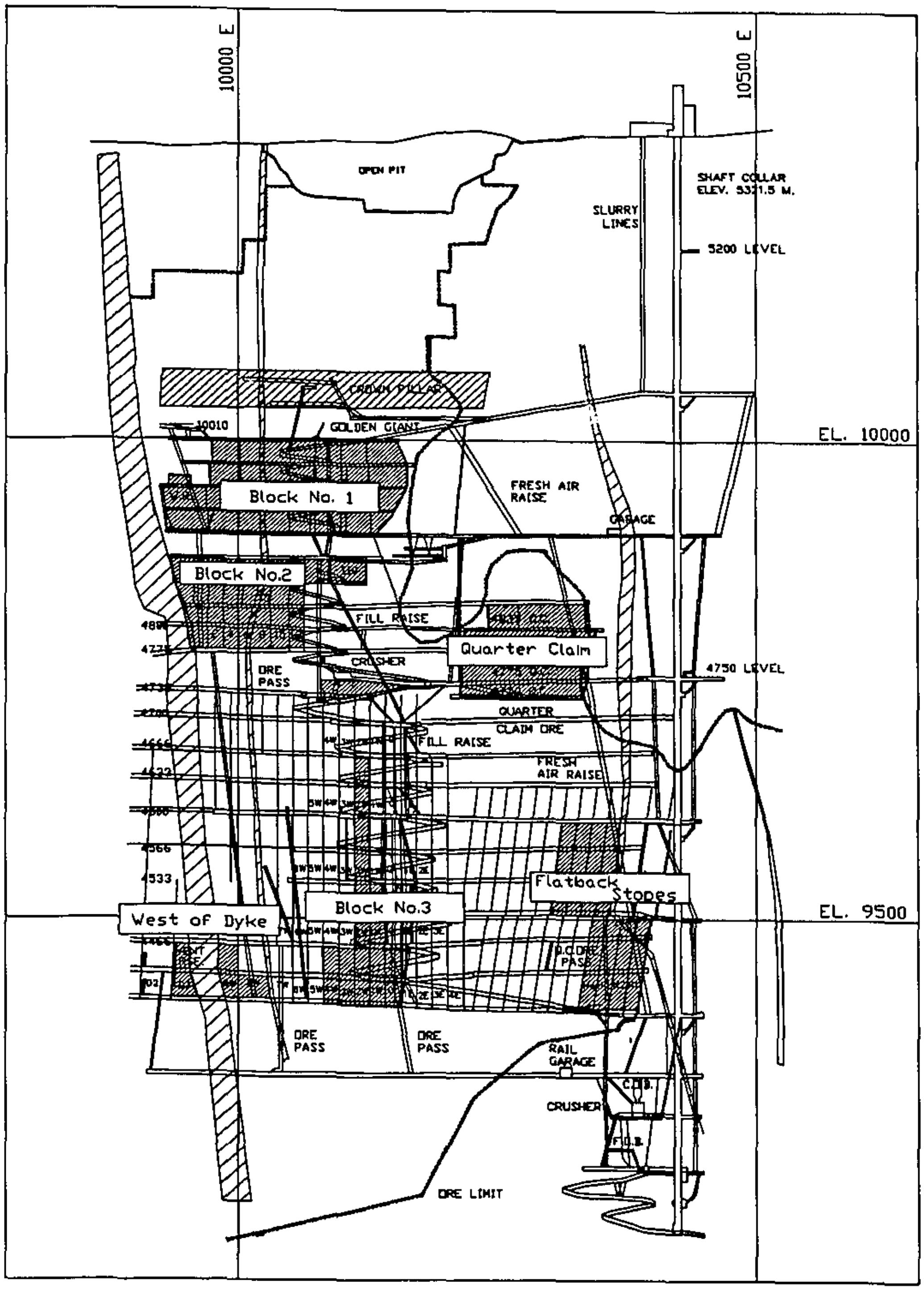

Figure 5.6 Mining Areas at the Golden Giant Mine, from Bawden (19)

This was at a point in the extraction sequence when approximately 1,000,000 tonnes of the total 19,000,000 tonne reserve had been mined. In the 1988 publication, the authors state: "Secondary stopes have tended to dilate open along existing structure during filling of adjacent primaries", and "Also large blocks have, in some cases sloughed off the secondary stope walls, especially along cross cutting diagonal shears" (19). They stated that these events were not surprising since numerical modelling had predicted stress levels of one third of the intact ore strength. Increased stress levels should not normally cause dilation in a pillar except at the pillar edges where bulging into the open stope is occurring. These effects may be the result of the presence of Barite and the dilation is similar to failures which will later be described at the David Bell Mine.

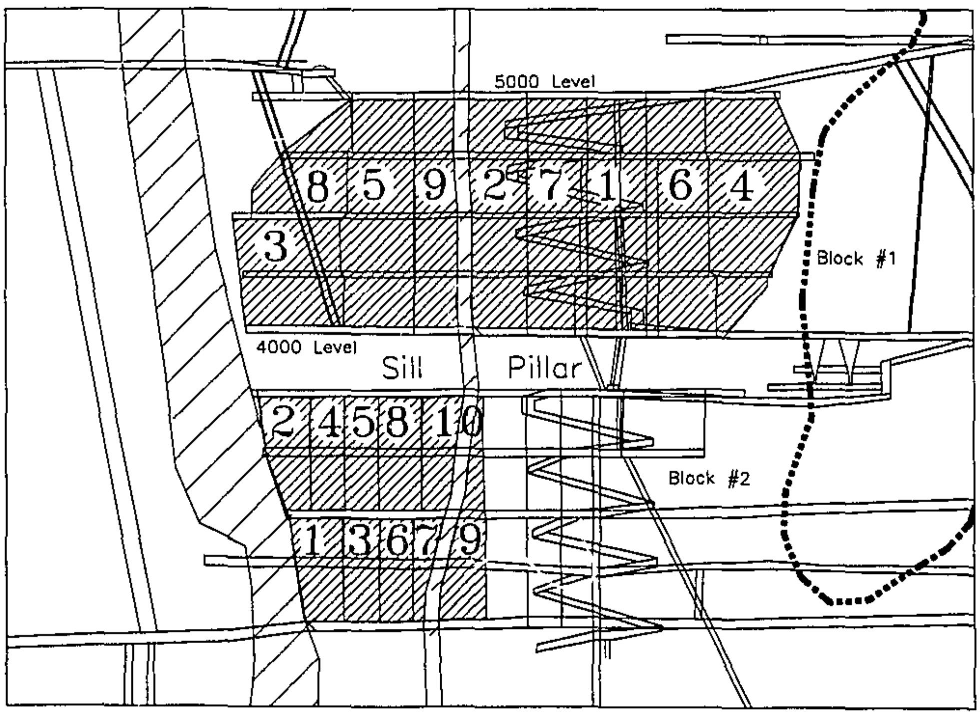

Figure 5.7 Golden Giant Mine, Sequence in Blocks 1 and 2

In the 1993 publication (17), Bawden describes the removal of the three main pillars in block one, stopes 6, 7 and 9. These pillars are described as being in the "yielded condition -- characterized by excessive loose, long continuous cracks in the back and severe deterioration of development openings near the stope end walls." Rehabilitation of the development openings was required and there was difficulty in drilling and loading the 54 millimetre blastholes. The blasting was switched to 165 millimetre holes and the use of the VCR mining method but deteriorating conditions necessitated a mass blast of the entire pillar causing significant damage to the overlying pillar. Closure readings confirmed continuing block closure.

The events in block one indicate several important points. The pillars were designed at a 4:1 height to width ratio and were as thick as they were wide. Considerable schistosity ran along the bedding and "hourglassing" of the pillars was reported, likely increased by blast damage from primary stopes, making the effective strike width of intact pillars much less. There is an indication from the dilation that these pillars had failed very early in the mining sequence even considering the low extraction opening and the relatively shallow depth. The presence of continued closure reinforces this assumption.

The use of homogeneous elastic modelling to indicate pillar stresses seems of little value in such a situation. Any elastic model not allowing post-failure analysis will show increased stress build-up in the pillars. Even if post-failure analysis is available, the failure criteria would be difficult to estimate.

Two main lessons can be learned from the events in block one. First, ground control can rapidly become critical in a bulk mining extraction sequence. Secondly, it is important to monitor the critical parameters so that adjustments can be made in the long range plans for remaining mining areas. In this case, designers relied on numerical modelling for stress prediction, but stress monitoring early in the mining operation would have been valuable.

The problems encountered in block one brought about adjustments to mining practices in subsequent blocks. The production delays, however, demanded alternate producing locations and permanently affected the long term sequencing plans.

5.3.2 The Mining of Block Two

The mining adjustments made in block two as a result of the problems encountered in block one have been described by Bawden (19). The overall development was already in place so changes had to be incorporated into the already defined geometry. The major changes involved mining the stopes with a 15 metre strike length and only a 50 metre height. Also the extraction sequence was changed to a retreat sequence from west to east as shown in

Figure 5.7. Bawden makes an important statement about implementing the retreat sequence as follows:

> "Production requirements are more difficult to achieve with the retreat system since a stope cannot go into production until the one next to it or below it has been backfilled. This leads to a crowding of resources and some inefficiency in a mining block the size of block 2."

In fact that inefficiency again restricted production and increased the requirement to produce from other areas, the flatback retreat area for example. Mining of block two was completed with the only reported problems being the dilation and sloughing of some brow areas.

5.3.3 The Mining of Block Three

The mining sequence of block three was developed as a result of problems experienced in block one and the stability provided by the sequence in block two. The sequence, as shown in Figure 5.8, has been described by Bawden as the pyramidal sequence. Established on the 4400 level, it is designed to advance upwards for six stope heights to the bottom of block two. Stopes are designed to be 15 m on strike, 66 m in height, full orebody thickness of an average 20 m and with drilling carried out on 33 m sublevels. This sequence is very similar to Morrison's "domed" sequence described in Chapter 2.

Bawden (17) describes this sequence as minimizing the development rate

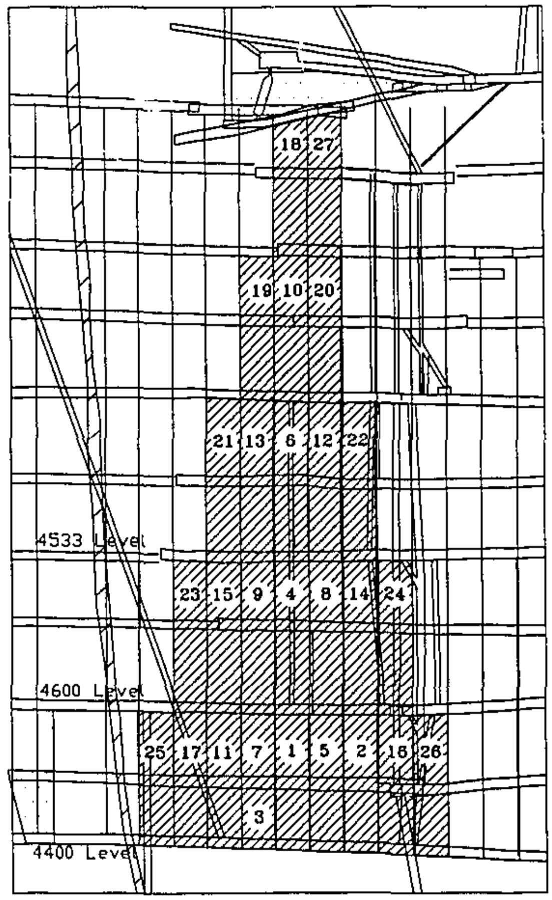

Figure 5.8 Golden Giant Mine, Sequence in Block 3

since the access ramp will intercept each level near the centre of the pyramid and advance east and west as required ahead of the mining. This, of course, requires the mixing of development and production crews on the same level demanding increased ventilation and handling of both ore and waste off the level. It also demands that each level and all services be maintained until mining to the orebody extremities is completed. Costs are increased by requiring the addition of a consolidation agent to all backfill after the initial dome has been created. Establishment of such a sequence initially reduces available working locations. To overcome this problem, mine staff report that some stopes were double lifted with a major failure of one stope wall.

Bawden also describes a very interesting occurrence. In order to provide an additional working location the sequence was started by leaving a pillar between the first two stopes on the 4400 level. This pillar is reported to have failed based on convergence measurements, overdraw from adjacent stopes, and deterioration of the crosscuts to the pillar. This failure occurs in essentially virgin ground after the extraction of only two stopes. Based on experience at the adjacent mines this failure was likely caused by blasting damage and a delay in filling.

The sequence of block three is now established and proceeding with apparent success.

5.3.4 The Mining of the Quarter Claim

A fourth Golden Giant mining area exists on the quarter claim, optioned from the Teck-Corona Operating Corporation, where the thin ore to the west of the shaft was mined early in the mine's life, as shown in Figure 5.6. Mining was by the Avoca mining method on a retreat basis, which allowed disposal of development waste into the mined out stopes at a time when production shaft hoisting was restricted by ongoing shaft deepening activities.

The presence of this area means that eventually the expanding pyramid of block three will have to interact with an opening filled with unconsolidated waste material. This will require a sill pillar to separate the two mining areas.

5.3.5 The Flatback Retreat Area

A fifth mining area exists above the 4500 level which has been mined on a retreat sequence from the boundary of the 8 level at the David Bell Mine. This area was inspected by the author during a site visit in the summer of 1991. At that time, the stope backs had caved to an unknown height over a strike distance of some 75 metres. This points out the problems of flatback mining sequences. It also creates another mining area which will cause an ever decreasing pillar with the expanding pyramid of block three.

5.3.6 Mining to the West of the Diabase Dyke

As mentioned in section 5.3.3, one section adjoining the Williams Mine, above the 4400 level and to the west of the diabase dyke, is to be mined by the Golden Giant operation. That mining area was just starting during the summer of 1991. This isolates the diabase dyke; the stress accumulation on the dyke will be of great interest as the pyramid of block three approaches it from the east. Grant and Potvin (58) indicate that when mining on both sides of the dyke is completed over a vertical height of 100 m the dyke yields and stress shedding occurs.

Their paper also describes the highly stressed environment in which mining in this area will occur. They describe alternate sequences for mining ore immediately below this area as modelled using BEAP and recommend a sawtooth sequence as used at both the David Bell and Williams properties, based on placing the recovery of less ounces of gold in jeopardy .

5.3.7 A Summary of the Golden Giant Sequence

In summary, the emphasis in the early stages of the Golden Giant operation was to achieve early production. With a high grade gold mine and contractual arrangements to meet, the economic benefits of achieving this objective were significant. Problems were experienced with the production sequence chosen for the first block of ore which placed additional pressures on designers to achieve the goal of early sustained production.

The problems which developed in block one were analyzed and the results used to design alternate sequencing strategies for the additional reserves. The use initially of several mining areas, however, requires a coalescence of these various areas as the overall recovery increases. Sill pillars already exist beneath the number one and two mining blocks and one will be required beneath the unconsolidated fill of the quarter claim area. Merging the mining of block three with the flatback area will be carried out in a highly stressed area as the ore to the east, on the David Bell property, has already been mined. As described in the previous section, recovery of ore below the 4400 level adjacent to the Williams Mine is also difficult. This will provide future sequencing challenges or innovative mining methods if the ore recovery is to be maximized.

5.4 SEQUENCING AT THE WILLIAMS MINE

The initial mining at the Williams mine was by open pit and in the B zone. Scheduling of this production was based on objectives involving early production and the ongoing litigation and will not be discussed here. Of interest is the "sawtooth" production sequence now being carried out in the B zone as shown in Figure 5.9.

The mining method is blasthole stoping, with stopes 25 metres high and 25 metres on strike. Where the ore is thicker than 25 metres, it is mined in two passes with the hangingwall stope mined and filled before the footwall stope is removed. The backfill used is cemented rockfill in all primary stopes.

Basically there are two sequences underway. The upper sequence is advancing in a domed manner and at this elevation the Williams operation will continue onto the Golden Giant property to mine all material up to the diabase dyke. The lower sequence is a retreat sequence away from the Golden Giant boundary where the ore on the Williams property is scheduled for mining first. At that elevation, Golden Giant will later mine the ore between the property line and the diabase dyke.

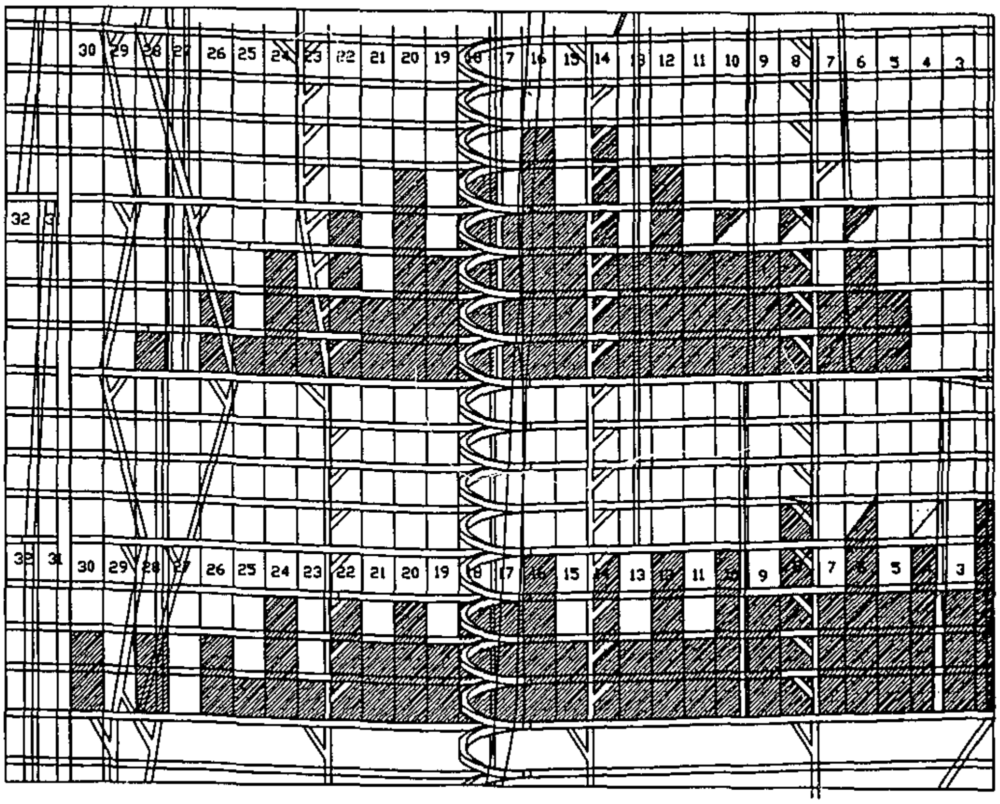

Figure 5.9 Mining Sequence at the Williams Mine

The sequence is designed to have primaries lead secondaries by a maximum of two levels. On one occasion the secondaries began to lag behind the primaries and a fall of ground was experienced from a wall in a garage office immediately overlapping the secondary (59). While it is impossible to equate this incident with stress build-up, it is similar to problems in pillars of 100 metres and 66 meters in height as described at the Golden Giant Mine.

Of particular interest in this sequence is the fact that the bottom requires the removal of a sill pillar underneath the cemented fill of the upper sequence, a problem similar to that at the David Bell Mine which will be described in detail in chapters 6 and 7. The difference here is that all stopes above this sill have been filled with cemented rock fill containing 7% by weight to a height of 25 metres above the sill. All undercuts were reportedly cleaned before filling to avoid the presence of debris at the bottom of the fill. The stope spans are much wider at Williams, up to 40 metres, and the stability of the fill over such a wide span remains uncertain.

The first stope in this sill, 25 m high by 20 m on strike and full orebody thickness, has now been removed and the procedure evaluated in a report by Golder Associates (60). Drilling was carried out from a 4 m overcut drift driven immediately beneath the fill with the fill supported by shotcrete. The fill was reported as having formed an arch 0.5 to 1.0 m above the drift immediately upon blasting and remained stable after shotcreting. Stability of the fill persisted on removal of this first stope.

It is planned to remove the sill on a continuous retreat sequence away from the Golden Giant boundary as this allows individual stope lengths to be adjusted as necessary. Also, in thicker ore, the stopes can be mined in slices on retreat from the hangingwall. In narrower sections it is proposed to drill the ore from the footwall, a method developed at the David Bell Mine as will be described in chapter 7. It is also proposed to measure convergence during the sill removal. This will provide a critical source of data for planning such sequences in future, especially data on the stability of cemented rock fill when exposed to closure strain.

The placement of higher strength fill for 25 m above the sill is another example of the sequence at Williams being planned in advance. The initial plan was modelled by Golder Associates using NFold, and the plan is followed rigorously. All out-of-ore development is in place early. One can walk along the level at the top of the bottom sequence and see the progression of activities from stope overcutting, to drilling, to blasting, to filling. The unique ore handling system using four ore passes, Eagle crushers, and conveyors for lateral transport seems to work well to ensure no bottlenecks exist in the ore flow process. The mine appears to be well planned and the operational execution of the plan is taken seriously.

5.5 SEQUENCING AT THE DAVID BELL MINE

The David Bell Mine, discovered in 1982, began production in 1985. The present mining rate is 1280 tonnes per day. In 1991 it produced 283,128 ounces of gold at a per ounce cost lower than any Canadian operation. The following sections introduce the operation and explain the long range mining sequence in preparation for the detailed analysis presented in Chapters 6 and 7.

5.5.1 Mining Methods

Mining at David Bell began with the use of narrow vein mechanized cut and fill on levels 1, 2 and 3 to the east of the shaft as shown in Figure 5.10. All levels are spaced at 100 metres vertically; these first stopes were silled out on all three levels over the approximate 100 metre strike extension of the orebody. The ore was about 2 metres wide with a dip of approximately 45^0.

Ore from this area was used to supply the mill while development of extraction openings was completed on levels 7 and 8 and the ore transportation system was developed on 10 level. The crown pillars of these three stopes have been left in place with no present plans for their recovery. The early extraction of this ore, however, allowed the establishment of cash flow at the mine before reaching the main part of the orebody in a manner which did not affect the planning of the main ore zone.

The predominant mining method used in the main ore zone is sublevel

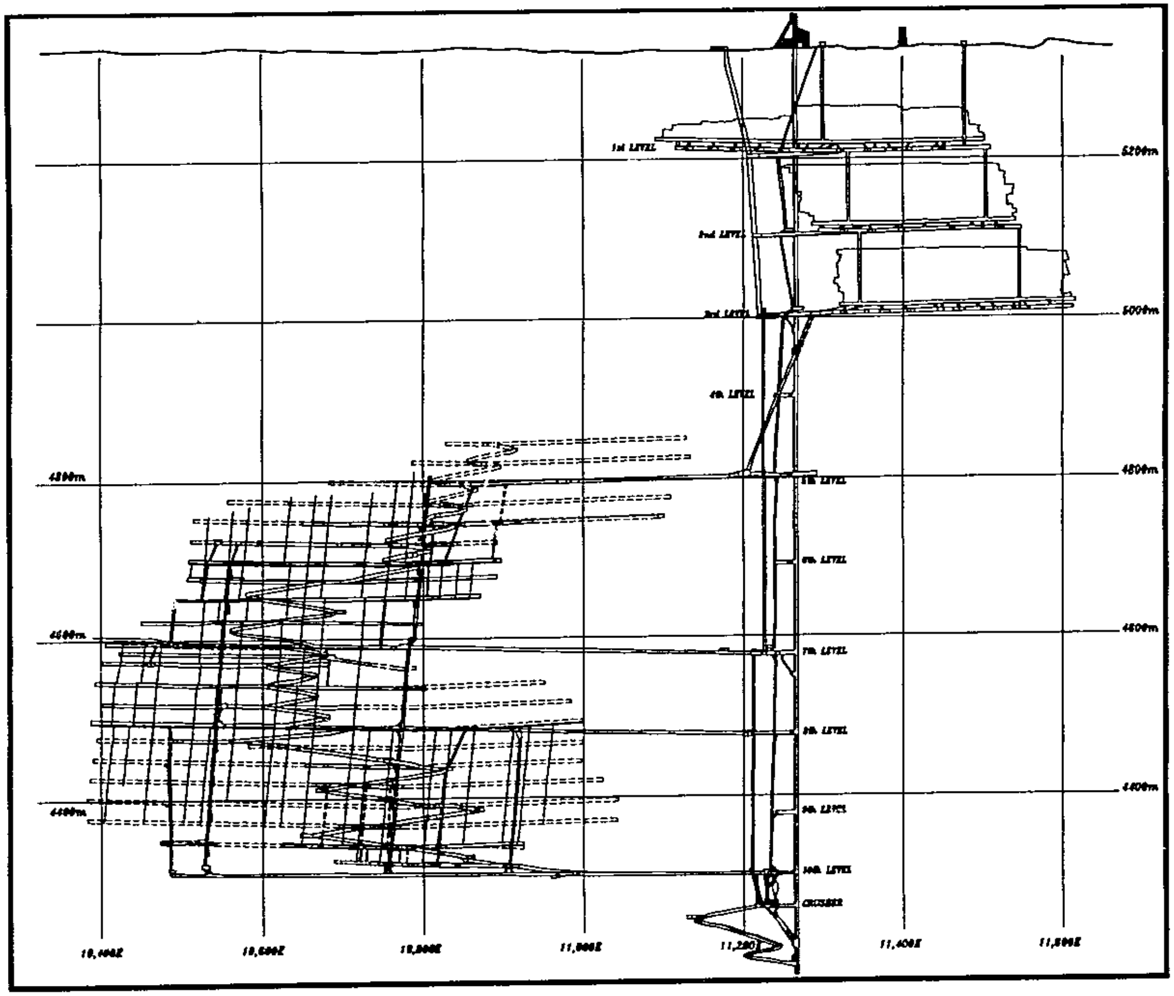

Figure 5.10 A Longitudinal Section of the David Bell Mine

blasthole stoping using a bored slot raise and small diameter blastholes. The sublevel interval is 25 m vertically. The initial design was to use 25 m strike length primary stopes and 25 m pillars. This was later changed to 20 m primaries and 30 m secondaries as will be described in chapter 7. The stopes are the full width of the orebody ranging from 15 m at the west end to as narrow as 3 m at the east end. Development and access are all in the footwall.

Backfill is hydraulically placed and consists of approximately 15% classified mill tailings and 85% alluvial sand. Fill barricades are poured at an aggregate to cement ratio of 20:1 with primaries filled at a 30:1 ratio and secondaries with uncemented material. Initially, normal portland cement was used in the backfill but this has been replaced by Reiss Lime slag activated by 10% normal portland.

Mucking of the stopes is done with remotely operated 6 yard LHD's. There are three ore passes, spaced approximately 75 metres apart, that empty at 10 level into a rail haulage system for transport to the shaft.

An adaptation of the blasthole stoping has seen the elimination of the slot raise with all drilling being carried out in a fan pattern from the footwall haulage drift. This adaptation has been important in ground control management as will be described in chapter 7.

5.5.2 The Main Zone Mining Sequence

During the first two years of the operation, development was completed on the central portion of the main orebody. That development, as shown in Figure 5.10, included: the 5 level as required to connect to a ventilation raise from 7 level, the 7 and 8 levels along with a ramp system connecting the two levels, the orepass system connecting with the main 10 level haulage system, footwall sublevel extraction levels with crosscuts to the orebody on 25 metre spacing, and silling out of the orebody at the sublevels. This development is shown in more detail in Figure 5.11. The development upwards from the 7 level was designed to remain ahead of the advancing stoping operations.

The silling out of the ore at the sublevels was generally carried out 4 metres high to full ore width to a maximum width of up to 15 metres and often preceded the sublevel development in waste. In fact, a decision was made to create an additional 8D sill midway between 8C and 7 level, ostensibly to allow the use of uppers in removing the ore below the previously mined 7 level stopes, but which also supplemented the ore supply during the difficult transition period. This meant that the approximately 21 metre ore pillar planned between the top of the 8C sill drift and the bottom of the 7 level undercut was now reduced to two 8 to 9 metre sills. The early creation of these in-ore sill drifts caused problems leading to ground falls and drill drift rehabilitation.

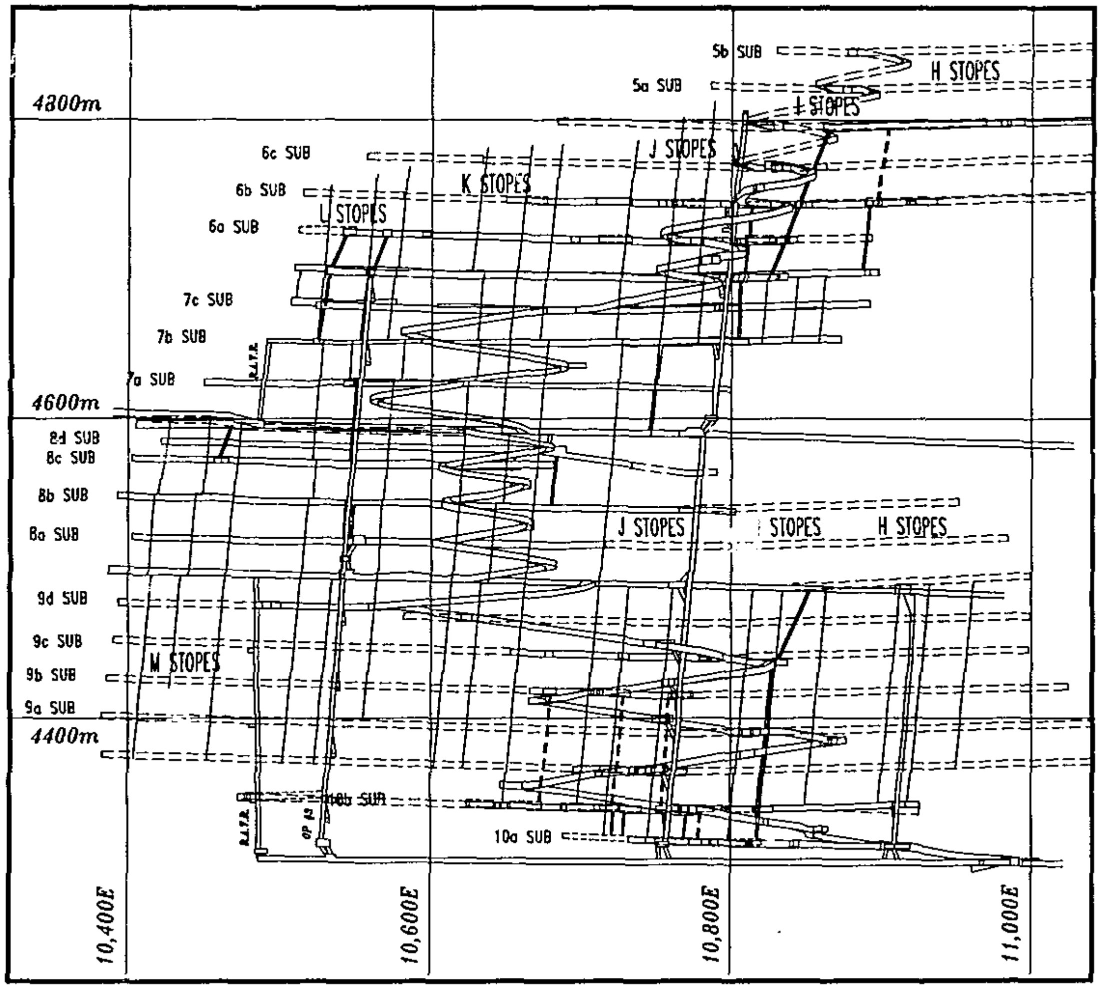

Figure 5.11 A Longitudinal Section of the Present Mining Area

Whether the creation of the 8D sill drift caused problems or in fact caused early destressing of the pillar will be discussed in more detail later.

The initial plan called for mining 50 metre high stopes beginning on both the 8 and 7 levels starting at the Golden Giant boundary and retreating towards the shaft. The offset to the east of the boundary on 7 level defines the location of the Golden Giant shaft pillar immediately above the 8C stopes. Mining was planned to proceed in a "sawtooth" pattern similar to the system previously described for the Williams mine but the extraction sequence sometimes became much more "flatback" because of lagging development. This mine plan necessitated the removal of a pillar between the 8C sublevel and the consolidated tailings backfill in the previously mined 7 level stopes, complicated by the presence of the 8D sublevel. This also necessitated placing consolidated fill in the bottoms of both the primary and secondary stopes of the 7 level.

This sequence of mining above the 7 and 8 levels, especially planning the successful extraction of the sill pillar between the 8C sublevel and the 7 level, which was chosen as the detailed case study for this research. Numerical modelling of the extraction sequence, instrumentation of ground behaviour during extraction, and observation of how operating productivity demands affected the planned sequence, provided an opportunity to better understand the demands of designing and executing a long range production sequence.

5.6 CONCLUSION

The Hemlo mining area is one of the newest and most important mining areas in Canada. The orebody is extremely diverse in terms of varying thickness and grade from one end to the other. The presence of three mines along this orebody introduces a complication of sequencing planning as they must interact along the common boundaries. The three mines have also been influenced by differing management priorities and the results of the efforts of three planning groups can be directly compared under similar geotechnical environments.

Two of the mines had access to an initial ore zone which provided an early cash flow without affecting the sequence of the main orebody zone. Two of the mines have adopted a "sawtooth" mining sequence which sees primary stopes lead secondary stopes by one level. The other mine has converted an initial 100 metre high primary and secondary stope sequence to a pyramid extraction sequence due to severe pillar failure problems. The implications of these approaches will be further discussed in chapter 8 by providing examples of the various sequencing objectives.

There has been a high level of cooperation among the mining operations which share information and have monthly meetings to discuss ground control problems. To better understand and illustrate the problems of developing and monitoring a mine sequencing plan, the sequence for the first two levels of the main ore zone at the David Bell Mine will now be discussed in detail.

CHAPTER 6

NUMERICAL MODELLING OF THE DAVID BELL MINE

The author first became involved in planning at the David Bell Mine in early 1988. John D. Smith Engineering Associates Limited, the rock mechanics consulting firm for the mine, had been contracted to design a fill mat for the stopes on 7 level. The design was to ensure that failure would not occur when the ore beneath was extracted. The design process included extensive physical modelling carried out in the Queen's University centrifuge laboratory as reported by Mitchell (61). While the fill mat was never installed as designed, the stability of the fill on extracting the 8D sill is described in chapter 7.

In support of the centrifuge modelling, the author was requested to carry out numerical modelling to predict the closure, as a percentage of orebody thickness, to which the fill mat would be exposed when the supporting ore was extracted. While the closure prediction was considered critical, insight into likely areas of high stress concentration and possible failure zones would also become available. This work was carried out using the MINTAB (62) displacement discontinuity code and will be described later in this chapter.

To undertake numerical modelling studies of the David Bell Mine, and to understand the ground behaviour, required information on the geotechnical environment. Information assembled from the three mining operations in the area is summarized in the following sections. While not all of this data has been used directly in the numerical modelling studies, RQD for example, it is included for completeness and because the state of the rock mass affects decisions made about the mining sequence.

6.1 THE STRESS REGIME

To adequately carry out predictive stress modelling, knowledge of the in-situ stress regime is essential. In many feasibility studies, this data may simply come from the use of Herget's equations (29) for stress prediction in the Canadian Shield. The initial MINTAB modelling carried out for the David Bell Mine was based on three in-situ measurements taken at the Williams Mine and the Golden Giant Mine by Golder Associates. The virgin stress state normalized for N-fold input (MINTAB requires the same normalized input) as provided by internal Golder reports (54) (55) is as follows:

Stress Component	$MN/m^2/m$ depth
σ_x	0.043
σ_y	0.028
σ_z	0.024 (Gravity)

The model input stress parameters were modified as more information became available and after on-site model validation. Table 6.1, as adapted from Kazakidis (63) provides a summary of stress measurements in the Hemlo area

Table 6.1 A SUMMARY OF IN-SITU STRESS MEASUREMENTS, HEMLO AREA

Hole[1] #	Average Principal Stresses	Magnitude (MPa)	Plunge[2] (dip)	Bearing (azimuth)	Ratio to σ_3
3	σ_1	36.1	01°	98°	3.17
	σ_2	16.0	20°	188°	1.41
	σ_3	11.4	70°	6°	1.00
5	σ_1	12.3	02°	04°	1.98
	σ_2	8.3	40°	96°	1.34
	σ_3	6.2	50°	271°	1.00
1	σ_1	29.0	20°	158°	1.93
	σ_2	24.1	20°	50°	1.59
	σ_3	15.2	58°	280°	1.00
2	σ_1	35.2	5°	34°	1.96
	σ_2	24.2	7°	304°	1.30
	σ_3	18.6	80°	160°	1.00
1080 m	σ_1	37.09	68.3°	291.4°	1.28
Level	σ_2	34.74	18.3°	77.6°	1.19
Williams	σ_3	29.03	11.3°	171.4°	1.00

PRINCIPAL STRESSES

	Trend[3]	Plunge	Magnitude
σ_1	358°	10°	0.0437 (MPa/m depth)
σ_2	093°	28°	0.0299 (MPa/m depth)
σ_3	250°	60°	0.0214 (MPa/m depth)

(1) Holes 3 and 4 located at Williams mine, level 10100 (300 m below surface). Holes 1 and 2 located at Golden Giant mine, level 4600 (720 m below surface).
(2) Positive plunge below horizontal
(3) With respect to mine north
Source: Kazakidis (63)

to 1990. It should be noted that the result from the 1000 metre levelat the Williams mine shows near isotropic results and was considered an anomaly. A series of tests in four holes by Golder Associates (64), using CSIRO hollow inclusion cells and the USBM deformation guage confirmed the results for this area, but the report points out the sensitivity of the results to the modulus assumed in calculating the stress results, especially when the stresses approach hydrostatic levels.

To ensure the validity of data used for this study and to provide data for ongoing analysis of mining of the 9 and 10 levels at David Bell, two in-situ overcoring tests using CSIR cells were conducted in the first week of June 1992. These two tests were carried out in one borehole in the 10B west drift at the mine. The results of these tests along with the data from the core material testing, as provided by Queen's University laboratory staff, are given in Table 6.2. The results are similar to tests 2 and 5 from Kazakidis and to the general theory used in all the models that σ_1 is generally perpendicular to the orebody with a ratio of 1.4 to 1.8 times σ_3, which is steeply dipping and comparable to what would be expected from gravitational loading.

6.2 ROCK MASS CLASSIFICATION

Data on the rock mass was collected by research assistants during the summers of 1990 and 1991 by carrying out joint line mapping in eight different areas of the mine (65). The joints were analyzed for spacing, orientation, and

Table 6.2 IN-SITU STRESS MEASUREMENTS FROM THE DAVID BELL MINE

(Depth approximately 1000 m.)

Test #	1-10B West 1(a) 1(b)	2-10B West 2(a) 2(b)
Unconfined Compressive Strength (MPa)	220.9 244.0 Avg.- 232.5	193.7 200.8 Avg.- 197.3
Young's Modulus (GPa)	55.8 61.0 Avg.- 58.4	67.0 51.9 Avg.- 59.3
Poisson's Ratio	0.26 0.24 Avg.- 0.25	0.28 0.20 Avg.- 0.24
Principal Stress Magnitude (MPa)	σ_1 = 44.6 σ_2 = 25.6 σ_3 = 23.5	σ_1 = 34.7 σ_2 = 25.5 σ_3 = 18.5
Vector Azimuth	4° 100° 218°	359° 267° 137°
Vector Dip	23° 14° 62°	10° 9° 76°

structure as required for the NGI classification system and a modified Mathews analysis was carried out by Leduc (65). Diamond drill core analysis was started by the geology department during the summer of 1991 and will continue in the future. A summary of the rock quality designator (RQD) as given by Leduc is provided in Table 6.3.

TABLE 6.3 RQD OF DIFFERENT ROCK MASSES, THE DAVID BELL MINE

HOLE	RQD F/W	RQD ORE	RQD H/W
6A-1	63.9	87.0	35.2
6A-2	---	80.1	54.3
6A-4	---	---	69.9
6-19	---	60.6	79.3
9D-14	90.2	70.2	---
9D-15	89.5	70.9	94.1
7B-23 (C-ZONE)	84.7	---	---

During both summers, 1335 joints were mapped in 8 different traverse locations and the results as analyzed using the DIPS program are shown in Table 6.4. The RQD values are determined by finding the mean joint spacing for each joint set present, then the value of J_v was found using the formula:

$$J_v = (1/\text{Joint Spacing A}) + (1/\text{Joint Spacing B}) + \ldots$$

RQD was then found using the relationship:

$$RQD = 115 - 3.3 * J_v$$

Table 6.4 JOINT SET ANALYSIS, THE DAVID BELL MINE

LEVEL	# OF SETS	RQD	LOCATION
5 WEST	2 + RANDOM	59.5	F/W
6 EAST	3 + RANDOM	82.7	H/W
7C WEST	3	66.0	H/W
7B WEST	3	63.7	H/W
7 WEST	3	60.4	H/W
7 EAST	2	35.4	H/W
9D CENTRE	2 + RANDOM	70.3	ORE
10A CENTRE	3	63.4	H/W

Diamond drilling information from one hole within 25 metres of the mapping area gave an RQD value of 79.3 which compares well to the mapping value of 82.7.

6.2.1 Joint Set Description

The three main joint sets identified at the David Bell Mine correspond to those identified by the detailed mapping carried out by Kazakidis (63) at the

Golden Giant Mine. His work includes stereographic projections of these joint sets which were not repeated in this thesis. Set A is roughly parallel to the orebody and lithological contacts with the northerly dip increasing from 57° on the fifth level to 65° on the 10A sublevel. This corresponds to the A set as described by Kazakidis as having a dip direction of 009° and a dip of 64°. Figure 6.1 is a west facing photo of the east end of the 6 level hangingwall which shows the steeply dipping A set in the lower left of the photo. Figure 6.2 is a photo of the intersection of the footwall and the back in the K78C stope undercut. The footwall is dominated by the strong A joint set.

Figure 6.1 Joint sets A and C

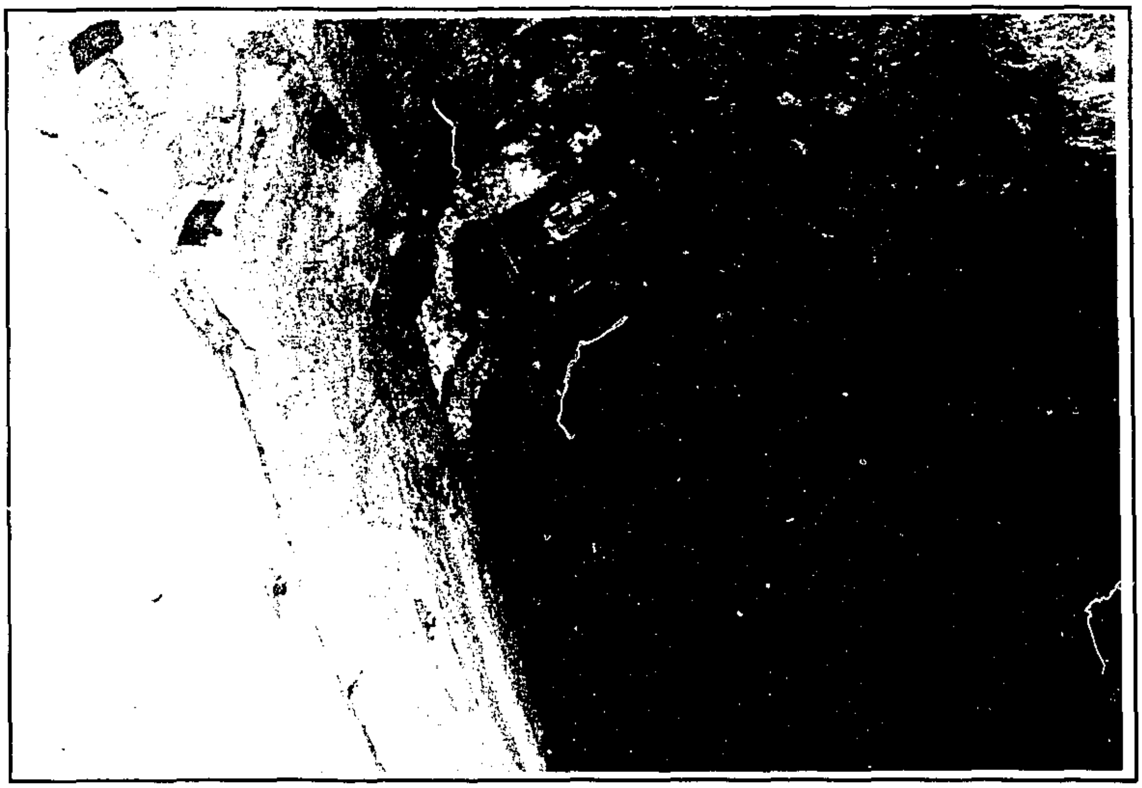

Figure 6.2 Joint set A on Footwall in K78C Stope

The B set is a steeply dipping joint set with a strike of 340^0. Kazakidis describes it as dipping at from 76^0 to 90^0 to direction 074^0 to 122^0. This set varies over the mine and tends to dilate if there is an open stope immediately adjacent to the mapping area. Figure 6.3, a south facing photo, shows the nearly vertical B set on the footwall of the K78C stope, on the 7C sublevel. Also seen here is the oblique C set running from the top left to the bottom right. Figure 6.4 shows the dilation of the B set in the secondary stope's undercut. This picture is of the J76C footwall and was taken facing south.

Figure 6.3 Joint sets B and C

The C set is a flat-lying set with a general 090° strike and 20° dip. Kazakidis describes it as dipping to the south at an average of 15°. The C set is seen in Figure 6.1 as the flat portion of the wall in the middle of the picture. The presence of the shallow dipping C set can cause wedge type failures in the backs of both the undercuts and the haulage drifts. In fact, at the Golden Giant Mine, development headings are driven with a "shantyback" profile to conform to the C set joints.

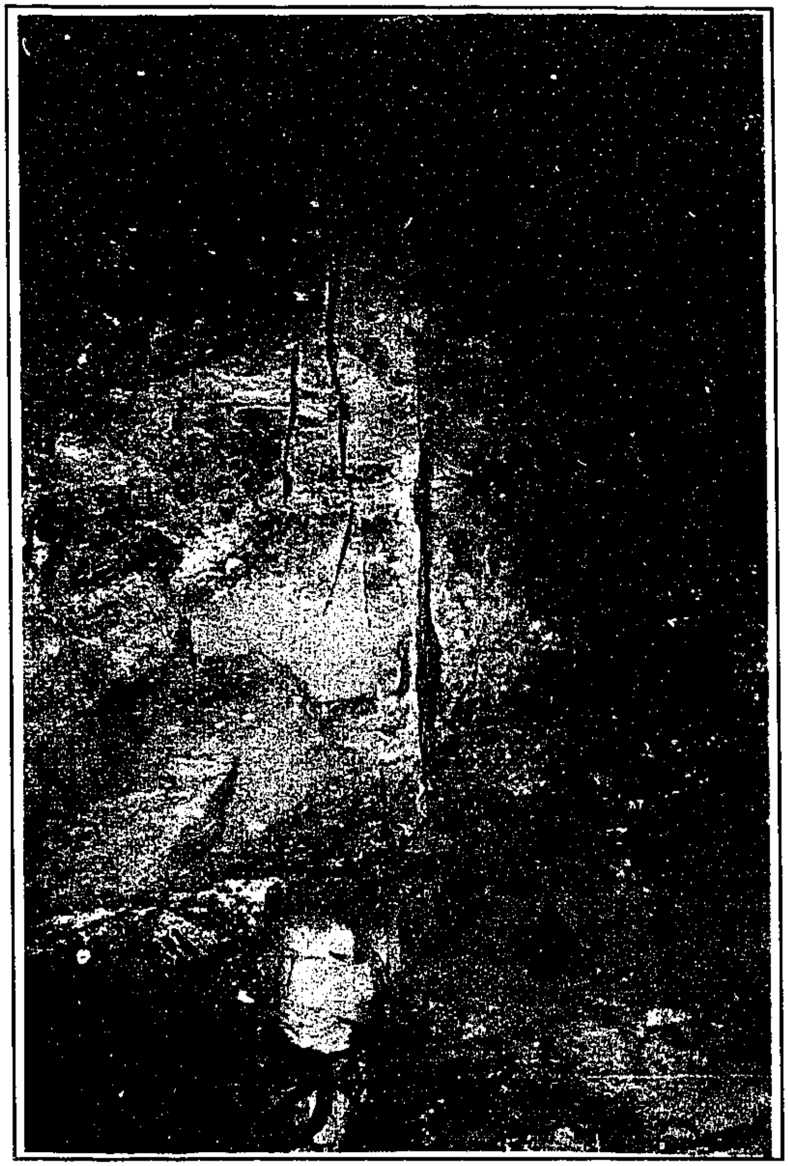

Figure 6.4 Dilation of Joint Set B

As can be seen, the three main joint sets provide almost ideal conditions for block failures. In development openings and ore undercuts the failure would be a wedge failure along the C set joints abutting the A set joints and bounded along strike by the B set joints. In the hangingwall, failure can occur along the

A set joints, sliding out along the C set joints, and being bounded along strike again by the B set joints. When one considers the degree of sericite alteration which occurs along the A set joints, the stress behaviour can be greatly affected by movement which occurs along the jointing.

6.3 ROCK STRENGTH ANALYSIS

Rock sample testing to determine the strength characteristics of the various rock types, particularly the Modulus of Elasticity and Poisson's Ratio to be used in numerical modelling studies, was carried out at the rock mechanics laboratory at Queen's University on three occasions. The analysis procedure meets ISRM standards as described by Archibald (66).

The initial testing was carried out early in 1988. Core came mainly from the footwall of the orebody as most exploratory drilling was carried out from footwall drifts. The data from these initial tests were used for the bulk of the modelling work carried out at David Bell.

No data existed on the hangingwall rocks as little drilling had been carried out from surface into the main area of the orebody and no hangingwall core had been tested. In 1990, a limited test was carried out on core provided from the hangingwall during drilling carried out to install an extensometer on 7 level. In

1991 an additional 10 samples were tested with core provided by ongoing exploration. Table 6.5 gives the results of the 1991 tests and the results from the previous 1990 hangingwall tests. The 1990 samples were undifferentiated in terms of their rock type. Table 6.6 shows the mean values for the various rock types of the hangingwall and for the hangingwall as a whole.

Several samples failed along shear planes, in tests tests 6-2 and 7-1 for example, failure occurred along fine sericite layers orientated at approxomately 50 degrees to the core length and no lateral strain was recorded. The data demonstrates the wide variation which can occur in the rock parameters depending on the amount and grain size of the sericite present in the hangingwall or the amount of barite present in the ore.

The 7C rock type of the hangingwall ranges in strength from 10.15 MPa to 162.19 MPa, the weaker sample being a medium grained quartz sericite schist showing a very well developed schistosity while the stronger rock was highly silicified and weakly schistose. The shear strength of the hangingwall rock is therefore greatly dependent on the coarseness and the content of sericite. While there has been little access available to the hangingwall, when a stope overcut has allowed access after filling, movement along the sericite zones has been noted confirming this lack of shear strength along these planes parallel to the orebody. While these zones may not greatly affect the

Table 6.5 UNIAXIAL ROCK STRENGTH ANALYSIS

SAMPLE #	UNCONFINED STRENGTH (MPa)	YOUNG'S MODULUS (MPa)	POISSON'S RATIO	ROCK TYPE	COMMENTS ON TEST ROCKS
1-1	54.17	13079	***	7D	GOOD SHEAR, HIGH BIOTITE
2-1	91.70	12776	0.17	7F	GOOD SHEAR, MEDIUM BIOTITE
2-2	116.84	16729	0.16	7F	GOOD SHEAR, MEDIUM BIOTITE
3-1	70.04	12708	0.16	7C	GOOD SHEAR, VERY HIGH COURSE SERICITE
4-1	98.50	15098	0.11	7C/F	GOOD SHEAR, FINE BIOTITE
5-1	112.06	18000	***	7C	GOOD SHEAR, SILICIFIED
5-2	162.19	19251	0.25	7C	GOOD SHEAR, SILICIFIED
6-1	19.17	4084	0.26	7C	GOOD SHEAR, VERY HIGH MEDIUM SERICITE
6-2	10.15	4673	***	7C	SCHISTOSITY, VERY HIGH MEDIUM SERICITE
7-1	36.55	9549	***	5D	SCHISTOSITY, HIGH MOLYBNDENITE ALONG SHEAR
1	91.5	20525	0.15	H/W	1990 SAMPLE
2	43.8	13840	0.19	H/W	1990 SAMPLE
3	40.8	9493	0.14	H/W	1990 SAMPLE
4	43.0	10239	0.14	H/W	1990 SAMPLE

Table 6.6 ROCK STRENGTH OF HANGINGWALL GEOLOGICAL UNITS

HANGINGWALL ROCK TYPE	UNCONFINED STRENGTH (MPa)	YOUNG'S MODULUS (MPa)	POISSON'S RATIO
7D CALC-SILICATE RHYTHMITE	54.17	13079	0.07
7F METASILTSTONE	104.27	14753	0.17
7C META-ARKOSE	92.50	13828	0.19
TOTAL	78.64	13818	0.16

transmission normal stresses, they may have a thickness up to several mm with the grain size of the sericite ranging from 1 to 5 mm. To estimate a single elastic strength parameter for the hangingwall as input for numerical modelling would be very difficult considering the variable thickness and consistency of this sericite zone. In fact, movement along these sericite zone is a likely cause of secondary pillar destressing as discussed in section 7.4.1. A light coloured band of sericite in the hangingwall of the M84D stope is shown in Figure 6.5.

The other problem mineral which exists in the ore zone is barite. The whitish barite is found in bands up to 1 metre wide which contain up to 75% barite. These bands are generally found in the wider western part of the orebody and have been associated with falls of ground in the backs of the ore undercuts. Figure 6.6 shows one of these barite-rich bands in the back of the undercut in the K78C stope, on the 7C sublevel. The band is the dark area in the middle of the picture with few rockbolts in it. The band, which appears dark because of staining from the mercury mineral cinnabar, has failed 3 cm up into the back. The failure of these bands occurs slowly upwards with small rocks falling out. As the failure advances higher into the back of the undercut, a brow is created in the adjacent rock which can then fail into the undercut.

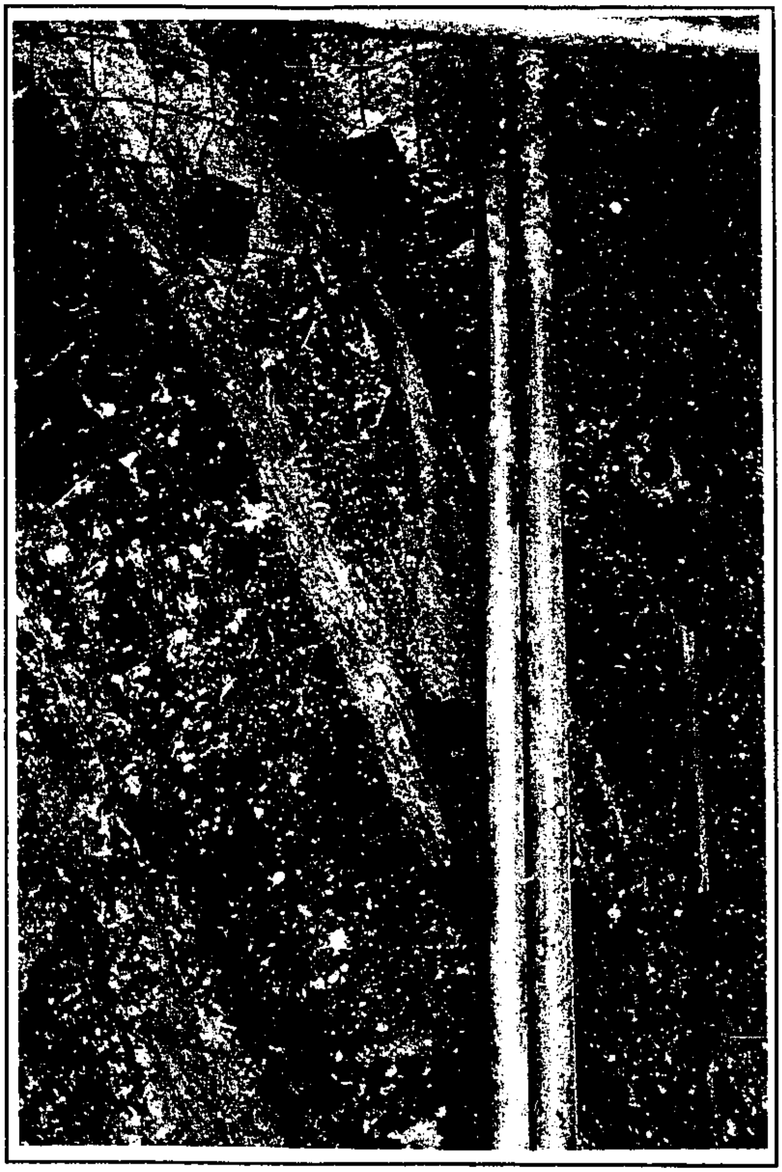

Figure 6.5 A Sericite Band in the Hangingwall

Figure 6.6 A Barite Rich Band in the Undercut of K78C Stope

Again, a problem exists in estimating the strength parameters of the ore zone. The approach which was taken in the numerical modelling was to use a weighted average assuming about 15% of the ore was barite-rich. In reality, it could be argued that the ore zone is only as strong as its weakest part, especially when that zone tends to squeeze and unravel as described above.

It is felt that both of these occurrences probably contribute to the fact that secondary stopes tend to destress after the primaries are removed as discussed in Chapter 7.

6.4 THE INITIAL MODELLING WORK

As described previously, the initial modelling work by the author was in support of a project to design a 7 level sill mat to provide fill stability when the 8 level stopes came up from beneath. Based on design principles supplied by Mitchell (61), it was felt that exposing the cemented backfill to more than 2 % closure would cause instability. This work (67) was completed at the time the first stopes were being excavated on the 7 level. Since this numerical model had not been validated through comparison to actual mine conditions, the results were considered to be relative only.

6.4.1 Choice of Numerical Model

At the outset of this modelling work, it was decided to use the displacement discontinuity model which is extensively used in Canada under various names, for example, MINTAB, DZTAB, ExamineTAB (18). The MINTAB code used in this project was provided to the author by CANMET (62) as adapted by them from the original South African code (68) to allow the inclusion of backfill and the capacity to restart the program after any step. This code also forms the basis of the NFOLD program, used by Golder Associates, which in addition allows overlapping orebodies and provides a post failure capability. All models were run on the Queen's University mainframe computer.

The following attributes of a MINTAB based program provided the justification for choosing this model for this project which involved the planning of an overall mining sequence:

1. It presents the mine in longitudinal section, the orientation of interest, and allows stress loading perpendicular to the plane of the orebody. All three stress directions are applied in the pseudo three dimensional model.

2. It allows a progressive mining sequence to be developed so that intermediate results can be evaluated.

3. It presents the results in longitudinal section to allow for ease of analysis, for example, by using contouring packages as shown in Figure 6.10.

4. It allows the use of three materials: the pillars, the ore and the backfill. In a tabular orebody such as the one at David Bell, the pillars and ore are generally the same material.

The alternatives to the use of MINTAB were generally two dimensional boundary element or finite element programs. Two dimensional programs are best used in assessing an in-stope stability problem, the sill pillar between two cut and fill stopes for example. While the model assumes that the sectional geometry projects infinitely into the plane of the model, in the case of a 60 to 80 metre long cut and fill stope this is a valid approximation. They are of little use in depicting the geometry of a vertical mining sequence such as those used in the Hemlo area where, if the section is taken through a primary stope, the secondary stope 10 to 15 metres away in both directions is ignored.

At the time of this work, the development of three dimensional models was underway. The three dimensional boundary program, BEAP, had been developed by CANMET but pre-processors and post-processors were not available. Three dimensional finite element programs were available on main frame computers but access charges were high and the learning curve long. The use of three dimensional models on PC based systems as described by Zhang and Mitri (69) was unavailable. Even now, the complexity of a 3D model required to evaluate a mining sequence requires so many elements that computing resources become a limiting factor. These problems are discussed in more detail in section 6.8.

6.4.2 Model Description

The initial analysis involved creating an 11 step extraction model based on the five year mine plan. Because the orebody in the area of interest varies in thickness from 12 m on the west to 6 m on the east and since the percentage closure was significant to the stability of the cemented sill mat, the model was run at thicknesses of 6 and 12 m. In addition, a third model was run at 6 m with 5 m of high quality backfill placed in the position of the sill mat.

The model, the portion from 8 level to 6 level being depicted in Figure 6.7, also outlines the stope numbering system as discussed in various sections of this and the following chapter. It should be noted that where the stope numbering system at the mine was changed at the latter stages of this project, the old numbering system has been maintained for consistency. The model consisted of an 80 by 80 matrix of 5 m square elements extending from 580 m below the surface (50 m above the 6 level) down for 400 m (50 m below 9 level). The model extended 400 m eastward from the Golden Giant boundary on the west. Mining Activity extended from the first six stopes above 9 level up to 6 level with all excavations being 25 m wide and 50 m high, in effect all stopes being double lifted as initially planned.

As is the convention in MINTAB, all stopes from a previous step were backfilled prior to excavation of the stopes in the present step. This means that

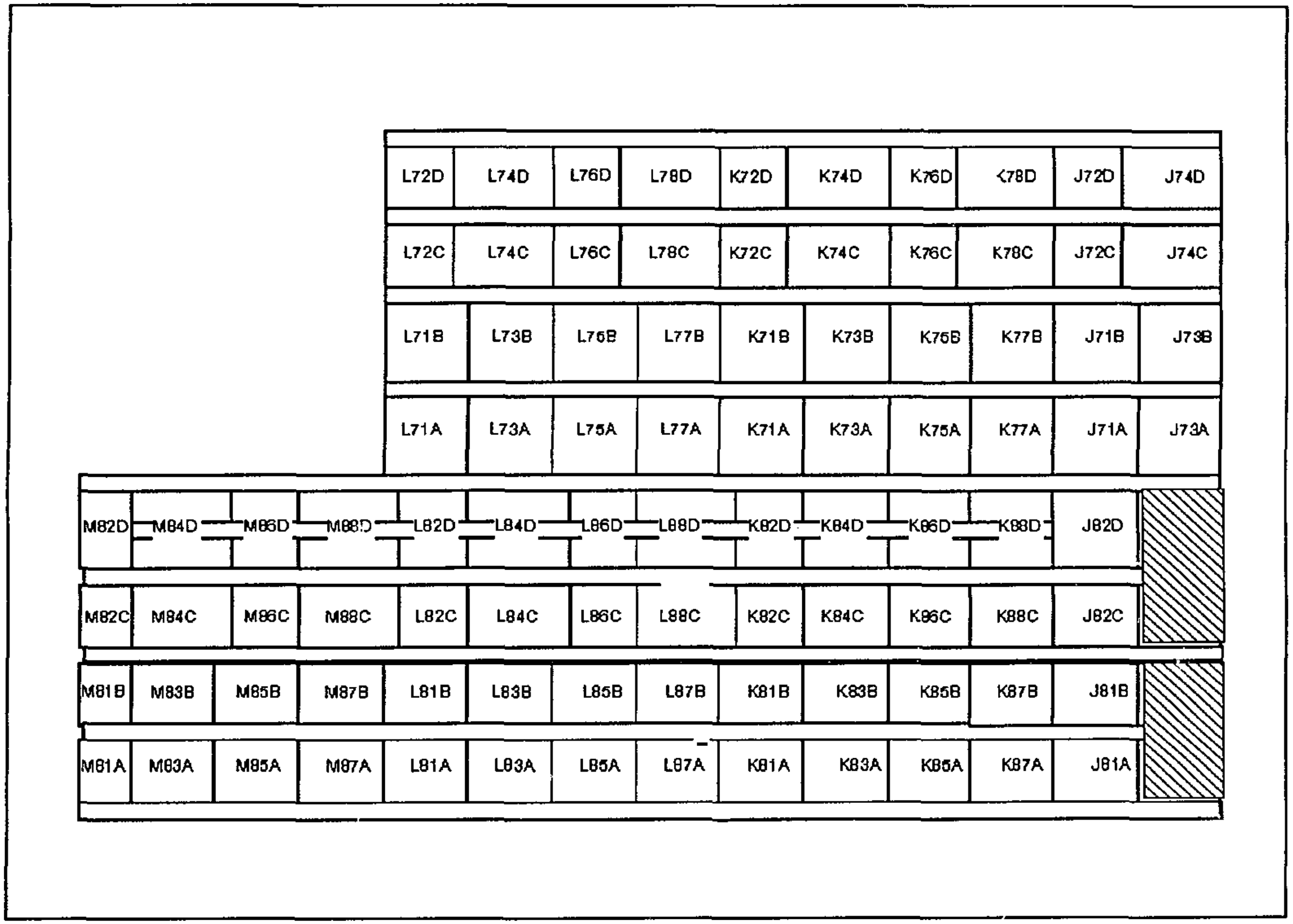

Figure 6.7 Stope Numbering at the David Bell Mine

the model has reached convergence to the pre-determined level, the default is 1 % or twenty iterations, before the backfill is introduced. This means that no convergence occurs with the backfill present until the next step is excavated. Whether this represents the reality in underground excavations is difficult to evaluate. This subject of timing models to the closure in the real world is further discussed in section 6.4.3 and has been included as a recommendation at the end of this thesis.

The input parameters for this modelling, again based on the Golder Associates input parameters from the William's Mine, were as follows:

In-situ Stress State	As reported on page 164
Young's Modulus	25,000 MPa
Poisson's Ratio	0.25
Backfill Modulus	50 MPa

The 11 step model was designed to show conditions at the most critical stages of the planned extraction sequence. Steps 1-4 are depicted in Figure 6.8 with steps 5-8 depicted in Figure 6.9. The graphics used in this thesis to illustrate the various model steps were created using ExamineTab whereas the initial work was carried out using MINTAB. The final three steps involve extraction of the final 8D stopes remaining in step 8 and six stopes above 9 level. The steps are described as follows:

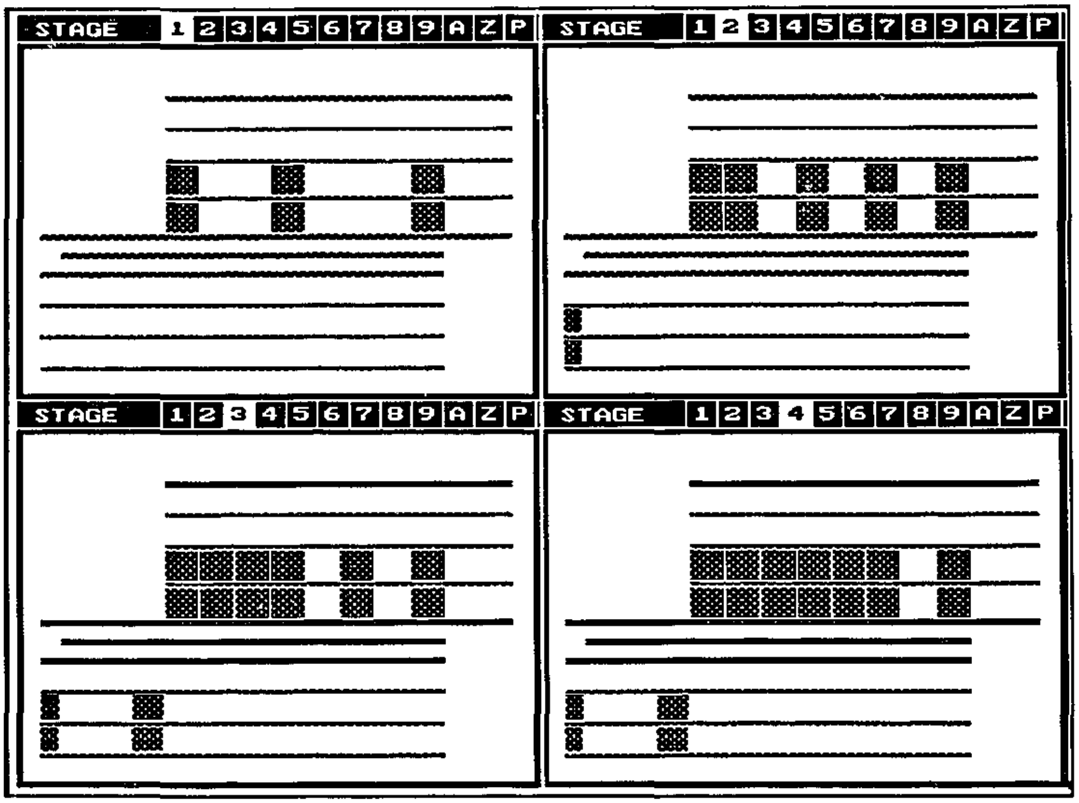

Figure 6.8 The Five Year Plan, Steps 1 to 4

Step 1. The first six stopes above 7 level.

Step 2. The first free fill face exposure, a 50 m wide opening, and a first single width pillar. The first stope was excavated above 8 level.

Step 3. The first four stope width excavation (100 m) and the first time backfill is exposed on both sides.

Step 4. The first six stope excavation adjacent to a 25 m pillar and the worst loading condition on a fill wall above the 7 level.

Step 5. The removal of the remaining stopes between 7 and 7B and between 8 and 8B, the first two stopes were excavated beneath the Golden Giant shaft pillar and the first stopes up to 6 level. This would be the condition before the first stope was removed beneath the sill mat.

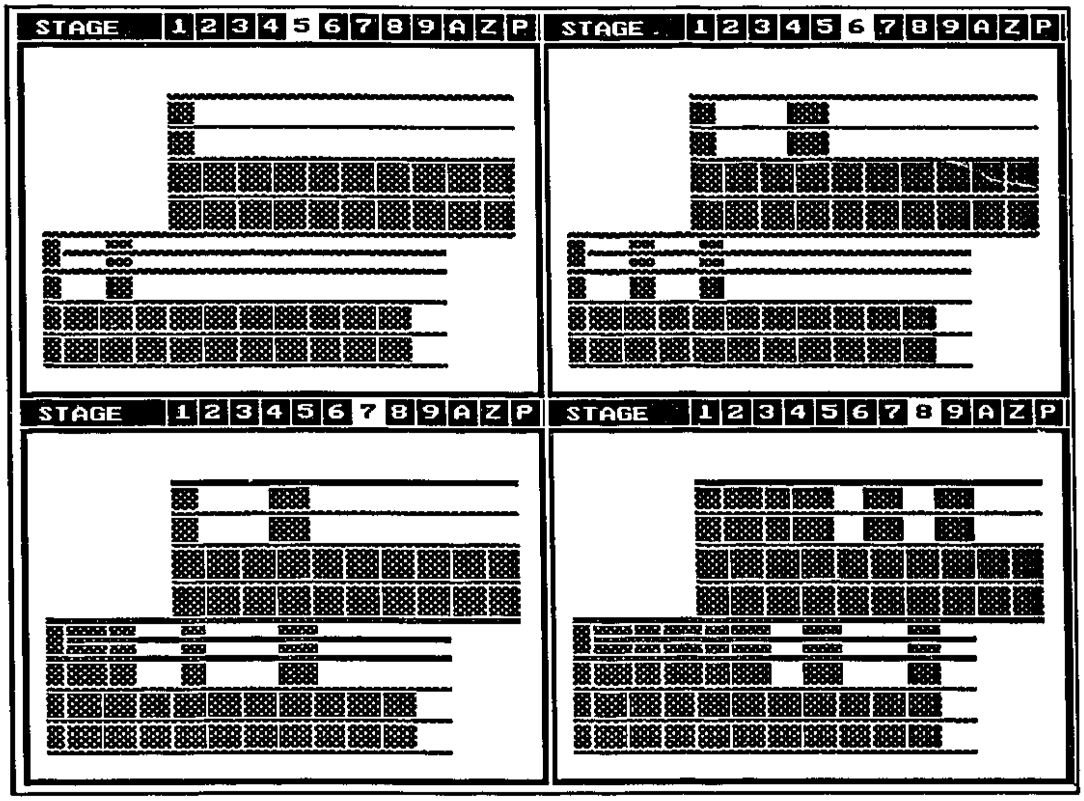

Figure 6.9 The Five Year Plan, Steps 5-8

Step 6. Excavation of the first stope beneath the sill mat.

Step 7. The second stope beneath the sill mat.

Step 8. The first double width exposure of the sill mat. The first stope above 9 level.

Steps 9, 10 and 11. Removal, advancing from west to east, of the remaining stopes beneath the sill mat to observe the increase in closure and the stress expected for the remaining pillars.

The two main output parameters from MINTAB models are the predicted closure which will occur in an underground opening and the stress accumulation normal to the orebody in the rock surrounding the orebody. Other parameters such as the stresses along the orebody and the energy released during the excavation are also calculated but were not used in this analysis.

6.4.3 Predicted Closure from MINTAB

The discussion of the closure results is based on the model having an ore thickness of 12 metres as this model predicted the greatest closure, although only marginally greater than the 6 metre model. That is predictable as the absolute stress applied to the model in longitudinal section is the same but the reaction occurs in a pillar of greater thickness to size ratio. In terms of its effect on free standing fill walls or on the sill mat, however, damage based on percent strain would be more severe in narrower ore.

In the elastic models, closure on the individual stopes of steps 1 and 2 was predicted to be about 20 mm. It is difficult to relate the modelled closure to real time. Whether this closure occurs before backfill placement is critical in assessing if any support is gained from the fill and how much damage may be caused with the additional closure from excavation of the adjoining stope. Backfill only assumes load if closure occurs after its placement.

Maximum predicted closure increases to 54 mm with the two adjoining stopes of step 2, and to 68 mm for the four open stopes of step 3. Closure increases gradually, as the open excavation increases, to reach a maximum of 183 mm for the 12 metre model as depicted in the contour plot in Figure 6.10. This maximum closure is shown to occur at a point in the extraction sequence which would overlap the 8D sill area on its complete removal. The maximum closure predicted for the 6 metre model was 177 mm.

This represents 1.5% strain for the sill mat in the thicker ore and 3% in thinner areas. Above 2% strain would likely cause failure in high cement content backfills. With a retreat sequence of sill pillar removal, however, the stope below the sill pillar would be extracted and filled before the maximum closure would be reached.

To determine the effect of the backfill in limiting closure, the 6 metre model was run without backfill throughout (it is impossible to place backfill with two parameters at different points in the model), but with very high quality backfill in a row of elements along the location of the sill mat. This in effect equates to a 5 metre thick layer of high quality concrete along the 7 level. The results as shown in Figure 6.11 show the ripple effect of the reduction in closure along the 7 level sill mat.

6.4.4 Measured Closure

This data demonstrates the importance of predicting closure as input to

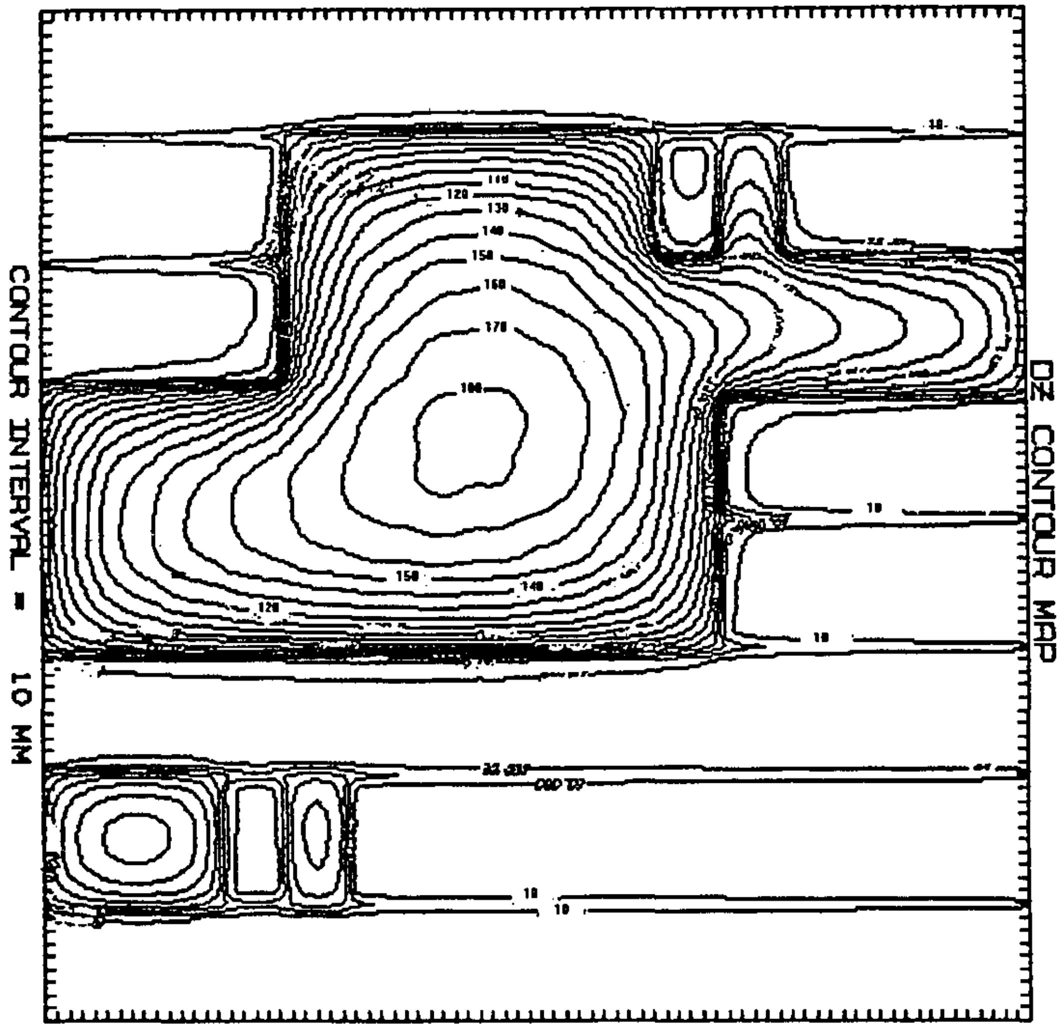

Figure 6.10 Closure at Step 11, 12-Metre MINTAB Model

the design process but also the difficulty of predicting the timing or validating the magnitude of the predicted values. Seven grouted, pre-assembled multi-anchor borehole extensometers were installed early in the life of the operation in an attempt to monitor mining induced ground movements and to validate closure predictions.

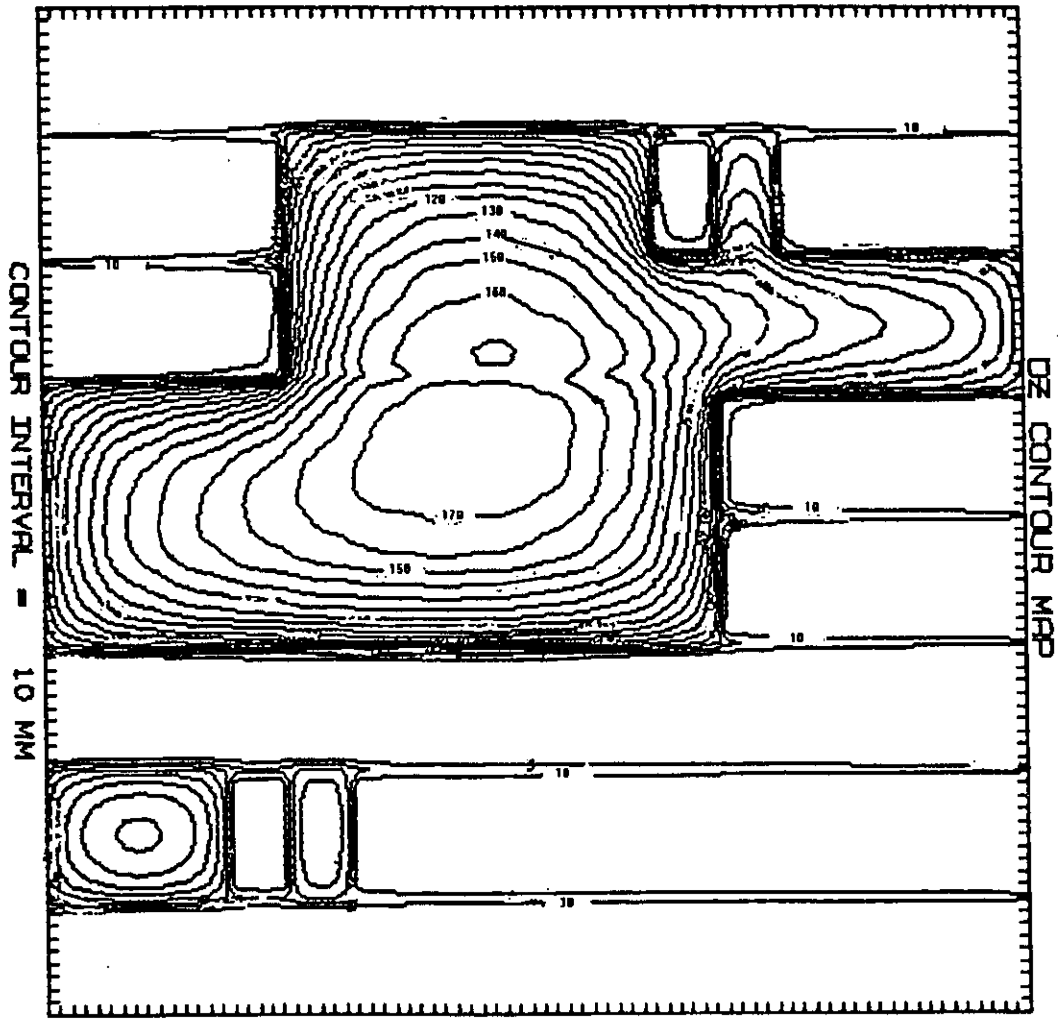

Figure 6.11 Closure at Step 11, Backfill Only at Sill Mat, MINTAB

Four of these extensometers were installed in the K75 A and B stopes immediately west of the 4th and 5th stopes to be mined. Two were installed from the overcut to the undercut to monitor vertical movement and two across the ore to measure closure. Some heaving of the floor of the K75A overcut was noted as the K77 stopes and the K73A stope were removed. The extensometer in the K75B pillar was monitored from October 21, 1988 to July 26, 1989, with the K75A stope being blasted on May 20, 1989. While up to 25 mm of movement were measured in total, the majority of this did not occur until July 12 to 26, 1989. The across ore extensometers gave conflicting results. The K75A extensometer monitored from October 21, 1988 until the pillar was blasted on May 20, 1989 showed very little movement with both compression and expansion taking place. The K75B instrument showed very little reaction to the K75A blast and again showed movement in both directions.

The other three extensometers were installed across L73A (stope 22) and across M87 A and B. Again, the results were inconclusive. The initial interpretation of this data was that slippage of the heads had occurred. With the subsequent observation that destressing of the secondary pillars was occurring, as discussed later, this confusion with this extensometer data may have been as a result of differential dilation of the pillars.

This points out the practical limitations of gathering valid closure data in bulk mining operations, especially in the absence of hangingwall development. The ideal condition would be to have a surveyed anchor point on both sides of the orebody some distance from the ore. Heads on the walls of the stope would then differentiate stress related closure from relaxation of the rock layers. Installing an extensometer in the rock mass means information is not available at the critical time, just after the pillar has been removed. Installing an extensometer previous to filling or even during the fill pour would likely give extraneous results due to fill settling and is not possible as access to the stope by personnel is denied unless extensive rock conditioning has been carried out.

An alternate would be to place an extensometer through the fill after it had consolidated. In fact, a hole was drilled through the fill from the footwall immediately above the 7 level to install an extensometer to measure closure as the 8C to 7 level sill was removed. Installation difficulty was encountered due to water draining from the fill. By the time the water flow had decreased, the sill was thought to have failed and further attempts were abandoned.

6.4.5 Predicted Stress Accumulation

The resolution of the stress parameters, as stated in section 6.1, to stresses normal to the 65^0 orebody would give model background stresses from just below 20 MPa at the top to just below 30 MPa at the bottom. This corresponds to the in-situ results recently obtained on the 10B drift at the mine. The increases in stress reported in the model are a resolution of the stresses from the excavated elements into the elements surrounding the opening. Since in MINTAB, the first element surrounding an opening reports an abnormally high result, the key result is an overall stress level in the core of a pillar. Since this model did not allow post failure analysis, the stress would increase beyond the rock strength if so calculated. It should again be emphasized that this initial modelling was not validated to mine results so can only be considered as relative or as being indicative of where high stresses are likely to develop.

In general, the stresses on pillar corners increase by 10 to 15 MPa up to step 4. As mining progresses between 8B and 7 level, the corners of all pillars are highly stressed, peaking at 98 MPa for the 12 metre model. In the 6 metre model, stresses were 10% to 15% higher than the 12 metre model, peaking at 113 MPa. In the sill mat model, the peak stresses were lower as load was shed to the high quality sill mat. In fact, the load reported on the sill mat reached 10 MPa, well above what it could actually support.

From this modelling work, it was concluded that the main area of concern was extraction of secondary stopes between 8B and 7 levels. It was considered likely that the sides and corners of these pillars would fail and that they should be contained rapidly in backfill to provide confining pressure to protect their cores. The overall increase in stress on these pillars was a maximum of about 70 MPa, above the 50 MPa increase measured by stress meters prior to failure as discussed later. It should also be noted that these models considered double lift or 50 metre stopes, not the 25 metre stopes which were extracted, and did not take the damage caused by 8D into account. In reality, this pillar will be shown as having failed earlier than predicted. Figure

6.12, a contour plot of the 12 metre model at step 6, demonstrates the locations of high stress concentration as the first sill stope is removed.

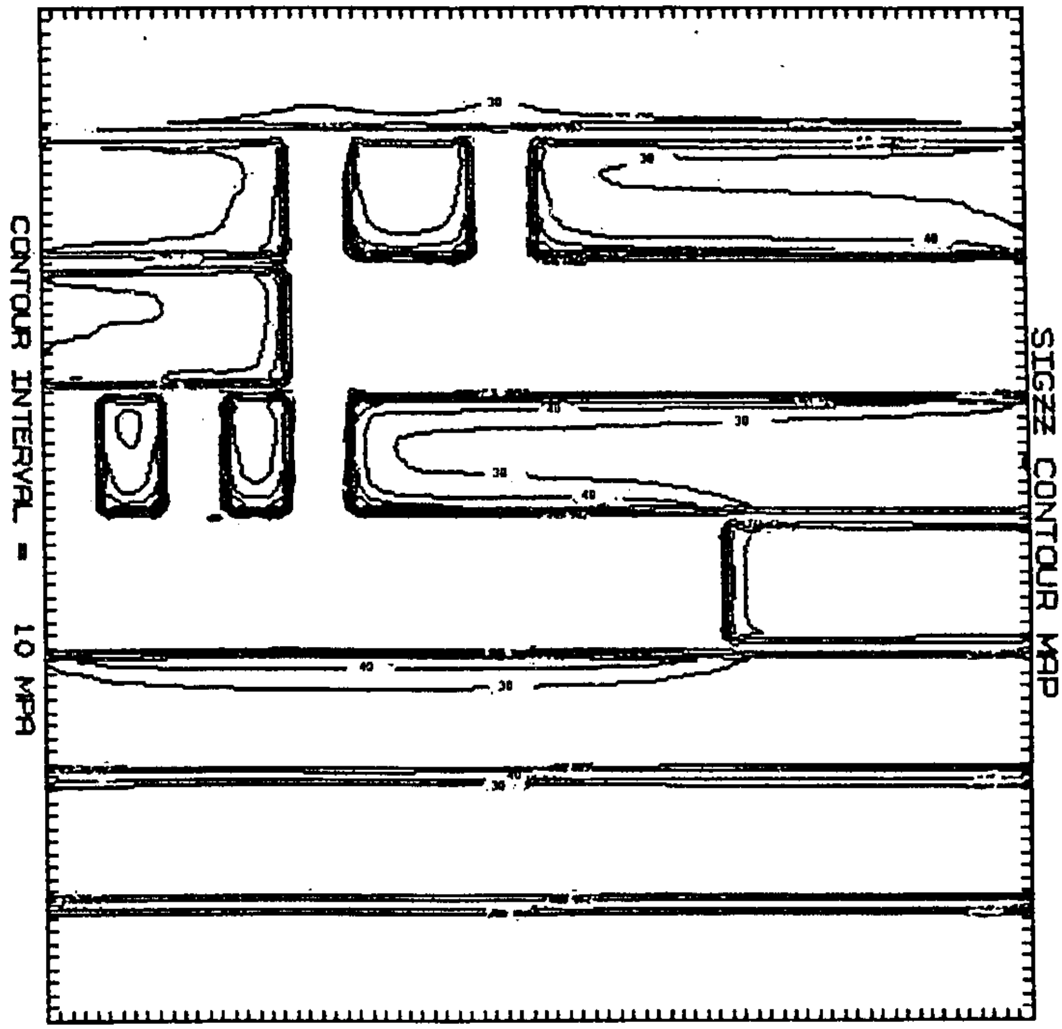

Figure 6.12 Modelled Stress at Step 6, MINTAB

6.5 COMPARISON OF THE MODELLED AND THE ACTUAL SEQUENCE

To better understand the ongoing modelling and instrumentation program since evaluating the initial plan, it is important to document where and why changes in the original mine plan occurred. Figure 6.13 gives an outline of the actual extraction sequence up to October 1990 at the time of removal of the 50th stope. This corresponds approximately to the end of step 5 in the initial model as shown in Figure 6.9 at which time 48 stopes had been excavated. The actual timing of the stope removal sequence for the first 100 stopes is given in Appendix A to better allow interpretation of modelling data and of stress cell graphs.

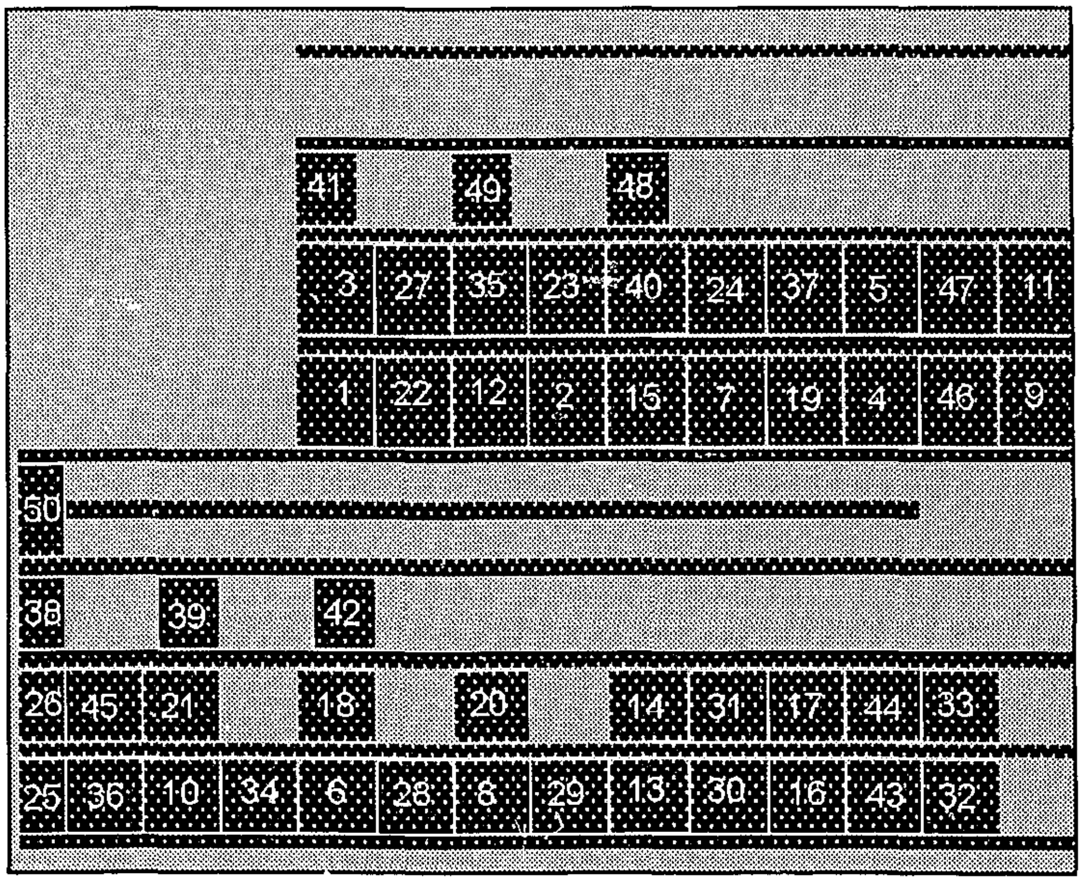

Figure 6.13 The Actual Extraction Sequence to Stope 50

The actual situation is, in fact, very close to the 5 year plan with the following notable exceptions:

A. The "sawtooth" pattern above 7B level and along 8B level is a reflection of the decision not to have primary stopes lead secondaries by more than one lift.

B. It was decided to shift to 20 metre primaries and 30 metre secondaries at stope 39 where mine geometry permitted.

C. An additional 4 stopes have been extracted to the eastern end of 8 and 8A levels thus extending the length of the sill pillar.

D. While a basic retreat pattern from the Golden Giant boundary is in evidence, it also can be seen that the use of single lift stopes will see an eventual sill pillar of only 25 metres in thickness.

6.6 VALIDATION OF THE MODELS

The initial numerical modelling was stated as giving only comparative results as the models were based on parameters available from adjoining operations and had never been validated by comparison with actual mine data. It has always been the opinion of the author that numerical modelling is best used by mine site personnel who can correlate observations from underground with modelling predictions.

6.6.1 Mine Site Validation

In January of 1990 mine management agreed and Examine2D, a two dimensional boundary element program, and ExamineTAB, the microcomputer version of MINTAB (both available from the University of Toronto civil engineering group), were installed on the mine's computers. This allowed rock mechanics personnel who dealt regularly with the underground conditions to validate the numerical modelling process. At that time, the author spent a week at the mine, training personnel in the use of the models and setting out a modelling program.

ExamineTab modelling carried out at the mine has been reported in an internal company report (70). A sensitivity analysis was carried out by varying input parameters until results agreed closely with the observed underground rock conditions. Following this analysis, the following parameters were chosen for ongoing modelling studies:

Young's Modulus	24 GPa
Poisson's Ratio	0.17
Unit Weight of Rock	0.0288 MN/m^3
Dip of Orebody	62^0
Stoping Thickness	7 metres
Horizontal to Vertical Stress	1.4

The first two parameters are based on a weighted average of the rock values assuming the orebody contains 85% non-barytic ore and and 15% of the baritic zones. The density is based on tests carried out on hangingwall rocks and the stress ratio based on one series of in-situ tests done at Golden Giant at a level equivalent to the 7 level.

The first model run at the mine was a 5 step model based on stress cell observations from stopes mined up to the beginning of 1990 as follows:

Step 1. Stopes 1, 2, and 3 which were excavated prior to installation of the first stress cells.

Step 2. Stopes 4 to 26: the stress cells on the 8D sublevel had shown a constant trend during this period and deteriorating ground had been observed. All the 7 level stopes immediately above the 8D are mined during this period with the primaries mined above 7A and 8A. Adjoining secondaries, L73B and L75B, have been left on 8A immediately above L84D and L86D; one of these is then removed in the next step.

Step 3. Stope 27: the rock containing the L86D stress cell was thought to have failed. Increased stress on removing L73B is shown in Figure 6.14 but cell access was lost before the relaxation was documented.

Step 4. Stopes 28 through 34: the stress cell in L84D, Figure 6.15, shows a constant trend; ongoing ground deterioration is observed on 8D sublevel. All mining is, in fact, restricted to the 8 and 8A secondaries.

Step 5. Stopes 35 and 36: At this time when the rock containing the L84D stress cell is thought to have failed. Stope 35, L75B, is the second of the secondaries on 7A just above the 8D stress cells. Figure 6.15 shows the jump in the stress cell in L84D, 8D undercut, at the time of the L75B blast and the subsequent relaxation of the rock. Rapid deterioration of the undercut prevented further access.

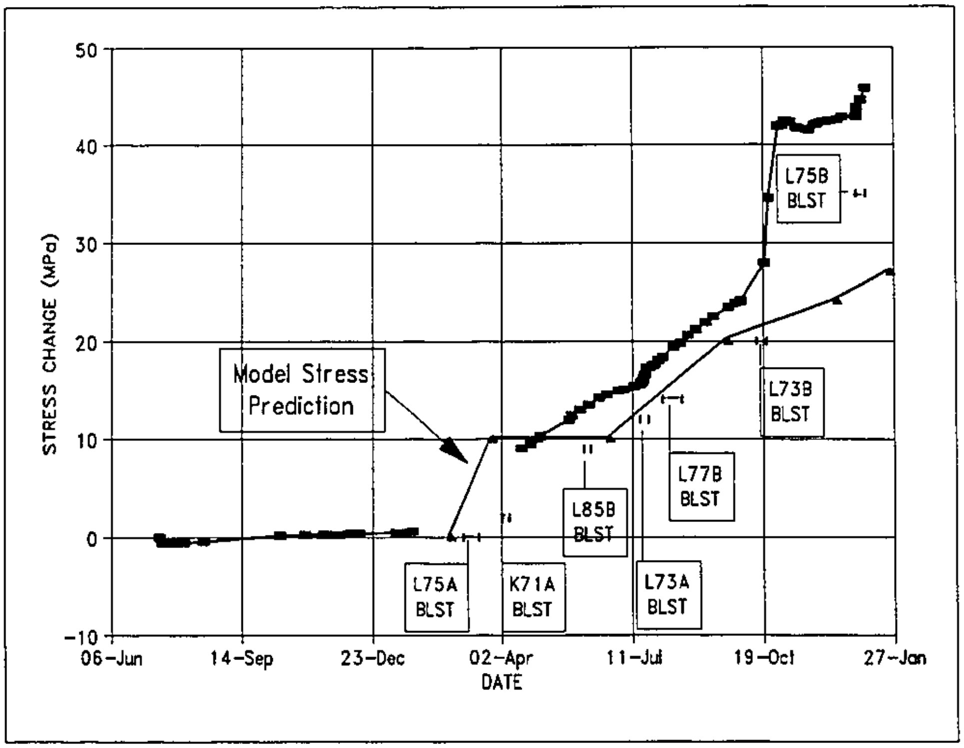

Figure 6.14 Stress Cell: L86D, 8D Undercut

In their analysis of the results, mine personnel pointed out several areas of agreement between the predicted results of the numerical modelling and the actual recorded stresses. A 17.5 MPa stress increase was predicted during step 2 of the model for the L86D cell versus the 24 MPa actually recorded, and a 17.5 MPa increase predicted for steps 2-4 of the model compared to the measured 20 MPa. After this validation of the modelling process, the mine carried out an analysis of the 1990 mine plan with predictions of where problems might occur. Because of changes in mine personnel, the predictions for this period were not compared to actual occurrences.

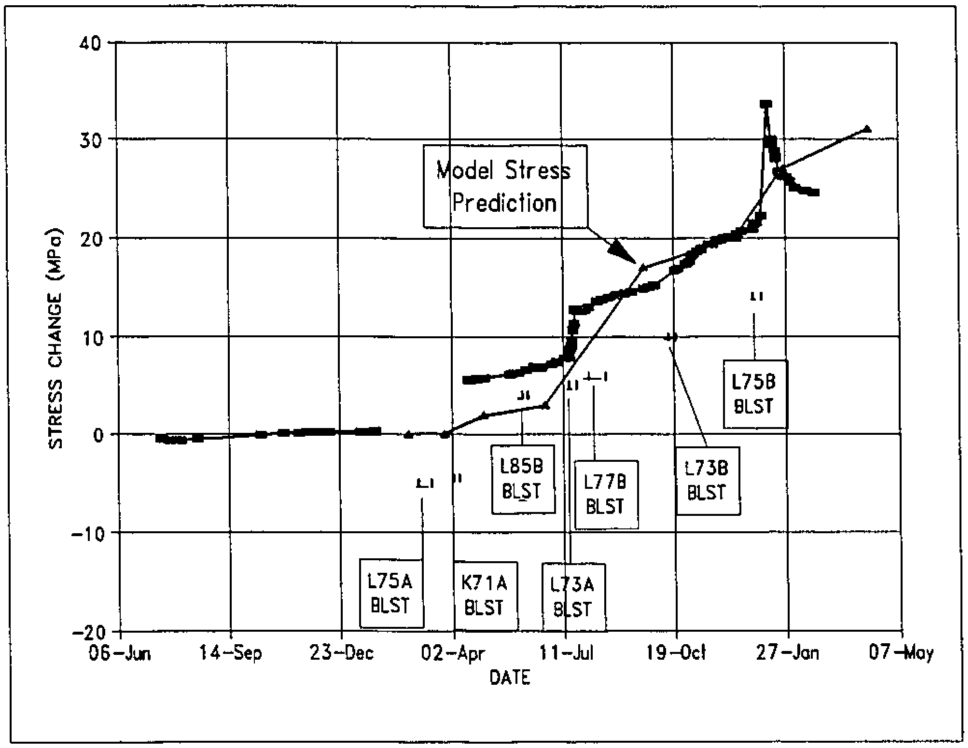

Figure 6.15 Stress Cell: L84D, 8D Undercut

6.6.2 Detailed Model Validation

All further modelling work carried out by the author used the input parameters developed in this initial validation work as showm on page 198. Subsequent detailed validation of the ExamineTAB models was carried out using two 8 step models. It should be pointed out that the modelling work in this project was initially carried out using MINTAB and this validation was carried out using ExamineTAB. The only difference in the models is that MINTAB allows the input of backfill but this does not sigificantly affect the predicted stresses. Also the Mintab work was always predictive, that is it followed a proposed mining sequence, whereas this validation follows the actual mining sequence.

The first model concentrated on validation of the 8D sill area, the area critical to this thesis. A stepped model of the first 40 stopes to be mined was created, corresponding to the stress cell data displayed in Figures 6.14 and 6.15. The steps in the model were chosen based on the stopes closest to these stress cells to best approximate the transfer of stress to them as follows:

Step 1: The first 11 stopes up to February 22, 1989. These include 5 primary stopes above 7 level and 3 above 7A level.

Step 2: The next three stopes to March 26, 1989. The L75A stope directly above the L86D cell is removed.

Step 3: The next 4 stopes to April 30, 1989. Removing Stope K71A now leaves 4 adjoining stopes open above 7 level

Step 4: The next 3 stopes to June 23, 1989. Stope K75A removal leaves 6 adjoining stopes open above 7 level.

Step 5: The next 5 stopes to September 20, 1989. Removing the L73A stope immediately above the L84D cell, and two primaries above 7A.

Step 6: The next 8 stopes to December 15, 1989. One secondary stope, L73B, is removed above the 7A level.

Step 7: The next 2 stopes to January 25, 1990. Four adjacent stopes are now open above 7A level and directly above the 2 cells.

Step 8: The next 4 stopes to April 10, 1990. Two more secondaries are removed above 7A. Only one stope now remains on both the 7 and 7A levels only one stope from the eastern end of the mining block.

The results of the stress increases predicted by the model after step 1 are plotted in Figures 6.14 and 6.15, using the triangular points, on the same time scale as the stress cell results. A stress level of 45 MPa was subtracted from the value predicted at the end of step 1 to correspond to the time of installation of the cells.

From Figure 6.14, it can be seen that the correlation of the model with the L86D stress cell data is very good from the start of stress increases in late March until the rapid rise begins in mid October. This difference at the end of the cell's life can be explained by stress transfer to the cell by the gradual failure of the intact rock above and to the east of the cell. The model considers all of this rock to be intact and so underestimates the stress level.

Figure 6.15 shows even better correlation. The final peak leading to the failure again does nor occur for the same reason discussed above. The fact that the cell fails at a somewhat lower stress level may be indicative of the higher barite content present to the west of the orebody.

The results presented here provide strong evidence of the validity of the models used up until the rock failed. Results from the second model validation will be presented in section 7.3.1.1 in discussions on stope back failures.

6.7 ONGOING NUMERICAL MODELLING

In the fall of 1990, after on-site discussions with engineering staff, additional modelling was carried out to aid in the planning of the extraction of the remaining stopes up to the first two stopes beneath the 7 level. It was realized that using elastic models to predict stress in this area after the stopes between 8C and 7 levels had likely destressed would not give quantitative results. It was still felt, however that the results were valid for comparative purposes when deciding on alternate extraction sequences. It was hoped to minimize the effects of the additional mining on the area and to look at the predicted stress level for the Golden Giant shaft pillar.

6.7.1 The TC4 Model

This model was run at the time when the 50th stope had been extracted. Four steps were used to reach the existing extraction level in order to get results at critical points in the actual sequence. The remaining stopes were removed with an additional 6 steps. The first four steps of the model are illustrated in Figure 6.16, steps 5-8 in Figure 6.17 and the final two steps in Figure 6.18, and analyzed as follows:

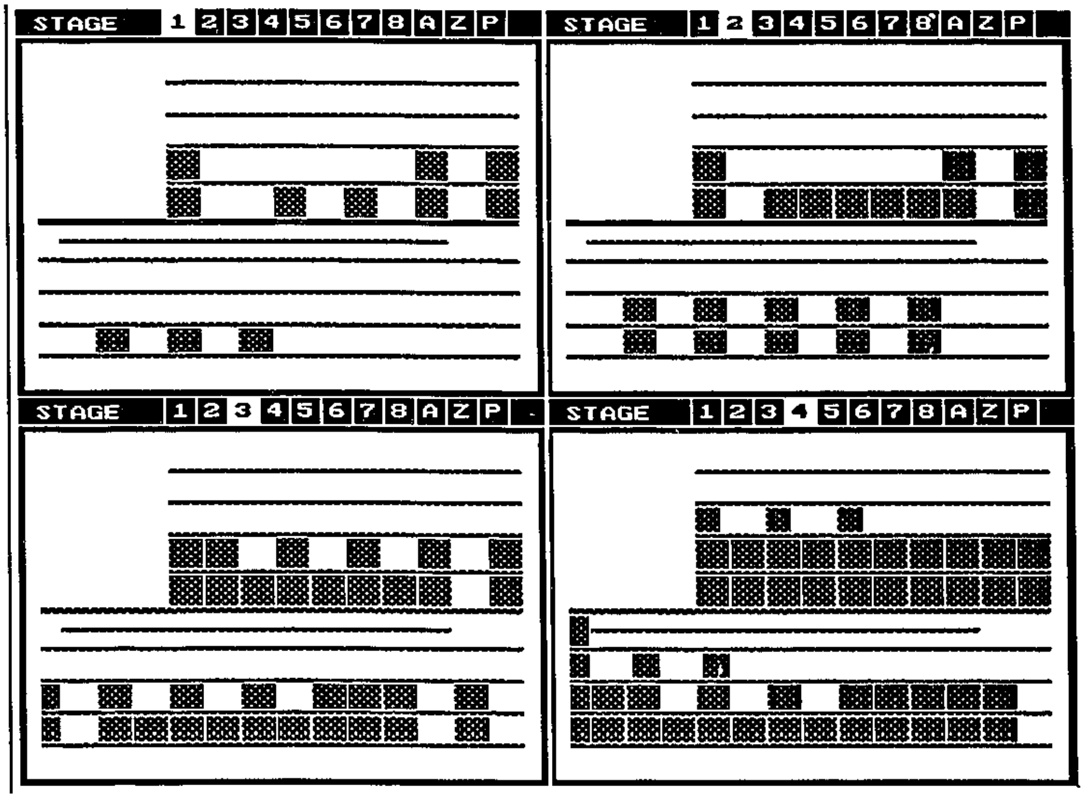

Figure 6.16 Model TC4, Steps 1-4

Step 1. Excavate all levels and 11 primary stopes. The maximum stress level is 41 MPa at the corner elements of M87, L83 and J71. Secondary stope cores are at 25-30 MPa, general abutments stresses at 20-25 MPa, with the 8D-7 level sill at 30-35 MPa.

Step 2. Excavate up to stope 21. Six adjoining stopes are open above 7 level (L75A to K77A). The maximum closure is 47 millimetres. Up to 50 MPa is predicted in corner elements of secondary stopes above and below 8A, up to 45 MPa in 8D-7 level sill with 5 millimetres of closure on intact rock. In general, secondary pillar cores are still at 25-30 MPa.

Step 3. Excavate up to stope 34. There is a double lift in K81 A & B, K83 A & B, and K85 A & B for 75 strike metres, a double lift on L71 A & B and L73 A & B. In secondary cores above 7 level the stress is 30-40

MPa and is similar in secondary stopes below 8B but with a maximum stress of 71 MPa on the corners above 8A. Stress levels in the 8B-8C sill are 30-35 MPa with 8D-7 level in the mid 40's. No real problems are indicated in K72C but failure was known to have begun. The maximum closure is 70 millimetres on K83.

Step 4. Excavate up to stope 50 (October 1990). The 7 and 7A levels are complete. Three secondary stopes remain above 8A. Secondaries above 7B have 60+ MPa on the corners but general pillar stress is around 35-40 MPa. The maximum stress occurs in the secondaries above 8A isolated by the early mining of K83B and K87B with a maximum of 74 MPa at the corners. The abutment stress has increased to 50 MPa from 42 MPa in step 3. The 8D-7 level sill is shown at 50-60 MPa but has in fact failed. Maximum closure is 81 millimetres on 7A.

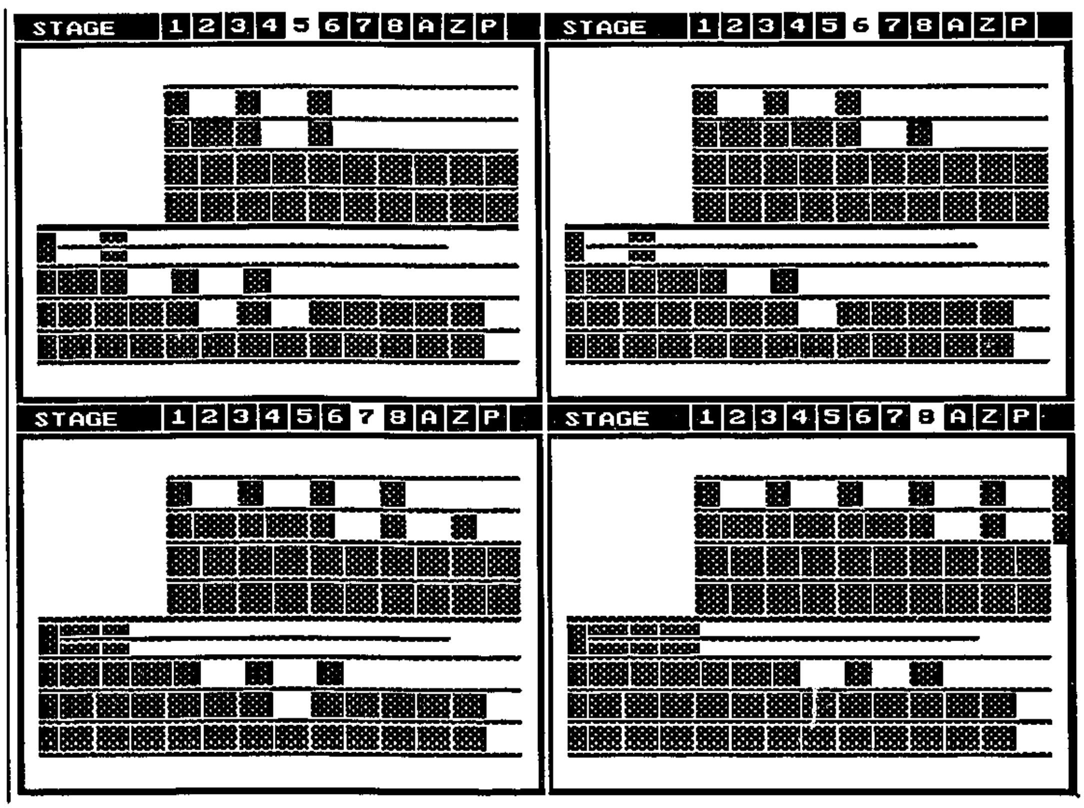

Figure 6.17 Model TC4, Steps 5-8

Step 5. Remove three stopes above 7C and the first secondary above 7B. Remove M87B, M84C and M86D. The cores of L87B, L83B and M88C are now in the 45-50 MPa range, approaching the levels at which the 8D-7 level sill failed. Closure changes by only 3 millimetres.

Step 6. Excavate stopes 59-62, the next secondaries above 8A and 8B. The stress jumps to 90 MPa in the adjacent secondary above 8B (L84C) but no major change occurs elsewhere. Closure jumps to 103 millimetres as 3 levels are now open along a 5 stope distance above 8B and 7B.

Step 7. Excavate stopes 63-66, remove the first secondary above 8C. The stress jumps to 92 MPa in one element of M88D but there are no major changes elsewhere and closure increases only 3 millimetres.

Step 8. Excavate stopes 67-76 (75 and 76 are not in the model). The stress jumps to 106 MPa on the corner of L88C as the secondary beside and below are removed. Stress increases on L82D as mining gets away from the shaft pillar. The corner element on the shaft pillar increases but the overall stress level in the pillar is low. There is a major jump at this step and outlines a possible decision point: whether the material between 8C and 7 should be allowed to catch up with the material between 8B and 8C before L84C is removed. It may be that an inclined front will work best when there is an abutment above to which the load may be shed. In this situation, it leaves the 8C-7 level sill in isolation.

Step 9. Excavate stopes 77-79. The stress jumps to 117 MPa on L84D and the shaft pillar corner to 97 MPa from 79 MPa. This is only one element, now contained in backfill. No other major changes occur.

Step 10. Excavate stopes 80-83 (the end of the 1991 plan). The only problem is the corner of 8C at L86D. Closure below the 7 level fill increases to 120 millimetres (1% to 1.5%) so failure might be expected.

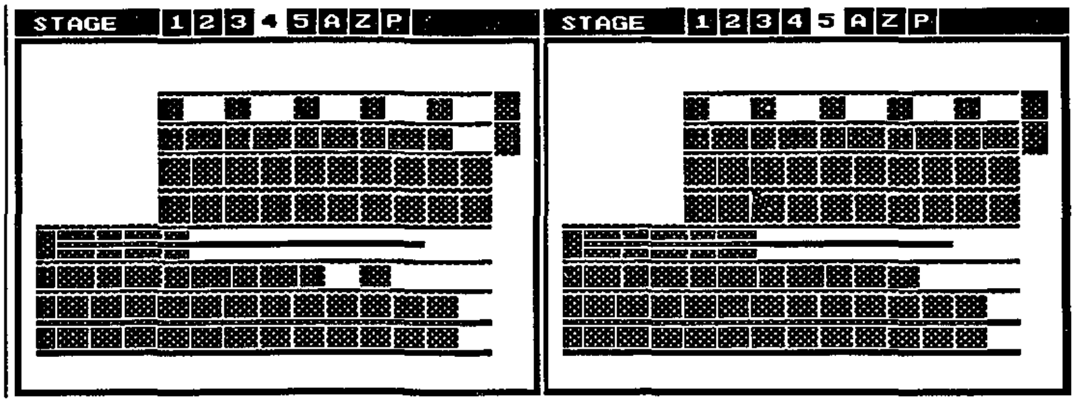

Figure 6.18 Model TC4, Steps 9 and 10

In summary, as mining progresses above 7 level, no major changes occur, as long as the sequence of primaries and secondaries is maintained, the load sheds to the abutment. It is obvious that the early removal of K83 and K87 stopes above 8A has caused, in the model, the remaining secondaries to display increased stress levels as they are isolated from the main abutment. An alternative to the sequence from step 7 to step 8 would be retreating 8B and 8C as a flat front as opposed to an inclined front.

The final MINTAB modelling was carried out in early June 1991. Four models were run starting from step 4 of the above model, that is at completion of stope 50 in October 1990. The models which were run in this series are the TC5, TC6, TC7 and TC8 and are described in the following sections.

6.7.2 The TC5 Model

This model included an additional seven steps to follow the latest plan submitted by the mine. The stopes mined to that time were extracted as step 5, that is, up to stope 63. The stopes to the east of J74 were eliminated as they would have no significant impact on the 7 level sill. Steps 5 to 8 are illustrated in Figure 6.19, steps 9 to 12 are illustrated in Figure 6.20 which is zoomed to better show the 8D sill, and the results are analyzed as follows:

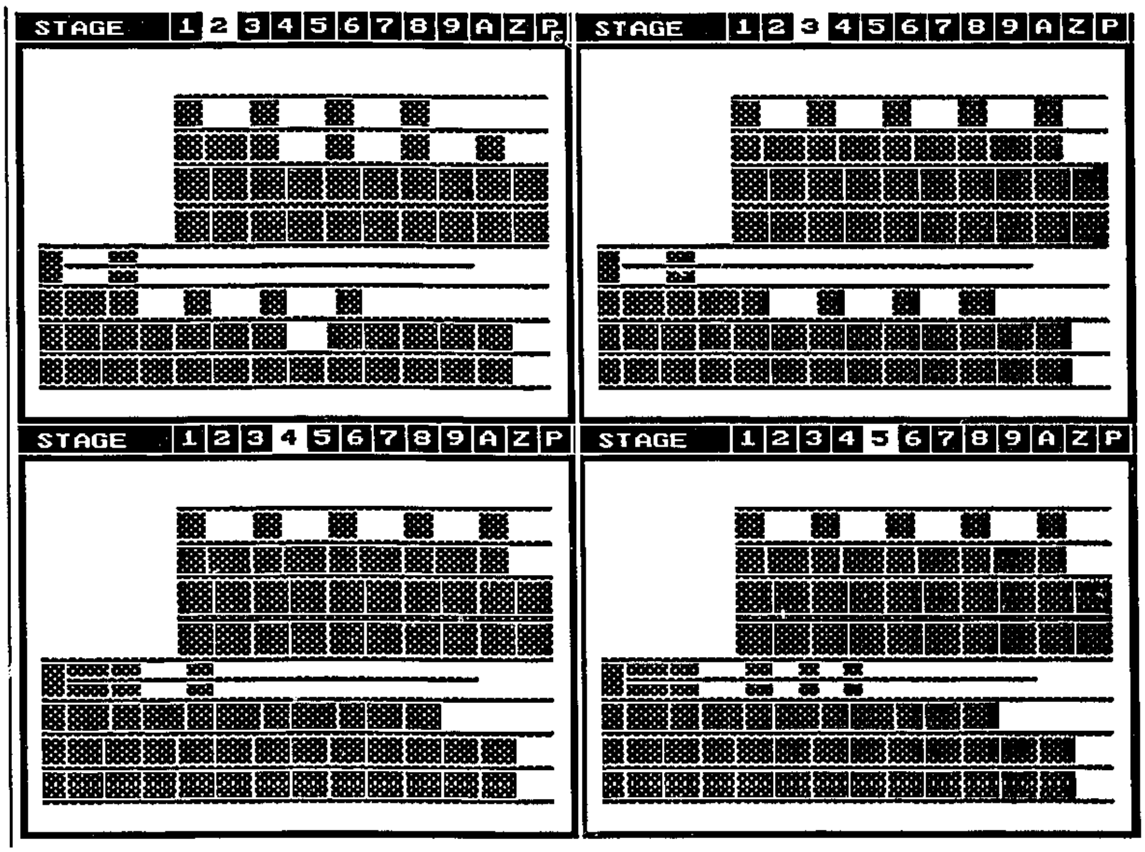

Figure 6.19 Model TC5, Steps 5-8

Step 5. The maximum stress level at step 4 was 74 MPa. This jumps to 87 MPa at the corner of M88C and to around 80 MPa at the corners of L87B. These two stopes were being extracted at the time of modelling and would indicate the ground behaviour at this model stress level.

Step 6. On the corners of the three 8C stopes, the stress level is in the mid 80's, with one extreme value of 95 MPa on the lower western corner of L84C, again providing a good validation reference.

Step 7. The maximum stress on the lower corners of M88D jumps to just over 100 MPa, with the core at 60-80 MPa.

Step 8. The maximum stress in M88D increases only slightly but the stresses in stopes 13 and 15 are high, up to 110 MPa at the corners but with the entire stopes highly stressed.

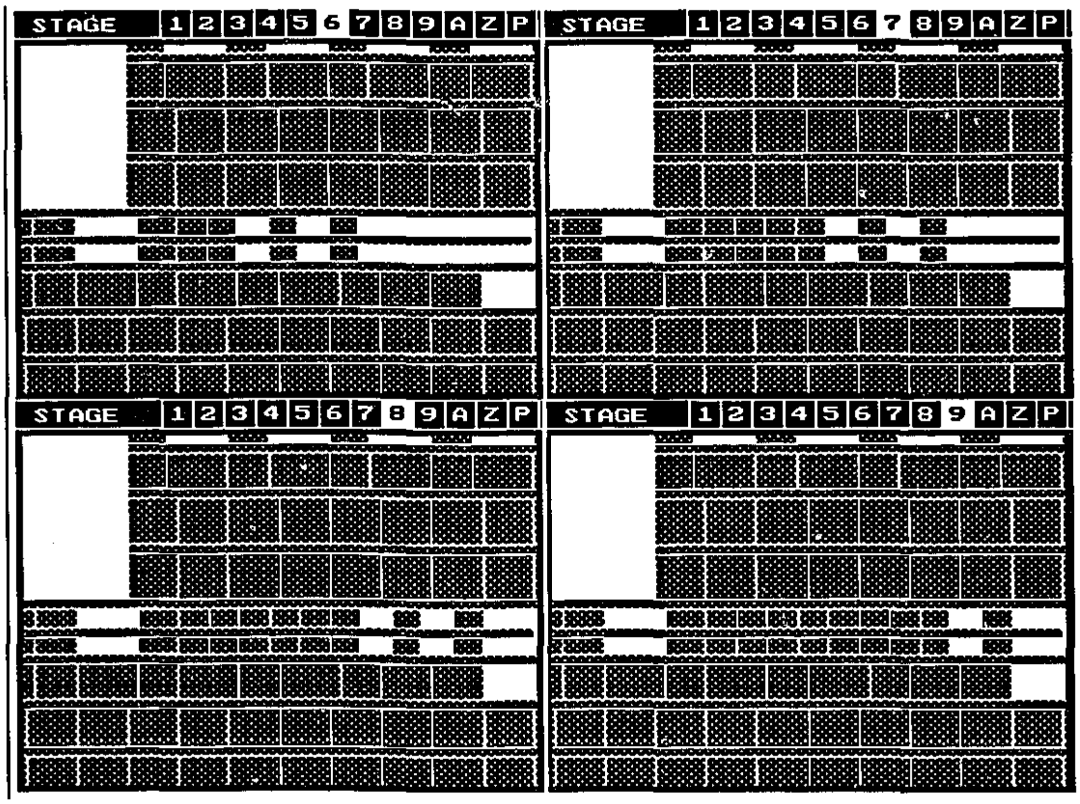

Figure 6.20 Model TC5, Steps 9-12

Step 9. High stresses are predicted up to 130 MPa in the second lagging stope. The corner of M88D is at 118 MPa.

Steps 10, 11 & 12. Stresses remain high at about 140 MPa in the isolated last stope and remain constant at about 125 in the lower corner of M88D.

6.7.3 The TC6 Model

This model, as illustrated in Figure 6.21, followed the existing plan up to step 7, the extraction of L82D. It then followed a retreat sequence for the remainder of the 8D to 7 level sill which was retreated in 20 metre wide slices as it was felt that the width of stopes on a retreat sequence would not be critical and production requirements would favour the size being maximized.

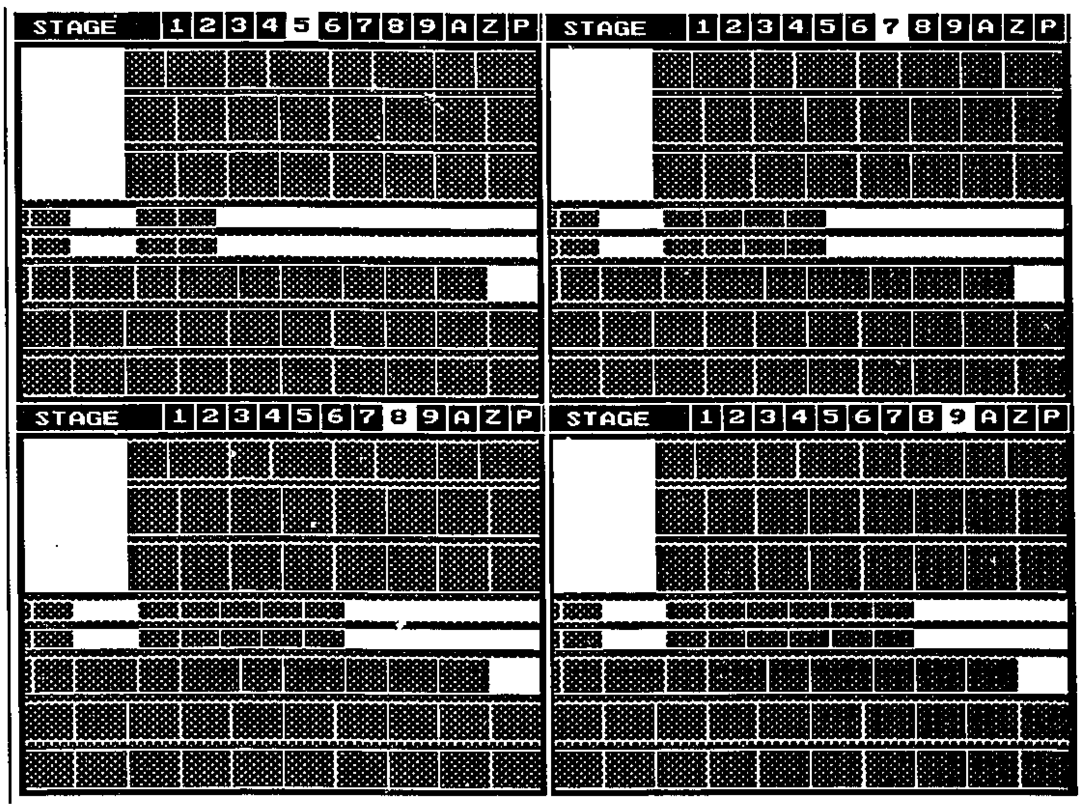

Figure 6.21 Model TC6, Steps 5,7,8, and 9

From analysis of the model data it could be seen that the maximum stress levels in the sill pillar are just over 130 MPa, approximately 10 MPa lower than those created under the existing plan. Also, this appears in only the two corner elements of the lagging sill which would be blast damaged and unloaded. The remaining core, however, is more stable than in the isolated lagging stopes.

6.7.4 The TC7 Model

This model was identical to TC5 except that M88D was extracted on step 9. This was just to see the effect of removing this stope on the level of stress in the sill.

The removal of M88D only slightly increases the load on the retreating sill beyond stope 15. This was only of theoretical interest since this block was being used for access to the hangingwall and had to be mined last.

6.7.5 The TC8 Model

This model, depicted in Figure 6.22, followed a different sequence after step 5 in that it delayed the removal of L84C, L88C, and K84C to retreat with the sill pillar. It is realized that delaying these stopes would cause production problems but might be considered if potential results justified the delay.

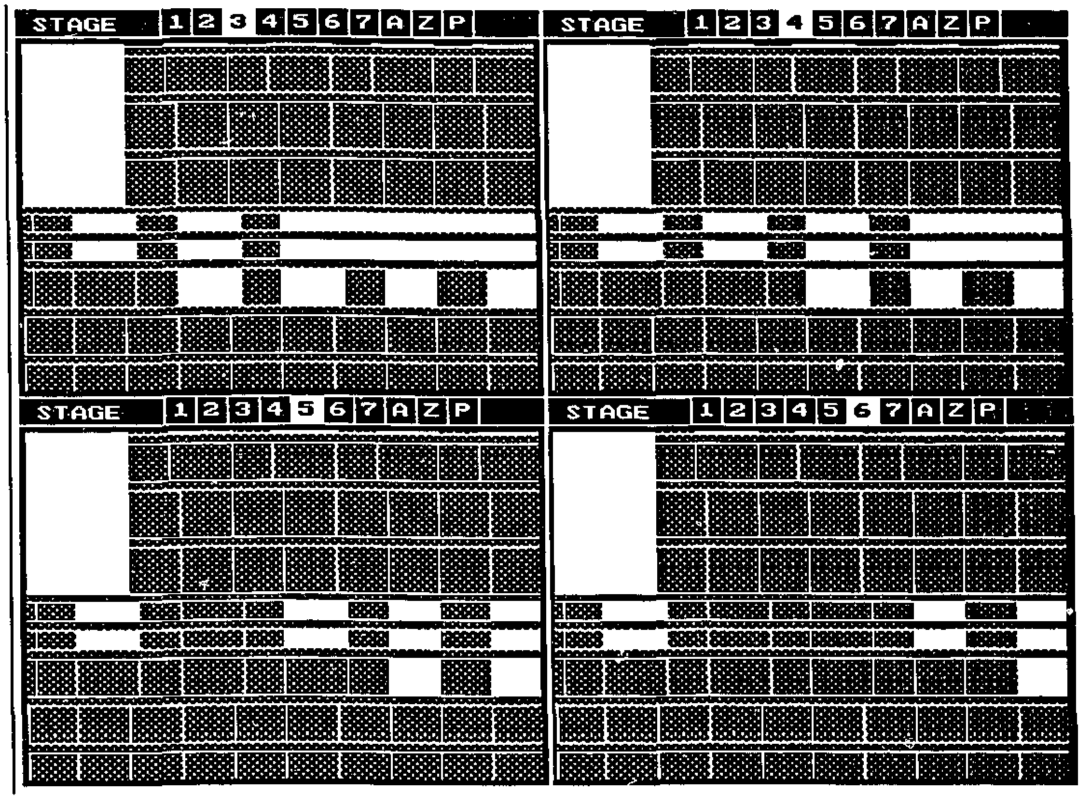

Figure 6.22 Model TC8, The last 4 Steps

This sequence did not significantly reduce the stress levels in the lagging sill stopes. In addition, the scheduling problems and possibly extra ore placed at risk do not justify any change in schedule up to the extraction of L82D.

6.7.6 Summary

The key points regarding the extraction of the 8D sill can be summarized as follows:

1. The plan to extract 15 metre stopes along the 8D sill would cause very high stress levels in the lagging stope.

2. The closure in the area would be greater than on a retreat sequence

and there would be an increased chance of dilution from a backfill failure. It was felt that the backfill would fail to a stable arch as demonstrated in the centrifuge and FLAC models run as part of a previous Mining Research Directorate research program (51).

3. A trial of the retreat sequence would not prevent reverting to the alternate extraction sequence if necessary. The retreat sequence would likely slow down the rate of extraction but if longer strike length is possible there would be savings in slot raises and fill fences.

As will be described in Chapter 7, points 2 and 3 formed the basis of the actual extraction sequence used.

6.8 PRACTICAL LIMITATIONS OF NUMERICAL MODELS

Throughout this research, extensive use was made of numerical models. At the early stages of the research, while a search for a case study was underway, MINTAB models were developed of mining practice at the Campbell Red Lake Mine, alternative mining practice which were being considered for the Macassa Mine and theoretical sequences which might prove useful in steeply-dipping tabular deposits. While the results from those models are not reported in this thesis, they helped the author gain an understanding of both the modelling process and the importance of developing a planned sequence early in the life of a mining operation.

This section is intended to summarize points the author feels are important in using numerical models during both the sequencing planning and the operational phases of the mine. No attempt is made to describe the mathematical basis of any of the models discussed.

6.8.1 The Displacement Discontinuity Models

While section 6.4.1 justifies the use of MINTAB for this research work, the following cautions must be applied to the use of the program:

1. The model overestimates the loading on elements immediately adjacent to mine openings, so general pillar loadings should be taken from data in the core of the pillar. This problem is increased as the modulus of the rock increases. The author, in running a model of the Macassa Mine, once made an error of three orders of magnitude in inputing the normal 80,000 MPa modulus through an input data file. The results were not immediately obvious as being different from the previous model.

2. The boundary of the model is considered stiff so care must be taken in assessing data in boundary areas. If openings are placed too close to the boundary the stress and closure calculations will be affected by the boundary. At least one opening distance should be allowed.

3. The model is controlled by stating the number of iterations allowed during a model run or by stating the percentage of unbalanced forces allowed at the conclusion of the run. It must be remembered that this process in no way relates to real time. Adding backfill at the end of a step does not provide the closure control which can be achieved by rapidly filling a stope underground.

4. Problems can develop during the iterative process in the backfill model. During a run of a David Bell model using backfill in the sawtooth extraction sequence, the model ran to completion during 19 iterations but developed positive stresses on pillars surrounding one stope that were an order of magnitude too high. These were balanced by high closure and negative stress on the stope in question. Other areas of the model with the same geometry behaved normally.

The ExamineTAB version of the model provides an ease of developing the model through the AUTOCAD based system and provides good interpretation tools through the HOOPS based graphics. By zooming into the model and by altering the colors and the ranges of the filled contours it is easy to evaluate the stress level at any point. There is no restart capability so any model over 9 steps has to be re-run, and there is no use of backfill. The modeller should also be aware that any element in the model which is not numbered in the extraction sequence defaults to step 1. In general, however, it offers a very user friendly modelling system for extraction sequences

These examples point out the need for users of numerical models to understand both the modelling process and the actual ground behaviour before the absolute values of model output can be used. DZTAB was used with considerable success at the Campbell Red Lake Mine because the mine behaves elastically, as discussed in Chapter 3, and because the users interpreted the results in connection with observed rock behaviour underground.

6.8.2 The FLAC Model

The author has successfully used the Fast Lagrangian Analysis of Continua (FLAC) code from Itasca to estimate the stability of openings in consolidated backfill (51) and to assess the stability of backfill using alternate binding agents (71). The program was used successfully to examine the stability of baritic zones in ore pillars (72). The author developed a mine wide model of the David Bell Mine to help in assessing the stope back failures. That model, as shown in Figure 6.23, was very complex but, apart from providing

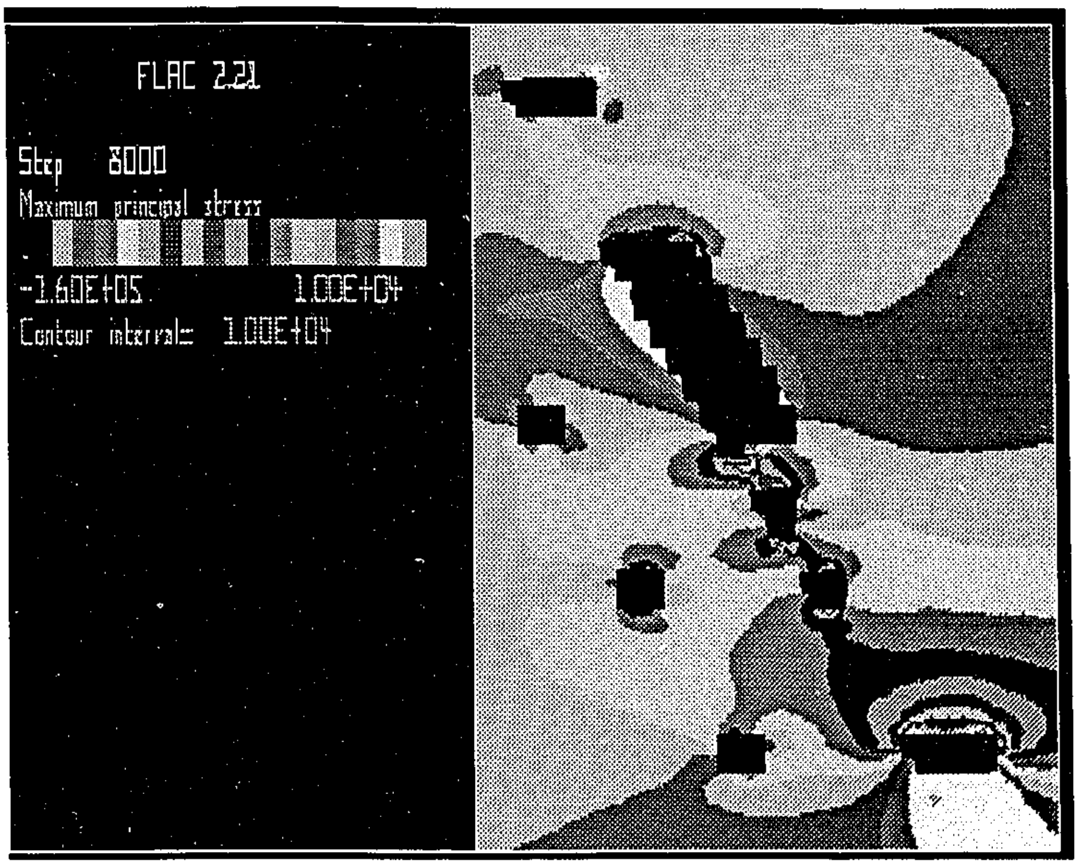

Figure 6.23 A Cross-sectional FLAC Model of the David Bell Mine

some colourful and thought provoking stress contours, did not provide definitive help in assessing the problem. The problem is that these situations are too complex to be modelled or evaluated using a two dimensional model.

Certain features of the FLAC program, however, demonstrate some very valuable attributes that any modelling code should possess.

1. The model can represent several material types which can have different constitutive laws. It allows large strain, non-linear stress-strain, plastic and elastoplastic material behaviour.

2. The program has a stepped problem solving process which allows events to be sequenced as they occur during the mining process. Again it must be remembered that this stepping process bears no resemblance to a real time process but if on-site observations were used to validate the model the real world could be approximated.

3. The program provides excellent graphical output capabilities. The capability of keeping up to 100 histories of individual parameters at any point in the model and to present these graphically aids greatly in the interpretation of the results.

The main disadvantages of the model are that it is only two dimensional and that input of the excavation geometry and the model parameters is tedious, time consuming, and requires an experienced user although later versions have improved the user interface. The model requires considerable run time, so if it were made three dimensional this could become a limiting factor.

6.8.3 Boundary Element Models

Except for MINTAB, the author has not extensively used boundary element models. Grant and Potvin (58) and Kazakidis (63) have described BEAP modelling at the Golden Giant Mine, and Pakalnis et al. (73) describe a BEAP model of the Detour Lake Mine.

A four step model was developed, using the EXAMINE3D program from the University of Toronto, of the David Bell Mine based on the TC5 model as described in section 6.7.2. More precisely, four models were developed as the program does not provide any stepping capability. This program became available only recently, therefore, these models were run merely to evaluate the program and no interpretation of the results was carried out.

There are several comments arising from the evaluation of the program:

1. Excavation geometry input is made easy by an AutoCad interface allowing the use of mine drawings. The input of model parameters is done interactively.

2. The model has no stepping or restart capability so the analysis of what happens at any step in the sequencing process requires the comparative analysis of two complete sets of output data.

3. The output data are very graphic and are provided through contour plots and isometric surfaces. Output at certain points or along certain planes are available but these must be predefined. This aids

interpretation but makes comparison to empirical data difficult. One must remember that contour plots are mathematical approximations.

4. The model assumes homogeneous, isotropic, linearly elastic material and allows only one material type. Even with that restriction, run times are long and output files are significantly large.

5. To give good results, the element size has to be small enough to fit into the excavation geometry. While element size can be varied throughout the model, more elements rapidly increases run times.

The EXAMINE3D program was developed to assist researchers investigating the mechanisms causing microseismic activity. The output is designed to provide data along certain planes for comparison with the measured microseismic activity. The model is not considered user friendly when developing mine extraction sequences.

6.8.4 Finite Element Models

While the author has not used finite element models, research at McGill has advanced their applications to Mining. Publications by Mitri et al. (69), (74), (75) describe both two and three dimensional applications. A paper by Quesnel and Chau (76) describes a three dimensional application at the Doyon Mine.

The author can appreciate the advantages of finite element modelling, specifically the ability to include multi material types in the model. The compute time for sequence based modelling remains a constraint but one which may be overcome as computation power increases, by use of Digital's new 64 bit Alpha chip for example.

6.8.5 Summary

Numerical modelling can be a useful tool in developing mine extraction sequences. While no ideal model exists, both MINTAB and FLAC have been used with success to evaluate certain aspects of ground behaviour during the extraction process.

The successful use of numerical models demands that the user understand both the modelling limitations and the expected ground behaviour. Models should be validated through on-site observations. Output should be considered relative rather than absolute values. Using models to perform a sensitivity analysis of the effects of the various geotechnical parameters is seen as a very useful exercise. As an example, if a boxhole shrinkage layout were modelled using varying input stresses, the results would show that eventually the stress would exceed the carrying capacity of the boxhole pillars. If the model were then validated as depth increased, one could then predict at what depth this was likely to occur. The mining layout could be modified in advance instead of waiting for the first rockburst to signal that a change was necessary.

Finally, in order for numerical models to be used at the mine site they have to be user friendly for the busy operating personnel. This means that input of opening geometry should coincide with the electronic mine plans. Output should be in a format that can easily be compared to empirical data from the mine. In the mine planning process, the planner must visually interact with a complex set of mine parameters. The use of three dimensional computer graphics to provide an interface between such tools as numerical models and the planning process is essential to the future of their development.

6.9 CONCLUSION

This chapter has described the level of information required both to carry out a geotechnical assessment of a mining property and to undertake numerical studies of the extraction sequence. Information gathered at the David Bell Mine by the author, by summer research assistants, and by the rock mechanics consultant has been merged with information from the two other mining operations in the Hemlo mining area. This information has served as the basis of numerical modelling studies of the David Bell Mine.

Numerical modelling studies of the David Bell Mine are described starting from the initial design stages of the main zone to the final extraction of a major sill pillar separating two initial mining areas. The process of validating these models has been discussed along with the difficulty of collecting certain information vital to the validation process, closure data for example. These numerical models have been used by mine personnel as one tool in planning the extraction sequence. The actual extraction sequence is compared to the planned sequence to show that while operational requirements and some stope size adjustments caused variations along the way, no significant variation occurred to the initial plan.

Finally, a comparison of the numerical models used by the author has been provided to better understand the usefulness of these tools in planning the extraction sequence. In the following chapter, the use of empirical data collected from an ongoing mine instrumentation program to augment these planning tools will be described.

CHAPTER 7

THE PRACTICAL ASPECTS OF MINE SEQUENCING

The initial five year, main ore zone, mining sequence at the David Bell Mine was developed in conjunction with the mine's rock mechanics consultant, to meet management objectives. Mining began on the 8 and 7 levels in the thickest and highest grade material, allowing production to commence before exploration and development were completed at depth. This established an upwards progressing sequence from the 7 level to the upper orebody limit. Ore along the common boundary with the adjacent Golden Giant Mine was mined first, thus eliminating a potential future problem area. However, the fact that two levels began production immediately necessitated a sill pillar below the 7 level. It also meant that a second major sill pillar would be required below the 8 level as mining progressed upwards from the 9 level.

As shown in section 6.5, the actual extraction sequence of the first 50 stopes did not radically differ from the initial 5 year plan which was numerically modelled. Individual stope extraction timings differed, as would be expected by the requirements of the short range scheduling activities. The major departures from the initial plan were the decision to extract 25 metre high stopes as opposed to 50 metre high stopes; and the failure to establish a retreating sequence away from the Golden Giant boundary.

This chapter documents and analyzes some of the departures from the plan which were necessitated by unforeseen conditions or the necessities of the short range planning priorities. It demonstrates the importance of the results from the extensive mine monitoring program which had been in place since mining of the main zone began, interpreted in conjunction with on-site observations of ground behaviour and numerical modelling results, to control and revise the mining sequence as mining progresses.

7.1 THE MINE MONITORING PROGRAM

As part of an instrumentation program begun in 1988, the mine has used Ground Movement Monitors (GMM's), generally in very unstable ground, to detect large ground movements, and Geocon vibrating wire type stress cells for stress monitoring. The GMM's are used for short periods, generally less than one month, to monitor ground movement in critical areas when personnel must be present. Generally, there are approximately 15 active stress cells and data exists on more than 40 inactive cells from mined out areas. The location of many of these stress cells was decided by the ground control supervisor in conjunction with the author. During 1990 and 1991 the mine assumed the installation cost of cells purchased by the author on a research contract.

The stress cells are installed in two different mine openings, in the back of the ore undercut or in the back of the haulage ways. The cells are placed in 5 metre diamond drill holes that are drilled up dip into the back. The stress cells are unidirectional so are aligned to detect changes in stress perpendicular to the bedding, and consequently perpendicular to the orebody. Some haulage cells in the footwall have been aligned to detect changes along strike.

Results from two of these cells have already been shown in section 6.6. The identification numbers of the stress cells correspond to the identification number of the nearest stope block, for example L86C-FW,8C would be a cell installed close to the L86C stope in the 8C footwall drift.

In general, the two types of stress cells behave in an expected manner. Firstly, the cells in the ore tend to show increases in stress as mining progresses towards them but reductions in stress are rarely recorded as the cells or access to them are lost as the stope immediately below is blasted. Secondly, the cells in the footwall haulage show surprisingly small stress increases as mining advances towards them but show stress reductions when the stope immediately above is blasted. These cells, because of the orientation, are immediately down structure from the overlying stope.

Rather than carry out an analysis of individual cells, it was decided to include pertinent cell records in the discussion of events critical to sequence planning. The graphed records were developed using a Quattro Pro spreadsheet used at the mine to record the stress data. All files were annotated by the blasting dates of the various stopes to aid in interpretation of the results.

7.2 HANGINGWALL GROUNDFALLS

The most common ground failure at the David Bell Mine is that of various sized blocks falling from the hangingwall into the open stopes, either during active ore removal or while the stope is open and awaiting backfill. These failures are not dangerous to personnel as ore removal is by remote control LHD machines so no access to the stope is required. They do, however, cause a dilution problem when they occur into an active stope and they necessitate secondary blasting if they produce blocked drawpoints. While an analysis of hangingwall stability is not considered primary to this thesis, a modified Matthew's analysis of stope stability was carried out by Leduc (65) as part of this research project. The following points are included as they are considered important in certain decisions which have been made about stope sequencing.

The hangingwall failure is facilitated by dilation of the A joint set which is parallel to the orebody and by the presence of sericite schist bands. These bands can be described as 90% sericite, with the sericite grains up to 5 millimetres in size, and can be several centimetres thick.

The B joint set, which is near vertical and perpendicular to the orebody, also plays a role in the failure. This set also dilates when the ore is removed and forms potential failure planes along the strike of the orebody. The hangingwall is further weakened if C set joints are present. These sub-horizontal joints are not always present, but when they are they have a spacing of 1-4 metres and are very continuous. This joint set, along with joint sets A and B, form large blocks which can fail with time if the mucking or filling cycles are delayed or the hangingwall is damaged by blasting.

A decision was made at the David Bell Mine not to install cablebolt support in the hangingwall of the bulk mining stopes. The usefulness of cablebolts installed in a radial pattern from overcut and undercut drifts to control such failures can be questioned. As well, the cost of hangingwall openings to provide proper cablebolt installation geometry would be significant in the relatively narrow ore of the David Bell Mine.

The fact that hangingwall stability is improved if the A and B joint sets have not dilated can be observed in the Level 1, 2, and 3 cut and fill stopes (65). These hangingwalls have been exposed for several years over a distance approximately 300 metres long by 10-15 metres high. These walls were supported by both cablebolts and Swellex and have remained stable. The differing ratio of the horizontal to vertical wall exposure in these stopes as compared to bulk stopes and the openings' interaction with the spacing of the joint sets are also significant factors. The stability of these walls convinced management to attempt a retreat sequence to remove portions of the 8D sill.

Hangingwall stability is also adversely affected by blasting damage. When overbreak causes the rupture of an individual bed or structural feature on the hangingwall it is difficult to prevent failure of the remainder of that feature into the open stope. This relationship was also observed in the cut and fill stopes. The adoption of a stope extraction method using radial blastholes drilled from the footwall drifts could only be considered if blast damage to the hangingwall could be avoided. With this method, increased attention to drilling patterns and reduced powder factors along the hangingwall due to increased spacing of the ends of the radial holes seemingly reduced the dilution problem.

7.3 THE COLLAPSE OF UNDERCUT DRIFTS

The David Bell Mine has had a history of stope backs failing along undercut drifts. In the early stages of mining, the ore undercuts were opened the full width of the ore zone, in some areas up to 10 metres wide, along the full length of the orebody. For support, 1.5 metre mechanical rockbolts were used. Many of the undercuts were created well before mining was scheduled, in fact, sometimes before the waste development was complete. This practice was stopped as soon as the problem was diagnosed and this has reduced the number of back failures. In previously established undercuts, additional ground support in the form of 3.6 metre grouted rebar was installed.

7.3.1 The Failure in 7B Sublevel

The first major stope back failure occurred in the 7B sublevel below the K72C and K74C stopes on September 29, 1989 (77). The fall of approximately 1500 tons was along a 40 m strike length and 9 m ore width and extended approximately 1 m into the back on the footwall and 2 m on the hangingwall. Crews were installing grouted rebar in the L76C area at the time. A tape extensometer in the L76C area had shown 4.64 mm of convergence in a 20 day period prior to the groundfall. Personnel just prior to the fall noted spalling in a bored slot raise in the K71B stope immediately below the fall area.

The author inspected the area in January 1990 and noted that the cablebolts which had been installed in the area were visible hanging from the back; the grout had obviously failed along the cables. Another smaller failure occurred in the area to the west of the initial failure in March 1990. Inspection of that area in January 1990 had shown cracks along the hangingwall and spalling along the footwall as if the back were attempting to rotate.

Much effort and discussion have gone into determining the mode and reason for the failure. Obviously the width and extended exposure time of the opening and inadequate initial support were all factors in the failure. Other recent failures have occurred, however, without these factors being present. Failures have also occurred at the adjoining Golden Giant Mine.

7.3.1.1 The Stress Effect

The presence of high stress was indicated by the inspection of the raise on the day before the failure. However, a stress cell at the top of the stope had not shown any appreciable increase in stress. The graph of the measurements from that cell, the K74C, 7C undercut cell, is shown in Figure 7.1. It was installed during the blasting of the two primary stopes, the L77B and K77B stopes, immediately to the east and west of the fall of ground.

The timing of the installation of the stress cell, as shown in Figure 7.1, corresponds well with the end of the second step in the TC4 numerical model as described in section 6.7.1. The predicted stress at the bottom of the K74C stope at the time of the failure was approximately 23 MPa. The graph of the stress cell corresponds well with the numerical modelling results. At the end of step 4, to early October 1990, at about the time of the K72C blast, the stress increase is about 7 to 8 MPa, very close to that shown in the stress cell. With the removal of the K76C stope in step 6 of the model, the stress jumps rapidly by about 20 MPa, again very similar to the reaction of the stress cell.

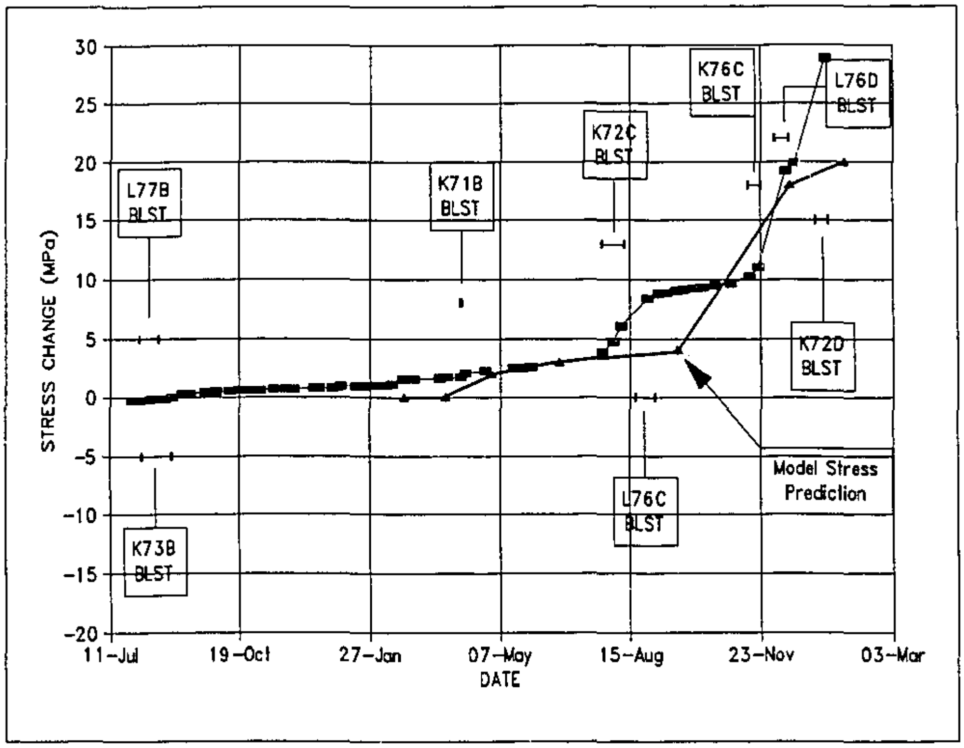

Figure 7.1 Stress Cell: K74C, 7C Undercut

Neither modelling nor stress cell data predicts the stress at the bottom of the K74C stope. Further validation ot this comes from a model of the actual extraction sequence was run to stope 69, the point at which this stress cell was lost. The results, ploted in Figure 7.1, show close agreement of the numerical modelling and the instrumentation results. As will be shown in section 7.4.2, the stress level in the secondary stopes increases rapidly only after removal of the adjoining primaries.

7.3.1.2 The Effect of Barite

The ore zone contains variable amounts of baritic ore. The baritic bands not only vary in thickness but also in barite content. The baritic bands are characteristically buff white, quite soft and range in thickness up to 2 metres but are generally 0.5 metre wide. Since these bands are so soft, they fail very easily and small pieces of the barite zone fall from the back. This leaves a trough in the back with brows on either side, allowing the adjacent layers to fail as the trough moves further up into the back.

Recent numerical modelling by Bos (72), using the FLAC modelling code has shown that the barite bands start to squeeze out of the model even at the moderate stress levels developed adjacent to one open stope. Work by Bos during the summer of 1992 led to the production of a contour map of the thickness of baritic zones as a percentage of the total ore thickness as logged in the diamond drilling at the mine. This contour plot, as shown in Figure 7.2, shows very close correlation between the location of groundfalls, the dark shaded areas in the plot, and the presence of high levels of barite which are shown plunging to the west across the orebody.

It is evident that the presence of baritic bands has been a major contributing factor in the occurrence of groundfalls.

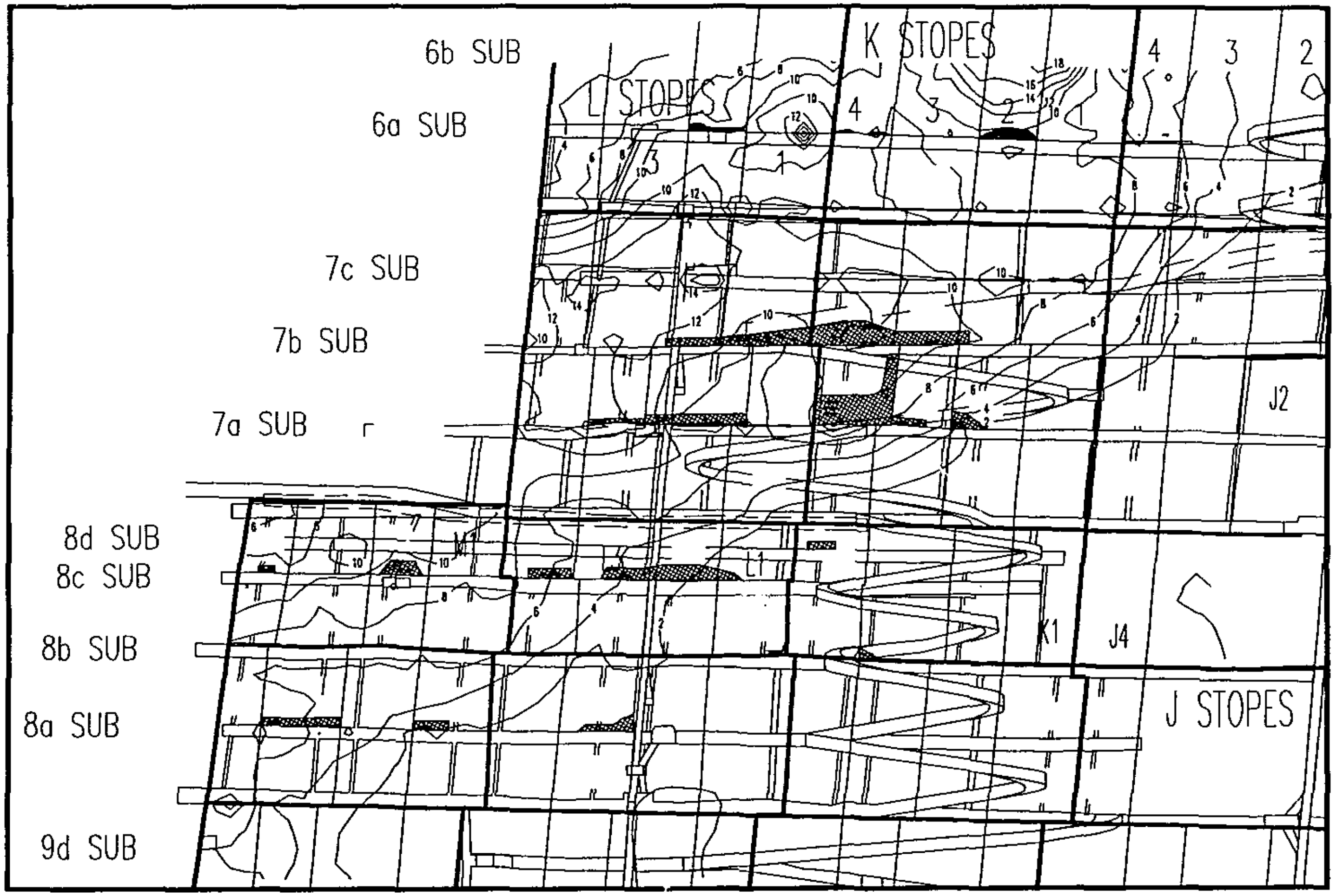

Figure 7.2 Barite Content Contour in Longitudinal Section

7.4 THE CHOICE OF PRIMARY STOPE SIZE

The initial choice of mining geometry had both primary and secondary stopes at 25 m in strike length, 50 m in height. This was quickly changed to 25 m in height, largely due to development delays during initial mining. Mine personnel observed, during the mining of approximately the first 20 stopes, that more stress related problems were encountered while extracting primary stopes than secondary stopes. Stopes 25 and 26 were initially designed as 15 m primaries along the Golden Giant boundary and based on the success of mining these stopes; stope 27 was the last primary designed at 25 m stope length.

Another important discovery was that there was a noticeable difference in the stability of the hangingwall in secondary stopes. There have been various theories put forward to explain this difference.

7.4.1 Theories on Hangingwall Stability

One explanation is that the hangingwall failures in the primaries are caused by bending failure from the intermediate principal stress which is parallel to the strike of the orebody and which is reinforced by the stress trajectory of the maximum principal stress bending around the opening of the primary stope. The failure of the hangingwall in the primary stope cuts the intermediate principal stress and the removal of the secondary stope cuts the maximum principal stress. The stress field is then greatly reduced as the near vertical minimum principal stress is only about 50% of the maximum principal stress. It is still adequate to provide a clamping action to overcome failure by gravitational load.

Important in all of these theories is the presence of the jointing planes and the sericite zones in the hangingwall. Access to the hangingwall is rarely provided but on a few occasions the author has been able to observe the immediate hangingwall from the overcut of a filled primary stope. Evidence of movement along the sericite shears was plainly visible.

The author's opinion is that the secondary pillars are in fact destressed to some degree as relaxation of the hangingwall occurs into the primary stope. Horizontal movement along the sericite shears allows adequate closure for the maximum principal stress to be reduced. While details of the rock behaviour can be crucial in choosing the optimum mine sequence, confirmation would require a specially designed instrumentation program installed from hangingwall access drifts.

7.4.2 Stress Cell Evidence

The behaviour of the stress front can be seen from an analysis of the stress cell data. The graph of the K74C, 7C undercut cell, as shown in Figure

7.1, depicts the stress behaviour at the top of the K74C stope, stope 73 in the extraction schedule, or more precisely the stress at the bottom of the K74D stope. It was installed just prior to the removal of the L77B and the K73B stopes, the number 23 and 24 stopes, which were the remaining primaries between the 7A and 7B levels immediately below the stope. During the removal of the remaining primaries and secondaries between the 7A and 7B levels the stress level changes by only 5 MPa. This is an unusually low stress increase when total extraction has been achieved along the level immediately below. It must also be remembered that the K74C stope had partially failed and that sublevel spacing in this area was somewhat reduced.

The first significant jump in stress level occurs when the primary stope to the west, the K72C stope, is removed as stope 48. This increase escalates when the primary to the east, the L76C stope, is removed as stope 54.

Two other stress cells were located in adjacent holes in the 7C haulage drift of the K72C stope. The results from the K72C-FW, 7C haulage stress cell are given in Figure 7.3 and the results of K72C-FW-AS, 7C haulage are given in Figure 7.4; the latter being a cell oriented to give stress readings along strike. Figure 7.3 shows that there is no measured stress increase as the stopes along the 7B drift are extracted, even the removal of the K72C stope immediately below does not affect the cell. The only major measured change is a 6 MPa drop when the K72D stope immediately up foliation from the cell is removed. The along strike stress jumps briefly and then is stable until some relaxation occurs with the removal of the K72D stope.

Exactly which theory this data supports is difficult to determine. The stress increase ahead of the mining front seems to have a very limited extension into the abutment, advancing in conjunction with the primary stope removal. Numerical modelling of low modulus rocks would predict a more gradual stress gradient. The changes in the footwall haulage are less than expected in a drift only 20 metres from the ore but the relaxation of the stress with stope removal is well documented.

7.5 THE ALTERNATE MINING METHOD

An interesting adaptation of blasthole mining has been developed at the David Bell Mine, initially in response to the falls of ground which occurred in the stope overcuts. The entire stope is drilled in a radial pattern from the footwall haulage drift of the level below. Since the footwall drifts are approximately 20 metres away from the ore zone, this requires the drilling of more than twice the hole depth in waste for the average 10 metre ore.

The initial cost savings were achieved because there was no need to

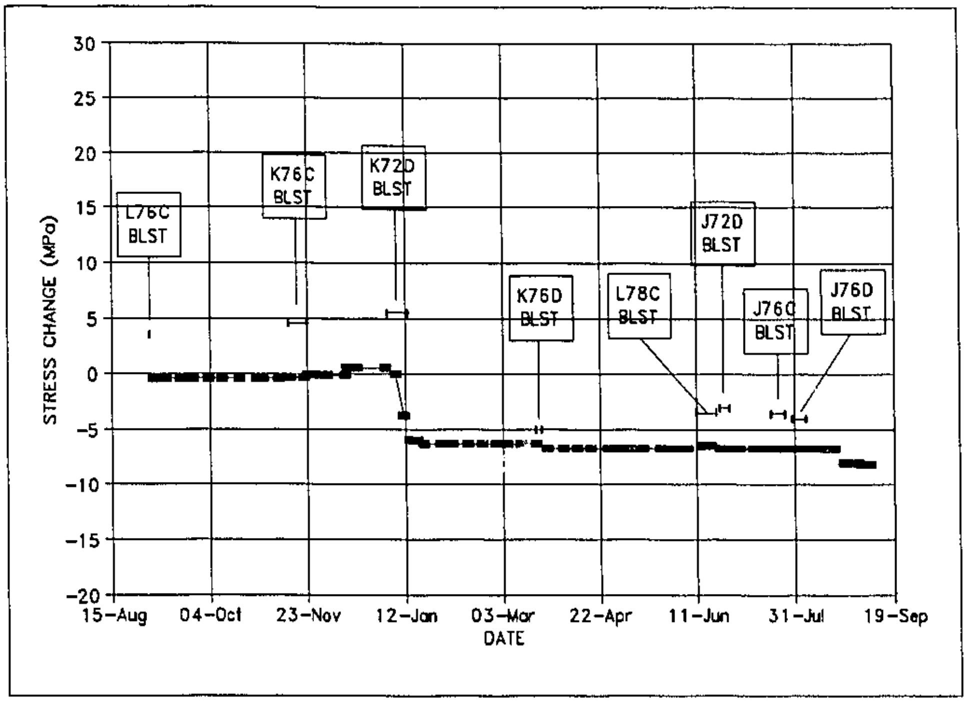

Figure 7.3 Stress Cell: K72C-FW, 7C Haulage

rehabilitate the overcut drifts in caved ground. However, even if the ground has not caved, blasting of the primary stopes may require rehabilitation of the overcut of the secondary stopes before personnel can enter. Not only is there a cost associated with rehabilitation but the time required may delay the extraction cycle.

There are additional savings to be gained from this method. In certain situations, the overcut can be eliminated entirely. Also there is no requirement for a slot raise or for two stages of blasting to expand the slot raise before the

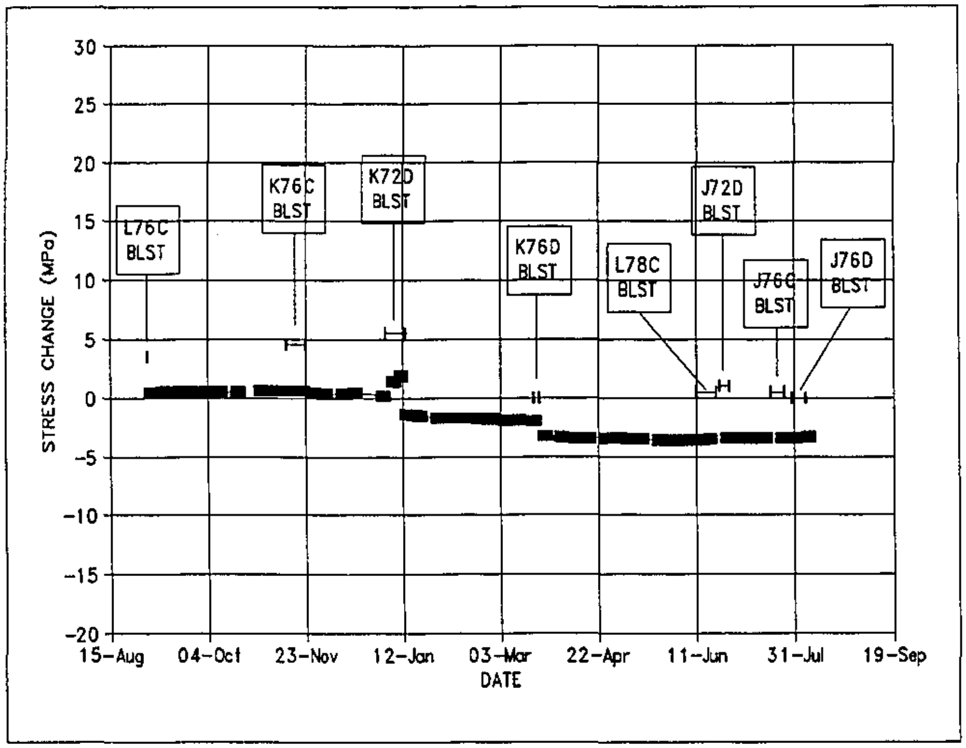

Figure 7.4 Stress Cell: K72C-FW-AS, 7C Haulage

remainder of the stope can be mass blasted. The stope can be run as a vertical retreat stope without the requirement for the large diameter holes; this could improve the stability of the hangingwall if broken ore was left in the stope during the blasting process.

This method could not be successful if fragmentation was not maintained or if the holes were not terminated promptly at the hangingwall leading to blast damage and dilution from hangingwall failures. There is no controlling feature to alert the drillers to terminate the holes; that is, no change in drill cutting colour or drill advance rate would be expected at the hangingwall limit. The orebody outline is quite regular however, so the plans for the blasthole depths can be accurately fixed.

Another potential problem is damage to the footwall from the collars of the radial holes if the loading of the holes is not closely controlled. None of these problems appear to occur and mine personnel report improved fragmentation.

This method helps deal with stress related ground control problems. The drilling and loading is carried out from the safety of the footwall haulage drift and a remote controlled LHD removes the broken ore. A remote controlled blockholer has the capability to drill and load oversize material for secondary blasting. The fill fence is erected in the safety of the crosscut away from the stope brow and the fill pipes installed from the top. This is, in effect, a non-entry mining system which has allowed the removal of the 8D sill pillar with a minimum of increased costs.

7.6 THE 8D SILL PILLAR

During development of the main zone, the mill was supplied from the cut and fill stopes on levels 1 through 3. As already stated, the main zone was silled out in ore quite early in the development stage. An extra sill was driven off the ramp halfway between the 8C and 7 levels, over a distance of approximately 275 metres, although no sublevel was developed at this elevation. Ostensibly, this drift would serve as a drill drift for upholes to extract the ore beneath the 7 level, although opening such a drift so far in advance of ore extraction would not be considered good mining practice.

The more obvious reason for this drift was to augment the ore supply during the transition from the upper levels to the main ore zone. While this may sound trivial, the undercut represents about 30,000 t of ore which in this area has an approximate value of $6,000,000; this is certainly not trivial for a junior company developing a new mining operation.

As a result of this decision, the 8C to 7 level sill which would have been 25 metres thick horizontally was now two 10 metre sills, both of which would have had a blast damaged zone along the top and bottom. As discussed in section 6.6, this sill was shown to have failed quite early in the extraction sequence.

The first sign of problems in the 8D sill drift was the snapping of rockbolt heads, the development of cracks in the back of the drift, and the increase in stress readings in the stress meters installed in the drift. A rehabilitation crew was assigned to the drift to install 3.6 metre grouted rebar. The author inspected the drift on the day the rehabilitation crew was pulled out of the area. Heads were snapping off both the rockbolts and the rebar. Cracks up to 25 centimetres wide had developed along the hangingwall and spalling was seen along the footwall similar to that observed in the 7B sublevel prior to the failure. Within two weeks of that inspection, the back had failed along much of the drift although inspection was impossible as access was available only at the ramp access and through one raise at the western end.

To control the further deterioration of the area the drift was filled with hydraulic backfill.

7.6.1 Documenting the Failure

The stress cell readings in the 8D undercut have been presented in Figures 6.14 and 6.15. The initial drop in stress was recorded in the L84D, 8D undercut cell but further access to these cells was not possible; in fact, the author still remembers the rapidity with which the final readings were taken. This relaxation corresponds to the observed cracks along the hangingwall.

Additional evidence of the failure can be seen in Figure 7.5 which shows the results from the stress cell installed in the L86C-FW, 8C haulage. The relaxation which begins in late January 1990 cannot be correlated to any of the blasts but corresponds to the failure recorded in the 8D drift. The failure in the 8C haulage

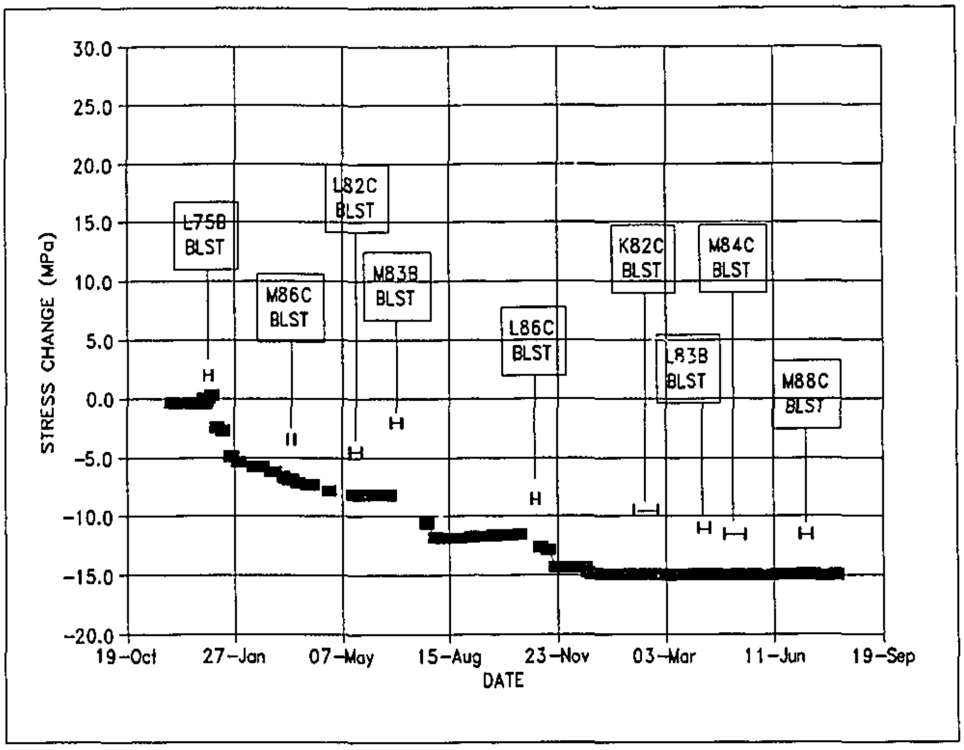

Figure 7.5 Stress Cell: L86C-FW, 8C Haulage

involves the material between the 8D drift and the 8C undercut, whereas the previous readings are for the material between the 8D and the 7 level. This progressive relaxation to November 1990 is consistent with a gradual failure of this lower sill.

Figure 7.6 shows the results for a cell installed in the L86C, 8C undercut. It was installed late so it shows only the end of the stress build-up but documents the relaxation in late January 1990. The results in Figure 7.7 show the gradual increase of stress on the L84C, 8C undercut cell on the western edge of the documented failure. There are several intact stopes to the west on this level. The stress increase is gradual from January to June 1990,

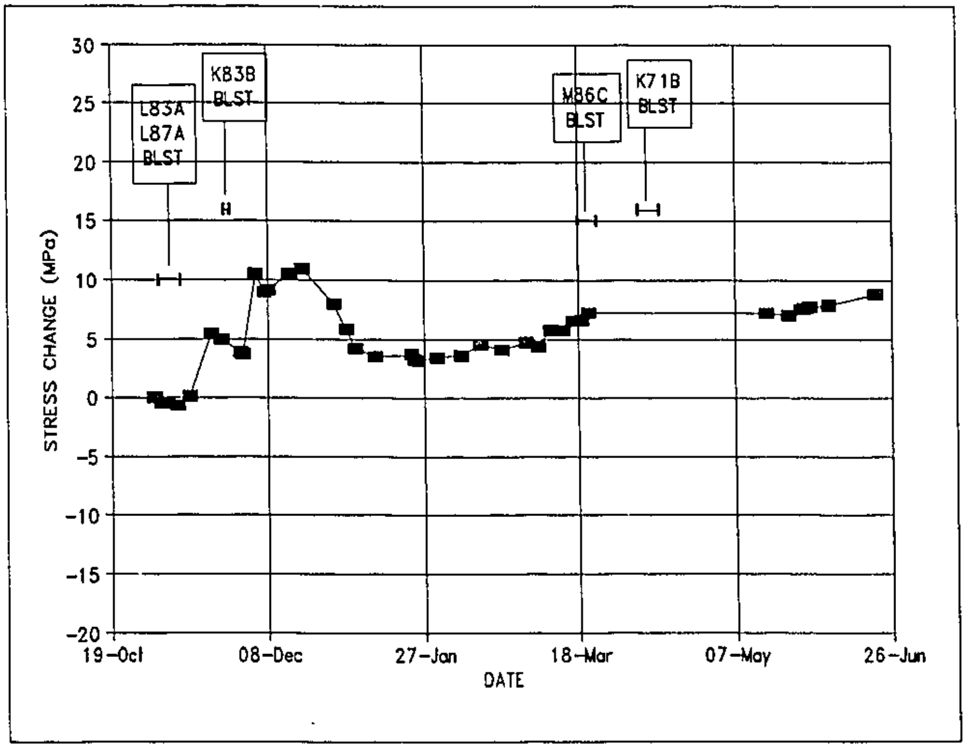

Figure 7.6 Stress Cell: L86C, 8C Undercut

consistent with a gradual failure of the 8D to 8C sill. The rate of increase accelerates on removal of the adjacent primary, the L82C stope, as stope 42. The rapid increase on removal of the M83B stope is unexplained unless there has been additional failure to the west along the overlying sill.

7.6.2 The Sill Removal Sequence

It was intended to use the alternate blasthole mining method described in section 7.5 to extract the 8C to 7 level sill pillar. The first difficulty presented

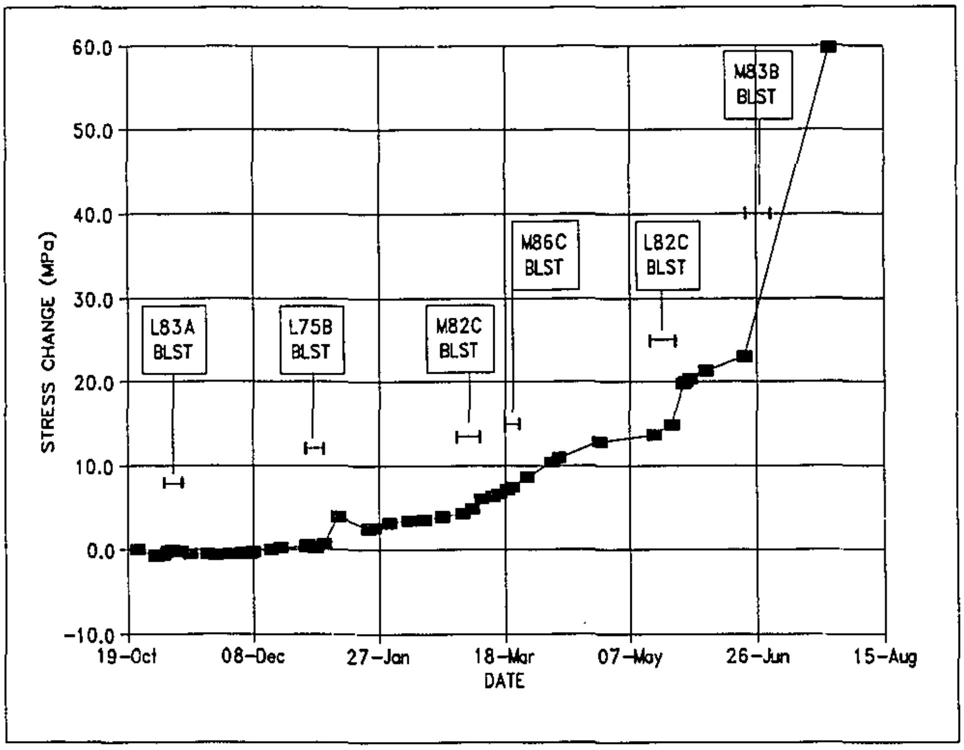

Figure 7.7 Stress Cell: L84C, 8C Undercut

by the presence of the 8D sill was that these holes could not be drilled from the 8C footwall drift into all the ore in the sill as the 8D drift created a section of ore which could not be reached through the collapsed rock and backfill. This necessitated hangingwall access for drilling this portion of the ore.

To overcome this problem, an access drift was developed across the backfill at the M88D stope as shown in Figure 7.8. This drift was created by placing an Armco arch and pouring backfill around it. A drift was then driven in the hangingwall for drill access as far east as L88D. To the east of L88D, the

8D drift was full ore width so ore above the drift was drilled by downholes from the 7 level footwall drift and the ore below it drilled from the 8C footwall drift.

Section 6.7 described the numerical models which were run to assess the stress build-up consequences of various sequences of extracting the 8C to 7 level sill. Although it was known that the ground had already failed, it was hoped that the elastic models would give relative data on resulting load and closure from the various alternatives. The stability of the overlying backfill was considered critical, as a major backfill failure would cause considerable ore dilution.

The first stope extracted was the K86D stope near the eastern end of the sill. The thickness of the ore at this point was 3 to 4 metres. Considerable rock rubble had been left on the floor of the 7 level, through which the cemented hydraulic fill had percolated causing consolidation; in fact, the rock rubble was more stable than the hydraulic fill exposed at the eastern end of the stope. The material failed to a stable arch approximately the height of the opening width, exactly as predicted by the centrifuge and numerical models described by Pelley and Mitchell (52).

The K82D stope was removed next and the results in the wider ore were similar to the previous stope. The stability of the hangingwall in these two

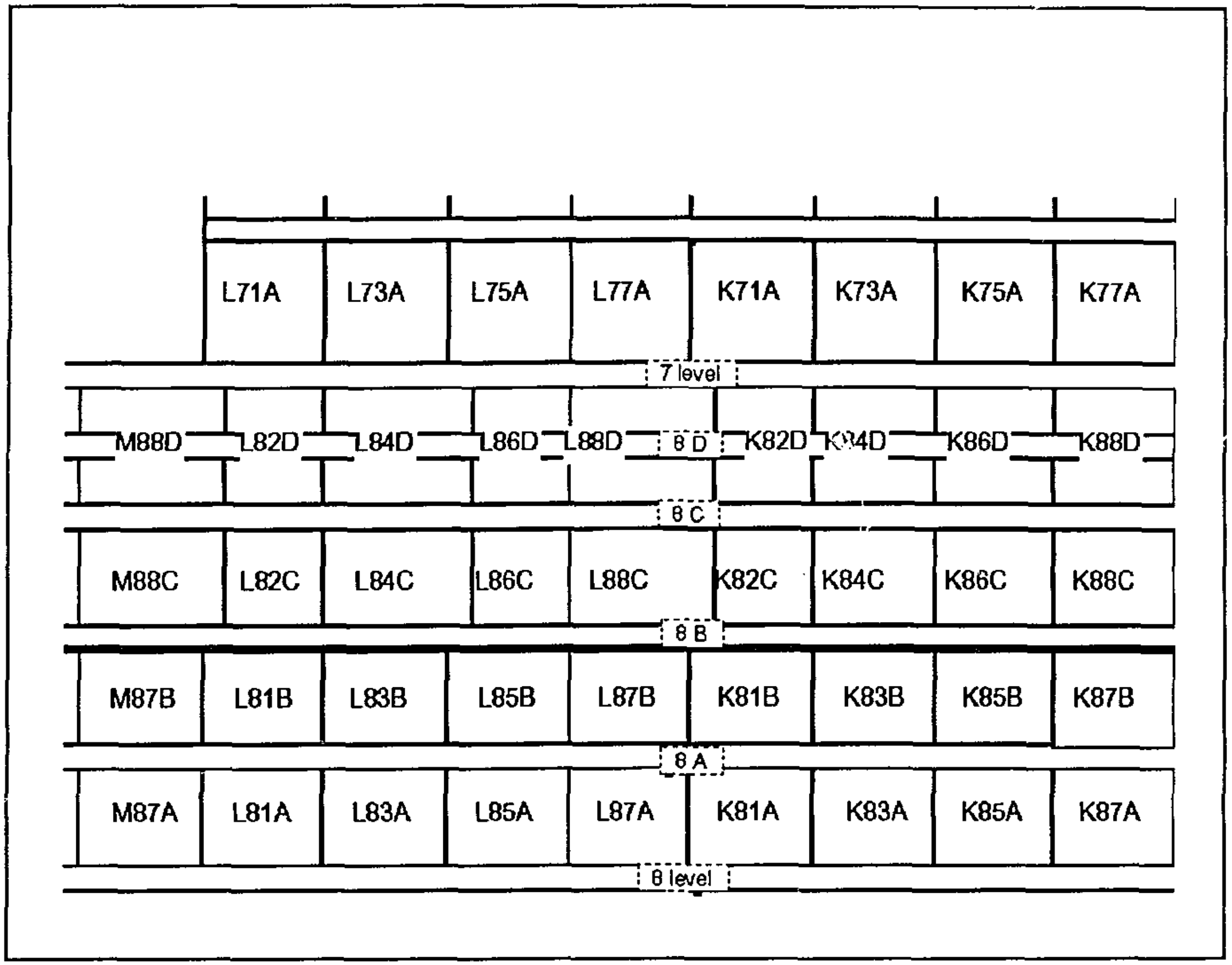

Figure 7.8 A Longitudinal Section of the Stopes at the 8D Sill

stopes, and a trial retreat sequence tried in thin ore at the eastern end of 7B level, led to the decision to extract three stopes together. The L82D, L86D and the L84D were extracted in that order using the radial holes and blasting the holes in a retreat sequence from the bottom up. All three stopes were extracted before the area was backfilled. When the final skin of ore was blasted below the backfill, the fill failed again to a stable arch. Considering the approximate 9 metre ore width at this end of the sill, however, this represented significant dilution and some ore loss was experienced.

As these three stopes were being mined, the K84D stope was extracted. Subsequent to filling these stopes, the L88D and M88D stopes were extracted to successfully complete the removal of the 8D sill by the end of 1992.

7.7 STRESS TRANSFER WITH GOLDEN GIANT

An interesting study in stress transfer is provided by the stopes at the western end of the 8B, 8C and 7 levels near the Golden Giant property. Figure 7.9 shows the results from the stress meter in the M83B, 8B undercut, in reality in the bottom of the M84C stope. The stress readings increase rapidly with the blasting of the adjacent M82C primary and continue with the blasting of the second primary, the M86C stope. The level of increase with removal of the first primary is higher than expected until one realizes that the flatback

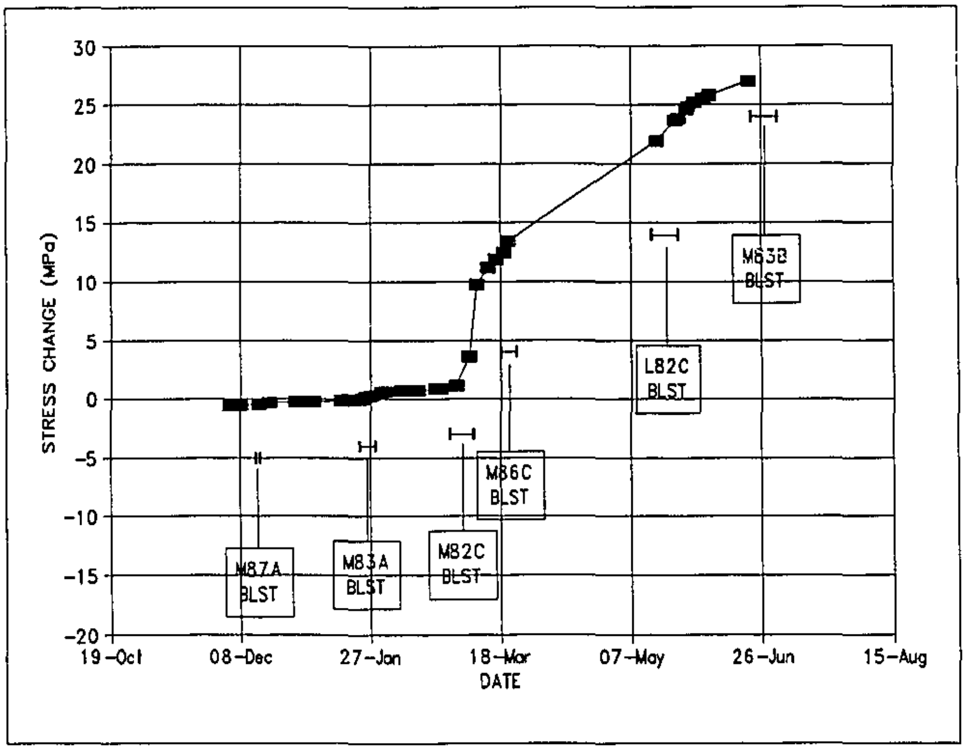

Figure 7.9 Stress Cell: M83B, 8B Undercut

stopes as described in section 5.3.5 at the Golden Giant Mine is immediately adjacent to the west.

The same situation is demonstrated by the results from the M84C, 8C undercut stress cell as shown in Figure 7.10. This cell, actually in the bottom of the M84D stope, shows a much higher stress increase with removal of the adjacent primaries than demonstrated by the cells at the upper mine abutment as discussed in section 7.4.2. Again, the proximity to the flatback opening at the Golden Giant Mine is a factor.

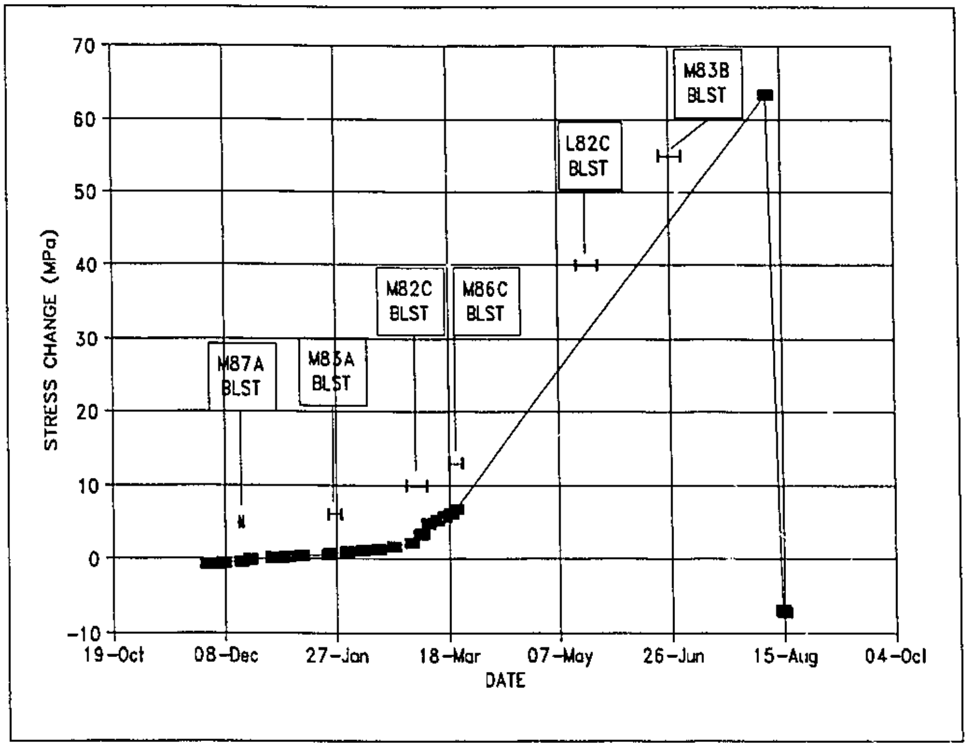

Figure 7.10 Stress Cell: M84C, 8C Undercut

Finally, the stress cell results in Figure 7.11 are from the M84D, 7 level undercut, in reality in the bottom of the Golden Giant shaft pillar. The removal of the adjacent M82D and M86D primaries caused a stress increase of only about 10 MPa because the stress is now transferring to the abutment, in this case the Golden Giant shaft pillar.

7.8 CONCLUSION

A mine monitoring program is important in understanding the ground

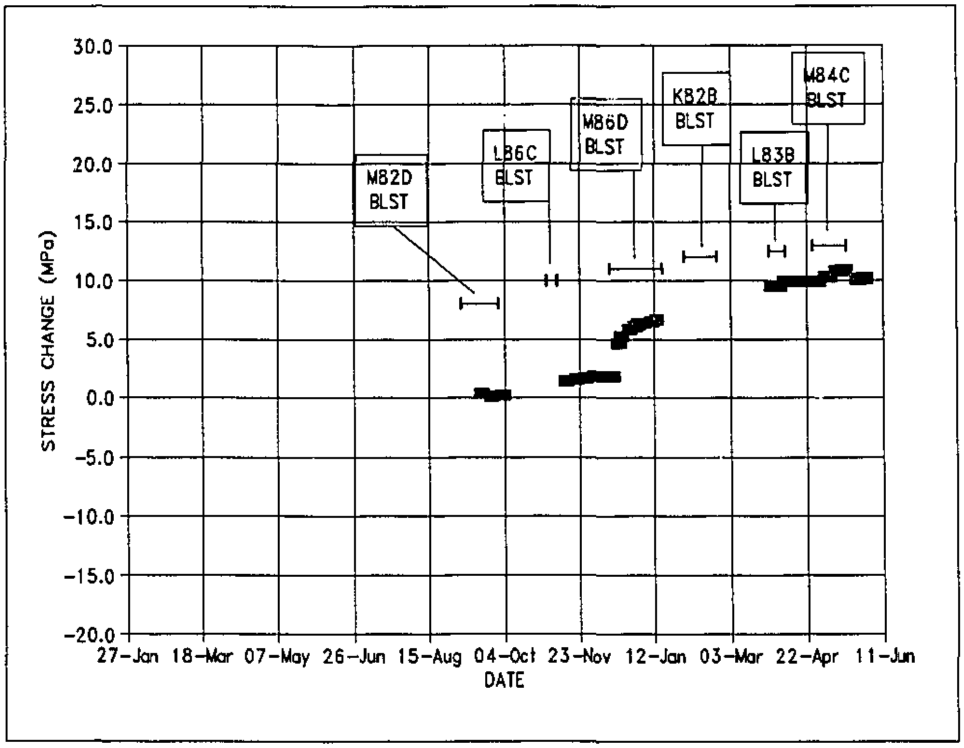

Figure 7.11 Stress Cell: M84D, 7 Level Undercut

behaviour; understanding that behaviour often leads to adjustments in the mine plan. Knowing that the 8D sill had failed led to a complete change in the philosophy of extraction planning. The intent to maintain a sequence which offered pillar stability was changed to one that ensured maximum ore extraction in the event of a major backfill collapse. Those adjustments changed the detailed extraction sequence but the changes were compensated for by the flexibility of the long range plan.

Adaption of the mining method to meet the objectives of the long range plan leads to a more logical mine sequence. The operating personnel did not divert to an alternate area when a major ground failure occurred but adapted the method to provide a timely extraction of the underlying stopes. While this decision may have been inspired by a lack of advance development, the result was the timely upward advance of the extraction front.

The decision to create an 8D drift was a decision of convenience made early in the mine life which has had long term consequences. This decision caused the early failure of the 8D sill area which in hindsight may have been fortuitous. The failure necessitated the extra development in the hangingwall and caused dilution from the backfill placed in the 8D drift. The dilution from the 7 level backfill was probably not much more than if the original sill had been intact, and possibly the dilution from the hangingwall was reduced by increased stability in a destressed state.

The important consideration is that operational decisions based on a knowledge of the ground conditions gained from numerical modelling, on-site observations, and instrumentation data led to successful completion of the extraction sequence with minimal delay, loss of ore, or increased costs.

8.0 SEQUENCING OBJECTIVES AND STRATEGIES

This chapter accomplishes two goals set out for this research project. First, the sequencing objectives introduced in section 1.4 are discussed in more detail in terms of their financial impact on a mining venture. Examples from Chapters 3 through 5 are used to illustrate how the move to bulk mining sequences has affected the mine designer's ability to achieve each objective. The classification of mine extraction sequences is then discussed based on recent changes in mining practice.

To evaluate the sensitivity of a project's economic outcome to the varied sequencing objectives, an analysis of their effect on the profitability of a mining venture was carried out. The following simple economic model was chosen as the base case for this analysis. Any similarity with the case study used in Chapters 6 and 7 is for convenience only, it is not based on actual mine costs.

8.1 THE BASE CASE

A first step in evaluating a deposit as described in Table 8.1, would be to establish the initial production level, possibly using Taylor's equations (78), at approximately 650,000 t per year or 2500 t per day based on a 5 day per week operation. Assuming such an operation required a capital investment of

$150,000,000 over a two year pre-production period, an annual operating profit of $50,000,000 would be required to give an after tax return of 18.6% on the investment, a modest margin over the 15% hurdle rate that would considered a minimum return on such a venture.

TABLE 8.1
BASE CASE ECONOMIC PARAMETERS

Ore Tonnage (t)	6,500,000
Ore Grade (gm/t)	12.8
Annual Production Rate (t)	650,000
Years of Operation	10
Gold Price ($/gm)	12.5
Effective Tax Rate (%)	50
Gold Recovery (%)	93
Revenue per tonne	$148.80

To achieve the required $50,000,000 per year operating profit requires the mine to have an operating profit of $76.92 / t, or to operate at a cost of not more than $71.88 / t. This equates to a cash cost of $6.04 per gram ($193.22 per ounce), not unusual for a mine of this grade and tonnage.

The economic sensitivity of this mining venture to a failure to achieve each of the objectives will be examined in the following sections. The results are summarized in Table 8.2. Examples of sequences which either attempt to achieve or which are counter to achieving each of these objectives are then provided, drawing on the discussions of sequencing from the previous chapters.

8.2. MAXIMIZING MINERAL RESOURCE EXTRACTION

Maximizing overall mineral resource extraction should be an objective in all high value orebodies. Governments may dictate this or encourage it through taxation. For example, in 1933 South Africa introduced an excess profits tax which favoured high recovery, low profit producers over low recovery, high profit producers in an effort to eliminate high grading and extend the life of orebodies. Similar Minnesota rules forced the Reserve Mining Company to operate a pit almost ten km long to produce at the deposit average iron grade although the grade varied only slightly from one end to the other.

8.2.1 A Financial Discussion

In 1954 Carlisle (79) described a pit where zonal grade variation allows increased metal recovery by decreasing the cutoff grade. His graph, reproduced in Figure 8.1, shows the average fixed costs (AFC) and average rate variable costs (AVC_R) which decrease with an increased level of recovery. The AFC depend on the fixed capital costs of the mine and its infrastructure and the AVC_R on the capacity dependent capital costs, the cost per metal unit of both being lowered as more tonnes are recovered. Carlisle shows the average level variable cost (AVC_L) increasing as the marginal level variable cost (MC_L) increases, as does the average total unit cost (ATUC). Since the additional recovered material is at a lower grade, the marginal cost increases mainly as a result of less metal being recovered per tonne handled at a constant unit cost.

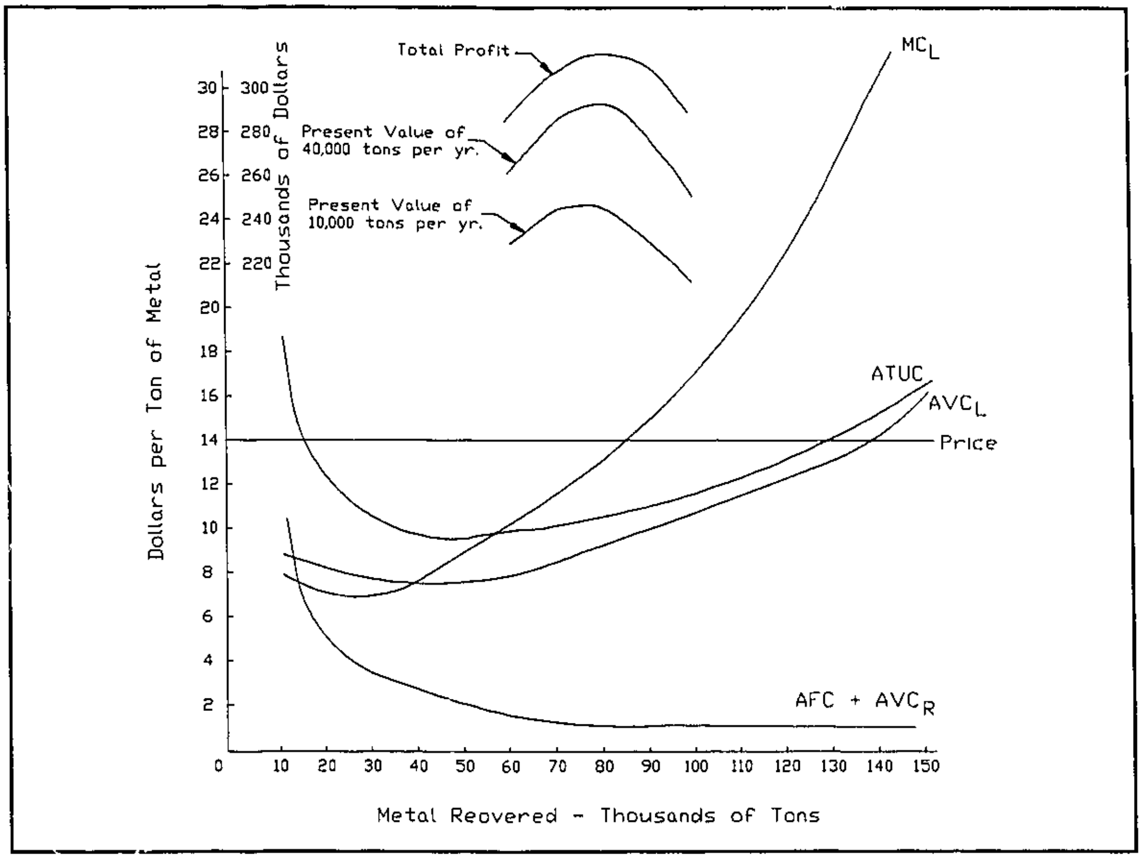

Figure 8.1 Alternate Levels of Recovery, from Carlisle (83)

Carlisle then shows that the maximum profit is realized as long as MC_L remains below the price of the product. The profit per metal unit is maximized at a point where the MC_L is the same as the ATUC, as will the profit per unit of time at a fixed production rate. The net present value will be maximized somewhere between these points, the exact optimum point depending on the relationship between the costs and the discount rate.

In underground mines the relationship is more difficult to evaluate. The fixed costs would be similar to those in an open pit and the rate variable costs would be based mainly on the main access facilities, the shaft and ventilation system for example, and should behave in a similar manner. There are, however, three major components to the MC_L curve as follows:

1. Similar to the gradational cutoff grade described by Carlisle, a cutoff grade may be applied to the hangingwall or footwall of the orebody based on the metal content paying for the marginal costs of mining, removing and milling the material. This is based on the assumption that all development costs for that stope have already been incurred reaching the higher grade material. Ore losses at this point are normally permanent as, once the stope is mined, the cost of returning for the remaining low grade material would be greatly increased.

2. There is a development cost required to access each area of the mine. If lower grade material exists at the orebody extremities, additional development costs must be incurred to access it, requiring a different cutoff grade. As lateral development costs are generally fixed based on longitudinal area, there is also a variable development cost per unit of metal based on the thickness of the orebody.

3. Both of these marginal cutoff grades are further affected by increased ground control costs required to maximize extraction. Increased ground support or better quality backfill may be required in all stoping areas, for example. These costs will usually be incurred throughout the mine life whereas increased extraction benefits only the total mine life.

To examine the economic sensitivity of the base case mining venture to maximized resource extraction, it was assumed that only 80% of the ore was recovered over an 8 year operating life at the same grade, capital cost, and operating costs. This would be the situation if 20% of the ore was left in pillars. The return on investment now drops by 10.9% to 16.56% from 18.6%.

It is realized that the rate of access level development would be increased per recovered tonne but this may be compensated for by eliminating some development, cross-cuts to pillar drawpoints for example. With less ore being recovered, however, savings in operating costs may be achieved, by eliminating a cementing agent in the backfill or by eliminating a more expensive pillar recovery mining method for example. If the operating costs could be reduced by 10%, to $64.70 /t, the return is once again 18.6%. If it were possible to reduce the ore recovery by leaving lower grade material, the situation would be even further improved.

Even in relatively high grade material, it can be seen that project economics, at the feasibility stage, are not sensitive to complete orebody extraction. Of course, as the mining operation proceeds, the future NPV will be maximized by extending the mine's life, particularly if decommissioning costs can be delayed. These are separate decisions, however, to be made under more certainty later in the life of the mining operation.

8.2.2 A Discussion of Examples

It is difficult to show that the change to bulk mining methods and sequences has enhanced the level of resource extraction, in fact it is difficult to provide an example where that was the prime objective. While there are excellent examples of where it has been achieved, there are other examples of where it has not been attempted or where it has been precluded or placed in jeopardy for other considerations.

The first highly mechanized mining method, post-pillar cut and fill mining, does not attempt complete recovery of the resource. Section 4.1.1 describes, at the Strathcona Mine, how metal recovery can be improved over ore recovery by reducing mining costs to allow a lower cut-off grade along a gradational hangingwall contact. Increased hangingwall recovery was precluded at Onaping by the vertical sill pillar shown in Figure 4.1, which prevented timely access.

From the traditional methods, total extraction has been precluded at the Campbell Mine by the boxhole shrinkage stoping described in section 3.1.3. While a high extraction ratio was achieved, approximately 83%, removal of the ore now tied up in sill and boxhole pillars has been impossible. In the F zone, broken ore in shrinkage stopes has been placed in jeopardy. While O'Flaherty (37) states that A zone sill pillar recovery was abandoned in 1970, a fatal accident occurred in 1989 while attempting just that. The use of an alternate layout, a vertical extension of the Stanleigh design as described in section 3.4 for example, might improve the possibility of recovering the total resource.

Total resource extraction has been placed in jeopardy at Brunswick Mines, as shown in Figure 4.2, although based on the economics of the time that was likely the financially optimal decision. Chapter 5 described the series of decisions which has caused some ore at the Golden Giant mine to be lost or placed in jeopardy. Not considering the ore tied up in the crown and shaft pillars which may eventually be recovered, there is ore in a sill pillar between blocks 1 and 2, below the quarter claim ore, and other pillars will develop as mining areas converge. Grant and Potvin (58) have described ore below the 4400 level near the William's boundary which will also be slow and more expensive to recover.

The evolution of the Sudbury philosophy of complete extraction, except for post pillar mining, was described in section 3.2. A major exception to that philosophy occurred at the Falconbridge Mine where ore adjacent to number 1 flat fault was lost on mine closure. Some ore has been lost on conversion to bulk methods, the skin of ore left around the VCR stopes at Levack for example, or placed in jeopardy as in the sill pillars left between the 1-5-9 sequences used at Lockerby, as shown in Figure 4.5.

The Hemlo area provides two examples of sill pillars between converging sequences, created in order to provide two working levels. The detailed analysis of Chapters 6 and 7 described the numerical modelling, instrumentation and mining method adaptation which aided the recovery of the 8D sill pillar at the David Bell Mine. That recovery was at the expense of a reduced production rate during removal, increased operating and development costs, and increased dilution. The recovery of the pillar at the William's mine is underway. Early planning and preparation for the removal of this sill should ensure success.

The best example of total resource extraction is provided by the Lakeshore Mine as described in section 3.1.1. Under adverse rockbursting conditions, a rill system was adapted which greatly reduced productive capacity. The temptation must have been to go deeper and place new areas into production, thus jeopardizing some reserves. That was not done, however and the extraction of the resource ended with the recent recovery of the crown pillar by Lac Minerals.

In general, all the single pass mining sequences provide complete extraction. The sequence adopted by Kidd Creek, shown in Figures 4.6 to 4.9, provides a high mining rate by repeating itself every three levels but extracts the sill pillars in a timely and complete fashion. The Creighton example given in section 4.3.2 shows a system which provides a large number of working stopes once the initial slot of stopes is established.

Generally, the switch to bulk mining sequences has improved the ability of most mines to increase resource recovery through the increase in productive capacity of the individual stoping areas. Therefore, fewer working areas are required to reach a certain productive capacity creating fewer pillars which either increase costs or place ore recovery in jeopardy.

8.3 THE OPTIMAL SUSTAINABLE RATE OF EXTRACTION

In an underground mine, the optimal sustainable rate of extraction is based on the thickness and longitudinal area of the orebody. The initial production rate may not be sustainable, for example, if a separate deposit is found late in the life of the operation and a mine capacity already exists or if the orebody thickness decreases in the latter stages of the operation. In general, especially from a ground control point of view, the greater the rate per area the greater is the difficulty of developing the optimum sequence

8.3.1 A Financial Discussion

Generally, developing a higher production rate for an underground mine requires a larger capital investment, although the increased investment is not linear per unit of productive capacity. As an example, the cost of a larger capacity shaft per unit of capacity decreases rapidly compared to the basic costs of providing the initial opening.

Carlisle (79) developed the graph reproduced in Figure 8.2 for a uniform deposit, mined by open pit, in which the limits of grade and tonnage are fixed. He assumes that the average fixed costs (AFC) and average variable costs based on the level of recovery (AVC_L) remain constant per unit of production. As the production rate increases, the marginal cost (MC_R) decreases as long as the economy of scale applies and the average total unit cost (ATUC) decreases.

As the extractive effort becomes too concentrated, the economy of scale is lost and costs may escalate rapidly. The total profit is maximized at a point where the ATUC is minimized, the point at which the MC_R increases above the ATUC. The profit per unit of time is maximized at the point when the MC_R matches the product price. The net present value is maximized between these points; the higher the cost of capital, the higher the justified production rate.

Again, the situation underground is more complex as an increase in productive capacity carries a higher cost than in an open pit. Koniaris (1) assumes that mining capacity in an open pit can be added at no cost, which is the situation in some heap leach operations in the United States where mining is contracted out. In an open pit, additional production units causes interference among units although not as rapidly as in an underground mine. In an open pit the productive capacity can be increased by increasing the size of productive equipment. In underground mines, that increases the cost of development

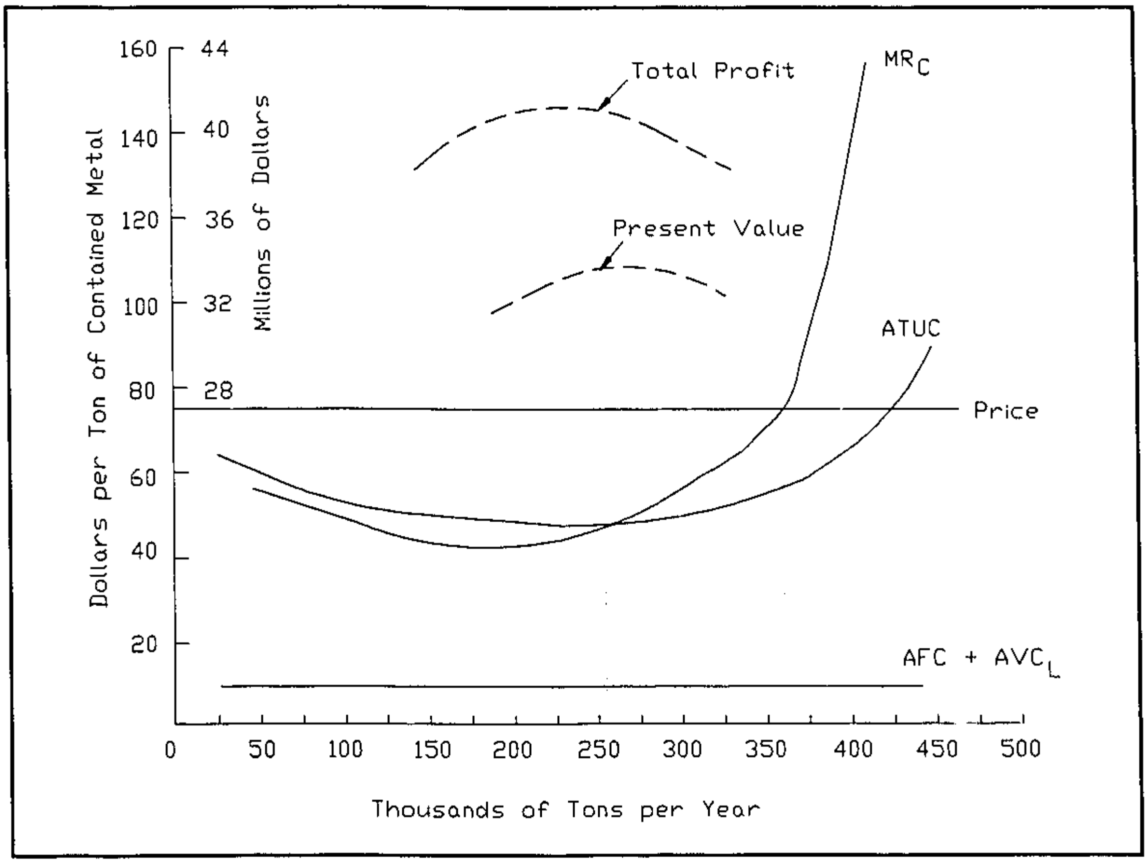

Figure 8.2 Alternate Rates of Recovery, from Carlisle (83)

openings and possibly increases ground control problems. The provision of additional working places underground also generates sequence problems.

In examining the effect of increased production on the base case economics it was calculated that a 10% increase in the production rate, at the same operating cost, in conjunction with a similar increase in capital costs in fact would reduce the return on investment by 5% to 17.7%. A 10% increase in productive capacity with no increase in capital costs would see the return increase by 7% to 19.9%. The reality is somewhere in between but is also dependent on how the operating costs varied with the changes in production.

Generally, the profitability of short life projects is not greatly enhanced by increases in productive capacity as the increased capital requirement is not justified by the reserve base. Also other factors are likely to affect the final choice of productive capacity, financing for example. Banks generally limit loans to amounts which can be repaid in half the reserve life using half the positive cash flow. This will limit the size of physical plant many junior companies can finance, placing a limitation on the mining rate which is attempted.

8.3.2 A Discussion of Examples

The introduction of bulk mining methods and sequences has enhanced the possibility of increasing underground production rates. O'Flaherty (37) describes the A zone at Campbell producing only 10,000 t per year using conventional cut and fill stoping and, as such, having 75 years of reserves. Since that is longer than the overall life, a reserve sequencing problem was developing. The move to longhole methods saw 56,971 t removed in one stope blasting sequence. Although the productivity per manshift increased by only 25%, because of the increased development required, the ability to produce from the area increased greatly.

One stated reason (46) for going to bulk mining at Brunswick Mining was that with only one major level, the 1000 level, remaining at depth to replace the three upper ramp cut and fill levels the area productive capacity had to be increased. The productive capacity of the Creighton mine, at depth, will have to be maintained from two to three levels maximum when the pillar removal from the upper levels is completed. The examples of the 1-5-9 sequences from the Falconbridge operations, as described in section 4,2 and discussed more fully in the following section as a cost reduction example, also greatly increased the productive capacity of the individual stoping areas.

While the capital cost of the physical plant and services will likely be the limiting factor in a mining operation's capacity, the use of continuous bulk sequences certainly provides the capability of increasing mining rates.

8.4 MINIMIZING UNIT COSTS

Minimizing the cost per tonne of ore extracted is a stated objective for most mine operators. It should be emphasized, however that the cost per unit of product is a more important criterion. Productivity may be increased and the cost per tonne hoisted may be reduced by handling dilution either from the host material or from backfill. This will have a detrimental effect on the cost per unit of product however.

8.4.1 A Financial Discussion

The profitability of a mine is moderately sensitive to changes in operating costs. The sensitivity is reduced somewhat by the Canadian taxation regime whereby decreased costs, and therefore increased profitability, leads to more rapid depreciation of capital expenditures and the more rapid assumption of full scale taxation. Conversely, increased costs are shared by governments to the level of the taxation rate, although the move to a minimum tax in some jurisdictions provides added incentive to increase profitability.

In the example under consideration, a 10% increase or decrease in the operating cost sees the base case 18.6% ROI change by 11.8% to 16.4% and by 11.5% to 20.75% respectively. Decreasing operating costs below the base case by 10% for the first five years of the operation, while allowing them to increase by 10% in the last five years, increases the ROI by 5% to 19.5%. As discussed in section 8.2.1, it would be better to lose 20% of the ore than to incur a 10% increase in operating costs to ensure its extraction.

Cost control has been changed with the introduction of bulk methods, however, because of increased difficulty in assigning costs to individual stopes. The provision of general services has become a larger proportion of the overall cost and becomes spread over production from various areas. For example, an LHD unit may muck from several stopes on multiple sub-levels in one shift. If there is a grade variation between individual stopes, the services are charged on a per ton basis as opposed to a unit of product basis. This complicates applying cut-off grades on an individual stope basis. There is a recent move to Activity Based Costing (80) in the mining industry to provide management better cost control in making such decisions.

8.4.2 A Discussion of Examples

Bulk mining practice has provided significant cost reductions in underground mining operations. There has been a major increase in both personnel productivity and in the productive capacity of stoping areas. The entire development of VCR techniques at the Levack Mine was driven by the requirement to provide a cost efficient alternative to undercut and fill mining to recover pillars between unconsolidated fill.

Two good examples of this increased productivity were provided by Falconbridge operations. As discussed in section 3.1.2, undercut and fill operations as described by Tims (33) were required to mine under a major flat-lying fault at the Falconbridge Mine. A similar fault at the Frazer Mine, as shown in Figure 4.4, caused little delay in advancing the 1-5-9 sequence described in section 4.2. In the same section, the sequence described by MacMillan and Ferguson (30) required that primary cut and fill stopes be advanced to the level above before secondary undercut and fill stopes could advance downward. All stopes would have required consolidated backfill and ground support would have been required on each lift. The 1-5-9 sequence provides a more productive and much less costly alternative.

A continuing cost reduction program is necessary if underground mining operations in Canada are to continue to compete in world markets. Regardless of whether bulk mining sequences meet all of the required objectives, they are going to continue to be used. A better activity based control of service costs will be essential to fully evaluate new cost saving methods.

8.5. MINIMIZING THE INITIAL DEVELOPMENT

Minimizing the capital cost and time required to reach revenue generation greatly increases the profitability of a new mine. In an underground mine, the time and cost include both vertical and lateral development. The initial vertical development is often reduced by using a staged development process. Lateral development can be minimized by mining ore closer to the shaft first or by choosing a mining sequence which allows early removal of secondary stopes.

In orebodies with variations in thickness or grade, this may include a strategy of developing the thickest or highest grade ore first, as the lateral development per tonne or per contained revenue unit would be minimized.

8.5.1 A Financial Discussion

In examining the profitability of the base case, a 10% decrease in capital costs to $135,000,000 increases the ROI of the project by 12.9% to 21%. Again it can be generalized that shorter life projects would be more sensitive to capital cost reductions than longer life projects. In an underground operation, however, reduced capital expenditures are not without potential problems. The capital cost of underground workings is very shaft and development opening intensive, the first being an area where reliability is more critical than price and the second is a function of the orebody geometry.

More typically, the timing of the capital expenditures can be controlled more successfully, the final depth of the shaft being delayed until a second stage of development for example. In the example, spending only $50,000,000 in each of the first two years and delaying $25,000,000 to years three and four increases the ROI by almost 20% to 22.3%. This is a very optimistic expenditure schedule, of course, which shows only moderate sensitivity. For a company with a cash shortage, however, the early establishment of positive cash flows while minimizing capital expenditures may have importance beyond just ROI implications.

Again, a staged development program is not without its pitfalls. A staged shaft deepening project presents conflicts between production and operating crews which can increase operating costs and cause difficulty in meeting hoisting schedules, particularly for development waste. The mine may be required to follow a two stage mining sequence giving rise to ore losses. Finally if adequate development is not available the mine may be forced to take short-cuts to meet production schedules which may be costly in the long term.

8.5.2 A Discussion of Examples

Many of the examples given in this thesis come from early established operations where a progressive expansion of productive capacity and the associated capital expenditures were the norm. Much of the ongoing expansion and capital expenditure was internally financed from the initial operation.

As examples, mining in the Campbell Mine F and A zones, as described in section 3.1.3, began near surface and progressed downwards to the 20 level. The initial 350 t per day production expanded gradually to 1150 t per day. Development was kept two to three levels ahead of production and ore was defined by a drift above, a drift below, and a raise through it as shown in Figure 3.8. The shaft sinking and deepening schedule from the Falconbridge Mine, given in section 3.1.2, also demonstrates this progression downwards.

In these and other operations, the sequence was not critical as the extraction rates were slow and stability was maintained by in-stope pillars. At the Campbell Mine however, the failure to adapt the layout and the sequence with depth, to decrease the extraction ratio, led to the permanent loss of some high grade reserves.

The introduction of bulk mining sequences better allows the delay of development as the productive capacity of individual stoping areas has increased dramatically. In fact, certain sequences can lead to reduced overall development mainly through increased main level spacing, with only ore extraction occurring on sub-levels along the orebody. Reducing the initial development openings, however, requires an earlier interaction between secondary stopes and backfilled primary stopes. The potential for ground control problems early in the life of the operation increases the requirement for a flexible mine design which can be adjusted with experience.

This thesis provides two Hemlo examples of delaying development. As discussed in section 5.3, both the Golden Giant and David Bell Mines were producing gold well before final completion of the shaft and bottom levels. This was accomplished at the David Bell operation mainly by relying on the cut and fill stopes on levels 1 through 3 as shown in Figure 5.10, sacrificing the ore in the sill pillars of the three stopes. There was an additional delay, however, of the ore between 9 and 8 levels which will have to be mined below the main ore zone as discussed in detail in Chapter 6. Whereas the 7 level sill mat underwent detailed design, no such design carried out for the 8 level. While the rock mass classification, numerical analysis, and instrumentation described in this thesis; and the experience gained by the mining of the 8D sill, will assist in the removal of the 9C to 8 level sill, additional ore has been placed in jeopardy.

The delay at the Golden Giant operation was based on rapid extraction of the number 1 mining block. When that block experienced problems, other areas had to be opened up. The quarter claim not only provided an additional ore source, but the use of Avoca mining helped to overcome the problem of hoisting development waste at a time when competition between production personnel and shaft deepening contractors existed for use of the shaft. The problems of recovering the ore beneath this fill have already been discussed.

While bulk mining sequences allow a delay of development, the financial consequences of not properly planning or not achieving the planned goals are greater. Problems are likely to occur earlier in the mine life as the extraction ratios of localized areas increase rapidly. The increased capital investment demanded by the modern mine and the requirement to externally finance much of it demands that the objectives of the chosen sequence be well defined and that the sequence be adequately flexible to meet unforeseen problems.

8.6 GRADE CONTROL STRATEGY

The grade control strategy may be to stabilize the grade over the mine's life or to extract a higher grade initially. Most certainly, short term grade fluctuations should be minimized as mill recovery losses can have serious revenue implications. In poly-metallic orebodies or those containing undesirable elements, this strategy may also include stabilizing the blend of the various metals into the mill so that all may be concentrated to levels acceptable to today's environmentally conscious smelters.

8.6.1 A Financial Discussion

Mine profitability is most sensitive to variations in revenue whether they stem from variation in grade, plant recovery or product price. In the case study, a 10% grade increase, accompanied by a 10% loss in ore volume, produces a 15.6% increase in the ROI to 21.5%. This would be difficult to achieve as the grade of the remaining material would be only 1.28 grams (0.04 ounces) per tonne, in effect this would be sorting out contained waste.

Interesting, however, is that a 10% increase in grade accompanied by a 20% loss in ore volume also increases the ROI, by 10.75% to 20.6%. What is startling in this case is that the grade of the material now left behind is 7.68 grams (0.24 ounces) per tonne. This is material above the cut off grade of 5.75 grams (0.18 ounces) based on the overall mining cost of $71.88 per ton used in this example. This is material internal to the orebody which could be left in regional pillars for ground support to possibly also decrease the mining costs.

The same ROI, 20.6%, would be achieved by recovering the entire orebody at a variable grade, increasing the grade 10% in the first five years and decreasing it by 10% during the last five years. The costs associated with such a strategy would be entirely dependant on the orebody geometry.

Another aspect of grade control is dilution prevention. In the case study, dilution of 10% reduces the ROI of the 11 year operation by 22% to 14.4%. Dilution of 20% reduces it by 42% to 10.7%. Dilution can stem from the failure of the host material, from backfill, or from dilution caused by the bulk mining system. This latter dilution of the reserve might, in fact, not be quoted as it would be built into the initial minable orebody.

8.6.2 A Discussion of Examples

The move to bulk mining methods has reduced grade control capability. As the production from a bulk stope is much greater, the number of required working areas has been reduced. With no selectivity in a bulk stope, if the grade fluctuates at the drawpoint, that fluctuation may find its way to the mill. This is more of a problem in smaller mining operations where one or two stopes may supply the required production.

The bulk mining sequence itself offers little grade control capability. The pyramid sequence as developed at the Golden Giant operation, described in section 5.3.3, is based on beginning the sequence at the bottom of the ore block and expanding outwards and upwards in a continuous manner. To leave behind low grade or waste material would require re-establishing the sequence on the far side of the inclusion. Leaving the pillar of waste behind would, in fact, destroy the ground control philosophy of establishing the sequence in the first instance. The same could be said of the one pass sequences described at Kidd Creek or Creighton Mines.

The other question is whether the use of bulk methods and sequences contribute to increased dilution. O'Flaherty (37), in describing the conversion of the A zone at the Campbell Mine to longhole methods, states that the potential exists for reduction of dilution. This is because longhole stopes can be mined at 1.2 metres wide whereas cut and fill stopes require a minimum of 1.8 metres to support personnel and equipment access. He also reports, however, that inappropriate blasting practices led to dilution of up to 100% in one stope. Dilution can also occur from backfill in cut and fill stopes, whether it be the unconsolidated rock fill previously used at the Macassa Mine or, more commonly, lightly consolidated hydraulic fill, when slushers or other mucking equipment dig below the ore fill interface.

In examining the methods in isolation, if irregular ore outlines exist waste within the stope boundary will be taken with the ore. The justification is that the reduced cost or cycle time compensates for the dilution. If blasting is not closely controlled, serious dilution can result. This dilution can come from the host rock failing into the stope from generally unsupported stope walls, or from backfill. Backfill dilution can occur in two ways. If overbreak reduces the size of the secondary stope the backfill now contained within the original volume is often designed into the system. In addition, failure of free standing fill faces can introduce high levels of dilution.

The sequence itself can also affect the level of dilution, particularly backfill dilution. In a one pass sequence all stopes except the first have at least one backfill wall. In the sawtooth sequence practised at the David Bell Mine 40% of the ore was taken as primary stopes with 60% being exposed to two backfill walls. In the 1-5-9 sequence, 50% of the ore is extracted before a fill wall is exposed but the remaining 50% is exposed to two fill walls. Unless primary, secondary and tertiary extraction are carried out simultaneously, the dilution at the tertiary stage can be significant.

So in this most economically sensitive objective of sequence planning, the use of bulk methods and sequences has reduced the capacity to control the grade and increased the potential for dilution. This points out again the requirement for good activity based accounting systems to allow more soundly based management decisions on the true economics of certain mining activities.

8.7 MINIMIZING GROUND CONTROL PROBLEMS

One of the initial justifications of this research was to analyze and assess how the switch to bulk mining sequences for underground ore extraction had affected personnel safety. The safety implications of sequencing fall under the objective of ground control. Consequently, discussion of this objective has been treated differently from the previous five because, in fact, achieving this objective affects the achievement of the others. The ability to control ground conditions affects the rate and level of extraction recovery, the operating and development costs, and the capability to control dilution. The inability to control the ground affects the safety of personnel and effectively causes the cessation of mining operations.

Bawden (17) discusses the implications of geomechanical design on the bottom line of a mining operation. He states "inadequate attention to the design of stope dimensions, support, etc., can quickly eliminate profitability from high productivity bulk mining methods". He lists the most critical costs as being dilution, support costs, blast damage and safety. In describing his perception of the relationship between the mine operator and designer, he states:

"Unless the characteristics of the rock mass have been seriously misjudged during the feasibility stage or the wrong mining method chosen, the first few years of mining are generally reasonable and the owners can enjoy a return on their investment without having to worry about underground stability problems and rockbursts."

He then continues:

"The next stage of the process occurs when the mine has reached maturity and when extraction ratios are high, pillars are being recovered and rockburst occurrences are relatively frequent."

While these latter comments may have been applicable to the mining activity as described at the Campbell Mine, the same author in a separate publication (19), describes how serious ground control problems developed in the initial mining block at the Golden Giant Mine. He further describes the failure of the initial pillar of the block 3 pyramid sequence at the same mine. While it is usual that primary stope extraction will not incur ground control problems, in many of today's bulk mining sequences the requirement to extract secondary or tertiary stopes may come early in the life of the operation. In the case study presented in Chapters 5 through 7 of this thesis, a major area of the mine advanced to full extraction in just over 5 years.

There is no evidence to indicate that bulk mining sequences have increased ground control problems as they apply to personnel safety. In fact, the evidence is to the contrary. O'Flaherty (37) states that the danger of being exposed to the ever decreasing pillars common in cut and fill stopes was one reason for switching to longhole methods at the Campbell Mine. Adapting the mining method at the David Bell Mine, described in Chapter 7, to handle the problems of ground falls in stope overcuts removed the requirement for personnel to rehabilitate the area, in fact it eliminated the requirement for the overcut. The use of the 1-5-9 sequence to mine through the flat fault at the Frazer Mine allowed the sequence to skip the overcut in the area of the fault thus avoiding any personnel exposure to the failed ground. Personnel operate in vertically spaced openings in which the ground has been conditioned and are not constantly exposed to new ground as in cut and fill stopes.

The earlier complete extraction of certain areas of the mine places more responsibility on the designer. The fact that problems can arrive earlier in the mine life means that the financial returns of the project are placed at increased risk. This places more pressure on the mine operators to provide a solution which may have negative long term implications. Part of the problem, however, is the perceived separation of the responsibilities of the mine designer and the mine operator. The design must be flexible and both engineers and operators must work closely together to achieve the objectives.

There is often too narrow a definition of ground control. It includes the support installed by the development crews, the care taken with blasting design and operational implementation, the ability of backfill crews to accomplish a timely filling of empty stopes, and many of the other operational activities critical to the mining process. The operators can be greatly aided by specialists who provide advice through numerical modelling and instrumentation. Results of the research program described in Chapters 6 and 7 were of benefit because they were presented to the operators by two senior ground control engineers who had operational experience. The author's numerical modelling results were interpreted by them in practical terms of how the short term plan might be affected by the indicated problem areas.

8.8 OPTIMIZING THE MINING SEQUENCE

There is no mining sequence which will provide the optimum solution for all orebodies. In the previous sections, the sensitivity of a project's economic outcome to various sequencing objectives was described and the results are summarized in Table 8.2. The most sensitive area involves grade variation or reduction, particularly early in the life of the operation. That may occur from grade variation inherent to the orebody or from diluting its initial quality with host or backfill material.

While there is little evidence of grade control affecting the mining sequences chosen in the Hemlo area, there is a wide variation in grade across

TABLE 8.2
THE ROI SENSITIVITY OF VARIATION FROM SEQUENCING OBJECTIVES

Parameter	Percentage Variation	ROI Variation
Resource Recovery	- 20%	- 10.9%
Recovery and Operating Cost	- 20% Recovery - 10% Cost	Unchanged
Mining Rate and Capital Cost	+10% Rate, + 10% Capital +10% Rate, Same Capital	- 5.0% + 7.0%
Operating Cost	+10% - 10% - 10% 5 years, +10% 5 years	- 11.8% +11.5% + 5.0%
Capital Cost	- 10% - 33.3% for 2 years	+12.9% +20%
Grade and Recovery	+10%, 90% of ore +10%, 80% of ore +10% 5 years, -10% 5 years	+15.6% +10.75% +10.75%
Dilution	10%, 11 year life 20%, 12 year life	- 22% - 42%

the total orebody. The main ore zone sequence at the David Bell Mine began in the highest grade material yet that was also the lowest accessible level at the time and to mine next to the Golden Giant Mine boundary made more sense from a geotechnical point of view. Grade control does not seem to be a priority at the various Hemlo mines, although the presence of variable amounts of barite and some refractory ore has caused minor problems. The fact that the orebody has been divided into three operations has reduced the grade variations at the individual properties. As discussed in Chapter 5, had one operation been established along the entire orebody the situation would have been very different. The one order of magnitude grade variation from west to east would have demanded separate mining sequences.

The main emphasis in choosing the mining sequence has been control of the increased stresses inherent at depth. Kidd Creek continued to use bulk methods at depth rather than switch to mechanized cut and fill mainly because of the success of the one pass sequence in controlling ground problems. Mining at Creighton below the 7000 level is only possible if the high stresses can be controlled by the one pass sequence used there. Mining should not return to traditional methods since both safety and operating costs have been improved by the newer sequences.

Operators can not, however, ignore the sensitivity of the economic outcome to grade control and dilution. This requires a refinement of the existing practices to pay more attention to ground support and backfill stability. The continued emphasis, however, remains on productivity and automation. A re-examination of the objectives is necessary in conjunction with the ongoing research in these areas if truly optimal solutions are to be found.

8.9 CLASSIFICATION OF MINING SEQUENCES

In classifying mining sequences, it is difficult to provide definitive classes because of the overlap of factors which occurs. Morrison (6), in classifying mining methods stated, "It is impossible for any classification to embrace all the differences in practice flowing from the possible variations in physical conditions." A classification of the sequences described in chapters 3 through 5 is presented in Table 8.3, based on the interaction between primary stopes and pillars. In some cases pillars remain to carry the regional stress load, in some they are allowed to fail and in some they are totally eliminated.

Classifications such as ascending or descending could be considered, especially as more experience is gained in operating under backfill. However, the control of strain energy, as originally offered by Morrison (6) to classify mining methods, seems to be of prime importance. With traditional mining practice in narrow high modulus orebodies, this strain energy was carried by small intact pillars which have proven both costly and unsafe to recover. The suggested improvements are to use larger regional pillars or to destress the pillars. The use of longhole mining as described at the Campbell Mine (37), while still being based on a "stiff" approach, allows the final horizontal stope length slice to be up to 15 metres thick and removed in one mass blast. The use of regional pillars in low grade material is also suggested.

Table 8.3 A CLASSIFICATION OF MINING SEQUENCES

1. ORE THICKNESS 1 TO 5 METRES

1. A Stiff Sequence. The modulus of the ore would likely be above 35,000 MPa. Primary and secondary stopes are not required as the ground is not allowed to yield. Stopes can be started consecutively along strike, restricted only by interference between stopes.

1. Stiffness maintained by stope pillars
   Mining methods: Boxhole shrinkage, cut and fill

2. Stiffness maintained by regional pillars
   Mining methods: Boxhole shrinkage methods, cut and fill mining, longhole - reduced sublevel spacing

3. Pillars destressed to allow closure
   Methods are as in 1 and 2 but individual pillars may have to be larger to carry the stress transferred from destressed pillars.

2. A Yielding Sequence . The modulus of the ore would be below 35,000 MPa. Primary and secondary stopes are sequenced along strike and pillars must be of a size to ensure failure. Different mining methods are required in the failed pillars.

1. Primary Stopes
   Mining methods: Boxhole shrinkage methods, cut and fill mining, longhole - reduced sublevel spacing

2. Secondary Stopes
   Mining methods: Square set, undercut and fill mining, longhole - underhand advance or drilled from off ore drifts

3. Pillar Elimination Sequences. The modulus of the ore would affect only the size of initial stope pillars. Stopes are mined sequentially on strike and vertically, stope pillars are removed on stope completion to shed the stress to the abutment.

1. Primary Stoping
   Mining Methods: Rill mining, cut and fill, longhole retreat

2. Pillar Removal, by vertical slices or by mass blasting

Table 8.3 A CLASSIFICATION OF MINING SEQUENCES (cont.)

---

2. ORE THICKNESS 5 METRES PLUS - The thickness must justify the access development. The in-situ ore modulus is expected to be lower due to fractures in the rockmass and affects only the sizing and the timing of the removal of secondary or tertiary pillars.

1. Single Pass Sequences. These sequences extract the orebody in one pass leaving no recoverable pillars, either as one stope or as a series immediately adjoining stopes.

1. Mechanized cut and fill methods
   Mining Methods: Post pillar cut and fill, ramp cut and fill

2. Pattern expanding in plan - one level
   Mining Methods: Blasthole mining, VRM

3. Pattern expanding in plan - multilevel
   Mining Methods: Blasthole, VRM

4. Pattern expanding in longitudinal section, the pyramid sequence
   Mining Methods: Blasthole, VRM

2. Multi-pass Sequences.

1. In longitudinal section, a secondary (1-3-5) or a tertiary sequence (1-5-9),
   Mining Methods: Blasthole, VRM

2. In plan, a secondary (1-3-5) or a tertiary sequence (1-5-9)
   Mining Methods: Blasthole, VRM

3. A Retreating Front Sequence.

1. A sawtooth retreat sequence
   Mining Methods: Blasthole, VRM

In operations with narrow lower modulus ore, pillars were allowed to yield in a non-violent manner. This requires both consolidated backfill and more expensive underhanded stoping methods to recover the pillars. The recent use of longhole methods has reduced the cost of recovering these pillars. Pillar elimination methods have really only been practised in the high modulus rocks at the Lakeshore Mine. The difficulty in placing backfill without pillars to contain it has been part of the problem. This was overcome at the Lakeshore Mine by rill mining, a difficult and labour intensive approach.

In thicker ore, a stiff approach is less likely as the thicker the rock mass the higher the likelihood of a reduced in-situ modulus. Pillar elimination methods become easier to manage with longhole bulk methods as the use of delayed backfill in larger voids means the interface with backfill occurs less often although the exposure area is greatly increased. While post pillar cut and fill still used a pillar for local support, it was designed to fail after being confined in fill and was not recoverable. The other one pass methods, the single level pattern at Creighton deep, the three level pattern used at Kidd Creek or the longitudinal pattern at Golden Giant all eliminate pillars completely and allow rapid closure.

The key in multi-pass sequences is controlled failure of the secondary pillars. The 1-5-9 sequence retains intact pillars until 50% of the ore is extracted allowing the tertiary stopes to yield. The same ratio is true for a 1-3-5 sequence but fewer initial working locations are available. As described at the Lockerby Mine in section 4.2, problems may develop if the yielding ground is allowed to lag too far behind the primary stopes.

In the advancing front sequences, as practised at the David Bell and Williams Mines, the secondaries can be allowed to fail or be maintained intact. This would depend on the nature of the ground and on how far ahead of the secondaries the primary stopes were allowed to advance. This offers many of the advantages of one pass sequences but provides additional working locations and removes the requirement to consolidate the total backfill mass.

Another differentiation between traditional and modern sequences is considered important. In traditional mining methods the ore was removed in slices which were 3 to 4 metres high by the entire stope length, up to 60 metres. In-stope lateral movement of the ore was required in cut and fill stopes. This was necessary because personnel required access to each advancing portion of that slice for drilling and blasting. Morrison recognized the advantages of a more vertical component to the removal geometry, in fact a closer balance between the vertical and horizontal dimensions of the stope geometry. This was achieved at the Lakeshore Mine by the tedious rill mining system, a system also proposed for Macassa but never used. It was tried at the Campbell Mine by shortening the length of the cut and fill stopes but reduced productivity saw it eliminated. It was introduced at Macassa in recovering the sill pillars at the top of cut and fill stopes.

With the advent of longhole methods, a greater vertical to horizontal ratio was possible. In narrow ore, a 60 metre high stope can be removed in three 20 metre slices, the last being removed in one mass blast. The newer bulk mining sequences use individual stopes from 15 to 30 metres on strike, similar dimensions across the orebody, and from 25 to 66 metres in height. The main limitations in height are the regularity of the ore outline, the stability of the stope walls, the stability of the free standing fill mass, and the accuracy of the drilling. Gravity is used for all in-stope ore movement and remote controlled equipment to remove it from drawpoints.

This geometry eliminates the stability problems inherent in extended flat mine openings. Reduced sub-level spacings allow secondary stopes to be mined shortly after primary stopes. The geometry is possible by distributing the energy required to break the ore into long blastholes drilled from widely spaced sub-levels. The significance of this is discussed further in the next section.

8.10 COMMENTS ON THE FUTURE OF MINING SEQUENCES

In examining mine sequencing objectives, it was apparent that the ability to achieve the financially sensitive objective of grade control has been reduced by both bulk mining methods and their associated removal sequences. Other objectives have been greatly enhanced, however, so the methods are here to stay. Two items should be considered to alleviate the grade problem.

Firstly, more accurate assigning of the increased service costs must be available to better assess the marginal cut-off grades. This may be done through activity based costing. As an example, the cost of disposing of large volumes of salt tailings at Central Canada Potash has brought about a redefinition of the true cut-off grade of the potash ore. While the total tailings volume will not be critical in a gold mine, the cost per ounce of material movement or of the leaching acid consumption would be of more interest.

The second possibility is to provide primary separation of the waste or backfill before hoisting. To date, most pre-concentration has been based on size differential or magnetic properties of the rocks. The author was involved in designing an underground pre-concentration facility for a Botswana nickel mine based on permaroll magnets. The capability exists to upgrade ore based on its optical or fracture characteristics but these techniques are unproven and generally require large facilities. The economic advantages of upgrading ore underground will increase as the mining depth increases and if improved fragmentation or continuous mining machines provide reduced ore size.

The grade control problem may be worsened by introduction of both continuous and automated mining systems. Even in bulk stopes, a measure of selectivity can be achieved from blasthole drill logs along the ore waste contact. Geologists provide some grade control from drawpoint sampling.

These activities will become more difficult as automation increases and intensifies the requirement to carry out some upgrading underground.

Just as the introduction of the in-the-hole drill served as the basis of many of the bulk sequences described in this thesis, automated or continuous mining machines will have a significant impact on the mining sequences of the future. A machine which continuously breaks rock requires access to every individual piece of the rockmass. As in hand held or carrier mounted drills, the machines which are being planned will require horizontal slices to achieve this access. The horizontal slices may have to be even longer to provide the productivity to justify the high capital costs of the equipment. This may require a complete revision of mining sequences, to ones which have proven to be detrimental to good ground control. A re-examination of the objectives is necessary in conjunction with this ongoing research into new mining methods.

8.11 CONCLUSION

Bulk mining sequences have enhanced the mine designer's ability to improve resource recovery, the mining rate, and greatly improved the unit cost of producing a tonne of ore. While mine development can be staged more easily using these sequences, the development costs of today's mines are high and the more rapid increase in the percentage extraction of individual mining areas can lead to earlier ground control problems. Any problem which interferes with sustaining the initial mining rate increases the economic risk of the project.

The safety of today's mines has been improved by the introduction of bulk mining. The reduced underground workforce is confined to conditioned access drifts away from the blasted ore in the stope. Many sequences have been designed to better manage stress related problems. This is not surprising as it is necessitated by the high extraction rates achieved early in a mine's life. The most logical classification of the mining sequences observed at the various mining localities is based on this control of the release of strain energy.

One major economic objective of mining, stabilizing the grade of ore to the mill, has been adversely affected by the bulk mining sequences. This is due to the reduced ability to mine selectively and the increased dilution which often comes from bulk methods. The effect of this may be compensated for in better calculating cut-off grades, in imposing more discipline and sound management during mining operations, and in introducing new underground ore sorting practices. Finally the effects of automation and continuous mining systems on the total economics, not just on productivity, and on the required changes to mining sequences should be evaluated before such systems are introduced.

9.0 CONCLUSIONS AND RECOMMENDATIONS

This research was carried out to aid in the understanding of how to best manage the underground extraction of a tabular hardrock orebody and has concentrated on an analysis of mining sequences used in steeply dipping deposits in Ontario.

To properly manage an extraction sequence, a set of objectives must be defined and the optimal strategy to meet them chosen. Factors which have influenced the success or failure of other operations must be understood and the mine plan developed using the best available information and modern planning tools. Finally, the ongoing operation must be monitored and methods or sequences adapted to ensure the success of the plan.

9.1 CONCLUSIONS

In comparing the factors influencing sequence planning in the various Ontario mining areas, the most significant technological change has been the introduction of bulk mining methods leading to completely new extraction sequences. From a design standpoint, the initial stress levels were predictable within the accuracy required for initial mine design. The most critical geotechnical factor differentiating the mine extraction sequences used in the various areas was the rockburst potential of the high elastic modulus rocks present in the Kirkland Lake and Red Lake areas.

The traditional mining practices in narrow vein shrinkage or cut and fill mining were carried out without much regard for optimization. Ore was not delineated much in advance of mining and the rate of development and stope depletion was slow. With low productive capacity per stope, many working locations on several levels were required. A mining operation could operate for several years before rockbursting or the high cost of secondary stope removal became a problem. Mining started near surface and progressed downwards and there often was a gradual expansion of mining capacity.

In the mining areas with high modulus or brittle rocks the extraction sequence was managed in three ways. Mine stiffness was maintained by small pillars above and below the level to allow flexible and low cost ore extraction but creating rockburst prone areas. The pillars were sometimes destressed before removal but this was slow and expensive and the stress shed to other pillars often initiated additional damage. Finally, one pass mining sequences were developed to eliminate pillars but these again were low productivity methods using conventional drill and blast techniques.

In mines with lower modulus rocks, the creation of yielding pillars was critical to managing the rock stresses during the extraction sequence. This latter practice is similar to the sequences used today and, as mechanisation increased, the modern sequences developed from them. This was done,in part, to take advantage of the increased productivity available from new drilling and materials handling equipment.

Increased mechanization led to the development of post-pillar and ramp cut and fill mining which provided impressive productivity increases with some sacrifice of total recovery. The introduction of bulk mining techniques with delayed fill allowed more flexible mining sequences, further increases in productive capacity, reduced costs and better ground control.

The changes in ground control practices were possible because of a total change in stope geometry. The vertical dimension of the individual stopes became at least equal to or exceeded the maximum horizontal stope dimension. The use of long blastholes, drilled from one pre-conditioned opening, allowed more material to be broken in the drill, blast, muck, fill cycle. Gravity and remote controlled mucking equipment eliminated the requirement for personnel to enter the stope after blasting. This geometry also eliminated the large horizontal sills of previous sequences which had proven difficult to recover and which often gave rise to rockbursts.

These sequences require more advanced planning as localized high extraction, with possible ground control problems and dilution, can be effected early in the life of an operation. Increased costs or reduced production early in the life of an operation can seriously affect the project's economic viability.

The case study of the Hemlo area orebody, demonstrates the increased complexity of sequencing planning as three mines interact along common boundaries. The David Bell Mine provided an excellent opportunity to plan, observe and monitor a mining sequence to complete extraction with the removal of the 8D sill pillar. While a lack of advance development sometimes affected detailed stope sequencing, the initial planned sequence was followed relatively closely and the ore was extracted in an efficient manner. The only quantifiable cost of the approach adopted was the requirement of additional hangingwall development and the dilution caused while extracting the 8D pillar.

The case study demonstrates that numerical modelling is useful in planning extraction sequences but with some limitations. The structural discontinuities in the orebody, the baritic or sericitic zones for example, could not be introduced into the MINTAB models; nor can they be introduced into most three dimensional models, Examine3D for example. While the behaviour of these zones can be examined using FLAC, their implications on a mining sequence can not be modelled.

When the numerical modelling results were used in conjunction with a mine instrumentation program, however, the results proved very useful. Knowledge of the failed state of the rock mass in the 8D pillar allowed modifications to the planned extraction. In conjunction with a very flexible mining method developed by operating personnel, the ground control problems were handled in a safe and cost effective manner.

The introduction of new bulk sequences has increased the chances of achieving most of the objectives of a mining project, especially the important objective of improved personnel safety. These sequencing practices will become more important as the depth of mining continues to increase. The research began at a time when tragedy had been caused by serious rockbursts. The problems of rockbursting have been reduced but not eliminated, however, as at the time of writing two miners are missing at the Macassa mine as a result of a rockburst on November 26, 1993 (81).

However, grade control and dilution, very sensitive factors in project feasibility, have been adversely affected by these bulk methods. A classification of the mining sequences described in the early chapters, shows that the control of the release of strain energy, as used by Morrison to classify mining methods, remains a viable criterion to differentiate extraction practices.

In looking to the future, however, improved methods to assign the increased costs of overall mine services to individual mining areas will be required to better assess the economics of changes in mine sequencing practices. The potential for such changes is great as the use of automation and continuous mining systems increase. It is also important that these systems be developed in such a way that they do not preclude the use of the improved sequences which have evolved during the recent past.

9.2 FUTURE WORK

The main recommendations for future work arising from this thesis are as follows:

1. **The effects of new technology on mining sequences must be assessed.**

A major trend identified while classifying the past and present mining sequences was a move to a more vertically oriented sequence as opposed to the flatter mining areas required to maintain the productivity of shrinkage and cut and fill stopes. New technology being developed must be examined in the light of how it will affect the mining sequences of the future. Research is ongoing into the development of continuous extraction machines to replace the drill blast cycle and into the use of plasma blasting techniques. Both of these technologies, delivered through a large capital intensive machine, may demand a sequence that requires a long flat slice of ore to be removed. In fact, maintaining the productivity of such a machine will require even longer stopes so that backfill and ground support cycles will not interfere with the machine's operation.

As an alternative, if plasma blasting could be delivered through a vertical platform, along a borehole for example, much more vertical sequences could be developed. Cost benefits from ground control and materials handling could be significant. These technological changes should be evaluated in an integrated manner so that they can be adequately assessed.

2. Development of 3D computer graphics tools.

Systems should be developed in 3D computer graphics which allow empirical mine data to be displayed in relation to the structural data. Certain stress cell measurements at the David Bell Mine were erratic, likely because of a structural feature separating the cell from the advancing stress front. Three dimensional displays would make interpretation simpler and allow easier comparison to output from numerical models.

3. Continued Development of 3D Numerical Models

The continued development of 3D numerical models should allow the introduction of structural discontinuities, work begun by Wiles (82) and Fotoohi

(83). While the capability of modelling varied rockmass behaviour is important, there is some question as to the detail that this capability can be introduced into 3D models that are large enough to model mining sequences. It is important that there be a stepping capability to allow the analysis of alternate sequences without detailed comparison of the results from two models. Perhaps the answer to both of these problems lies in having output capability to three dimensional solids models which could serve as input into more detailed modelling of specific areas of the mine or allow the comparison of output parameters. These solids models would also serve as the point of integration of the overall planning system as outlined in recommendation 5.

4. Instrumentation programs are required.

More and better instrumentation is required to further evaluate the numerical models. As discussed in section 6.4, numerical models operate in an iterative or a stepped nature and there is no correlation of timing in the models to timing in the real world. This is important, for example, if the effects of backfill on limiting mine closure or the stability of backfill in top-down mining sequences is to be assessed.

Also, instrumentation would better correlate the boundary conditions imposed on models to the far-field stress conditions in a mine. Model boundaries can be fixed or allowed to move as internal closure occurs, with quite different predictive results. At some distance from an orebody, closure ceases and stresses redistribute. While having a large boundary area around a model may reduce this effect, an understanding of these far-field reactions could greatly improve the formulation of numerical models.

5. **Development of an integrated planning system for underground mines.**

Systems are available to carry out geostatistical analysis of orebodies although some problems still exist with tabular deposits. Geomechanical design is advancing rapidly, although as noted above the display of empirical data could be enhanced. Mine openings can be displayed in various computer aided design (CAD) packages but the systems do not provide the qualitative and quantitative data necessary to carry out economic analysis. Integration of these systems with an activity based cost system is required so that the optimality of a design can be judged through the rapid evaluation of the alternate extraction strategies.

9.3 CLOSING COMMENTS

A complete geomechanical design has three phases (17), (58). The first phase involves collecting all available geomechanics data including rock strength and the stress environment. The second phase involves choosing the method of extraction and evaluating the potential effects on the rock mass through the use of stability analysis and stress modelling. This should be carried out keeping operational constraints in mind. The third phase closes the design loop through monitoring the rock mass by visual observations or through instrumentation programs. This allows the initial design to be enhanced and the design tools to be calibrated.

That design rationale has been documented in Chapters 6 and 7 of this thesis. In addition, however, the thesis goes beyond this single example to examine how varying geomechanical parameters have affected operating practice at various operations, mainly in Ontario. The extraction methods and sequences used at these various operations are described, analyzed and classified. The practices are compared not only to the production constraints of a mine but to the parameters affecting project profitability. The response of the rock mass has been documented from site visits to many of these operations and from the analysis of publications describing the rock behaviour.

It is hoped that this analysis of the design process will assist in the production of safer and more productive mine extraction sequences in the future.

REFERENCES

1. Koniaris, E.A., **A Positive Approach to Long Term Planning in Open Pit Mines**, Ph.D. Thesis, Queen's University at Kingston, August 1991, pp 7-37.

2. Makuch, T., **Optimization of Hoisting Productivity and Design of Shaft Access and Mine Production for Ore Below 4000' at Placer Dome Inc. Campbell Mine**, Master's Thesis, Queen's University at Kingston, 1994, pp 66-70.

3. Lizotte, Y., **Blasthole stoping for narrow vein mining**, CIM Bulletin, Volume 84, No. 945, January 1991, pp. 35-41.

4. Robertson, B.E., **Mechanized narrow vein mining at the Dome Mine, Timmins, Ontario**, CIM Bulletin, Volume 79, No. 885, January 1986, pp. 39-44.

5. Morrison, R.G.K., A Philosophy of Ground Control, Department of Mining and Metallurgical Engineering, McGill University, 1976, pp 125 - 130.

6. Hedley, D.G.F., **A Case History of Rockbursts at Elliot Lake**, September, 1983, CANMET Report MRL 84-16 (OP)(J), pp 1-31.

7. Cook, J.F., Bruce, D., **Rockburst control through destressing - a case example**, Rockbursts: prediction and control, pp. 81-91. The IMM, London, England, 1983.

8. Hedley, D.G.F., Neumann, M., Makuch, T., Blake, W., **Rockbursts at Campbell Red Lake Mine**, January 1985, CANMET Report MRL 85-31 (OP), .

9. Hedley, D.G.F., Bharti, S., West, D., and Blake, W., **Fault-Slip Rockbursts at the Falconbridge Mine**, September 1985, CANMET Report MRL 85-114 (OP).

10. **Improving Ground Stability and Mine Rescue, The Report of the Provincial Inquiry into Ground Control and Emergency Preparedness in Ontario Mines**, T. Stephenson, Chairman, 1986, pp 91.

11. Section 5a, Ontario Regulation 694 under the Occupational Health and Safety Act, R.S.O. 1980, chapter 321, as amended, pp R11 - R12.

12. FLAC User's Manual, Itasca Consulting Group, Inc., Minneapolis, Minnesota 55414, pp 3-1 to 3-23, 1987.

13. Zambo, J., **Optimum Location of Mining Facilities.** Akademiai Kiado, Budapest, 1968, pp 145.

14. Gerling, R. and Helms, W., **A Model for Long Range Scheduling of the Mining Sequence for Flat Lying Orebodies.** Proceedings of the 19th APCOM Symposium, Pennsylvania State University, 1986, pp. 333-342.

15. Yi, R. and Strugul, J.R., **Analysis of Cutoff Grades Using Optimum Control Theory**, Proceedings of the 20th APCOM, Johannesbourg, Republic of South Africa, 1987, Volume 3, pp. 263-269.

16. Potvin, Y., Hudyma, M.R., and Miller, H.D.S., **NSERC Project #CRD-8612 Integrated Mine Design Project**, Volume 1, October 1988.

17. Bawden, W.F., **The Use of Rock Mechanics Principles in Canadian Underground Hard Rock Mine Design**, Volume 5, Comprehensive Rock Engineering, E. Hoek, Editor, Pergamon Press, 1993, pp 247-290.

18. Curran, J.H., Corkum, B.T., ans Wyllie, J.A., EXAMINE^{TAB} Version 1.0, A users guide, September 1990, pp 45.

19. Bawden, W.F., Sauriol, G., Milne, D. and Germain, P., **Practical rock engineering stope design case histories from Noranda Minerais Inc.**, CIM Bulletin, Volume 82, No. 927, 1989, pp. 37-45.

20. **Report of the Special Committee on Mining Practices at Elliot Lake (1961)**, Morrison, R.G.K., Chairman, Ontario Ministry of Mines Bulletin 155.

21. Thomson, J.E., **Regional Structure of the Kirkland Lake - Larder Lake Area**, in Structural Geology of Canadian Ore Deposits, CIM Jubilee Volume, 1948, pp 627- 632.

22. Dunbar, W.R., **Structural Relations of the Porcupine Ore Deposits**, in Structural Geology of Canadian Ore Deposits, CIM Jubilee Volume, 1948, pp 442-456.

23. Morrison, R.G.K., **Report on the Rockburst Situation in Ontario Mines**, Transactions CIMM, Volume XLV, 1942, pp. 225-272.

24. Horwood, H., **General Structural Relations of Ore Deposits in the Red Lake Area**, in Structural Geology of Canadian Ore Deposits, CIM Jubilee Volume, 1948, pp 322-328.

25. Arjang, B. and Vaillancourt, R., **Field Stress Determinations at Macassa Mine**, CANMET Report MRL 85-63 (TR), pp 1-18.

26. Arjang, B., **Stress Determinations at Stanleigh Mine**, Cctober 1984 CANMET Report MRL 84-119 (TR), pp 1-18.

27. Arjang, B., **Review and Evaluation of Ground Stresses at Campbell Red Lake Mine, Ontario**. CANMET, Elliot Lake Laboratory, July 1987, MRL 87- (Tr), pp 1-28.

28. Makuch, T., **Rock Mechanics at Campbell Red Lake Mine: A Progress Report**. Internal report, August 11, 1987, pp 1-7.

29. Herget, G., **Regional Stresses in the Canadian Shield**. CANMET Report MRP/MRL 80-8 (OP), January, 1980, pp 1-24.

30. MacMillan, P.W. and Ferguson, B.A., **Principles of Stope Planning and Layout for Ground Control**, Underground Mining Methods Handbook, Hustrulid, AIME, 1982, pp 526-530.

31. Robson, W.T., **Rock-Burst Incidence, Research and Control Measures**, Transactions CIMM, Volume XLIX, 1946, pp. 347-374.

32. Weldon, L.S., **Mining Methods at Lakeshore**, Transactions CIMM, 1936, pp. 157-198.

33. Tims, S.A., **Undercut and Fill Mining at Falconbridge Mine of Falconbridge Nickel Mines**, Underground Mining Methods Handbook, Hustrulid, AIME, 1982, pp 650-659.

34. Morrison, R.G.K., **A Report on Rockburst Potentiality at Dickenson Mines Limited and Campbell Red Lake Mines Limited**, Internal Report, pp 1-25.

35. Campbell Red Lake Mines Limited, Mining Brief presented to Ground Control and Emergency Preparedness Committee, January 1985, pp 1-80.

36. Neumann, M., Makuch, T., Hedley, D. and Blake, W., **Practical Applications of Pillar Destressing at Campbell Red Lake Mine,** Proceedings of the CIM Eighth Underground Operators Conference--1987, Elliot Lake, February 23-25, 1987, pp 1-12.

37. O'Flaherty, M., Bourchier, F., **Application of Narrow Vein Sublevel Longhole Mining at Placer Dome Inc., Campbell Mine,** Presented at the 11th Underground Operators Conference, February 21, 1993, Saskatoon, Sasketchewan, pp 13-1- 13-9.

38. Arjang, B. and Nemcsok, G., **Review of rockburst incidents at the Macassa mine, Kirkland Lake.** CANMET Report MRL 87-21(TR), 1986.

39. LeBel, G.R., Quesnel, W. and Glover, W., **An analysis of rockburst events during sinking of Macassa no. 3 shaft.** Presented at 89th CIM Annual General Meeting, Toronto, 1987, pp 1-14.

40. Hanson, D., Quesnel, W.J.F. and Hong, R., **Destressing a rockburst prone crown pillar - Macassa mine.** CANMET division report MRL 87-82(TR), 1987.

41. Personal Communication, William Quesnel, Rock Mechanics Engineer, Lac Minerals Ltd., 1986.

42. Quesnel, W.J.F., de Ruiter, H. and Pervik, A., **The assessment of cemented rockfill for regional and local support in a rockburst environment, LAC Minerals Ltd., Macassa Division.** Innovations in Mining Backfill Technology, Hassani, F.P., Scoble, M.J., Yu, T.R., Editors, Balkema, Rotterdam, 1989, pp 217-224.

43. Oliver, P.H., **Ground Control in Transverse Cut and Fill Mining Operations at Inco,** Proceedings of the 10th Canadian Rock Mechanics Symposium, Queen's University, Kingston, September 1975, Volume 2, pp 1-27.

44. Golder Associates, **An Internal Report to Rio Algom Mines Limited on Stanleigh Mine Design,** 1983, pp 1-97.

45. Madsen, D., Moss, A., Salamondra, B. and Etienne, D., **Stope development for raise mining at the Namew Lake mine.** CIM Bulletin, Volume 84, No. 949, May, 1991, pp. 33-39.

46. Personal Communication, Robert Fisher, Production Planning Supervisor, Brunswick Mining and Smelting, 1988.

47. Hanson, D., **Lockerby Mine--A Case History**, Professional Development Seminar on Rock Mass Failures in Hard Rock Mining, Canadian Mining and Industrial Equipment Exhibition, Laurentian University, June 1-3, 1988.

48. McKay, D.L., and Duke, J.D., **Mining with Backfill at Kidd Creek No. 2 Mine**. International Symposium on Mining with Backfill, Lulea, Sweden, 1983, pp 12.

49. Ministry of the Solicitor General of Ontario, Office of Chief Coroner, Verdict of Coroner's Jury, Area 5741, Dr. T. Fischbacher, May 16, 1990, Explanation of Jury Recommendations, Paragraph 7.

50. Wittchen, V.C., Croxall, J.E. and Yu, T.R., **Roadheader Excavation in Consolidated Rockfill at Kidd Creek Mine**, CIM Bulletin, Volume 83, No. 937, May 1990, pp. 41-45.

51. Pelley, C.W., **Stope Redevelopment through Stabilized Backfill**, a contract report submitted to the Mining Research Directorate, June, 1990, pp 1-77.

52. Pelley, C.W., and Mitchell, R.J., **Physical and Numerical Modelling of Openings in Cemented Hydraulic Backfill**, Proceedings of the 11th Symposium on Engineering Applications of Mechanics, University of Regina, May 11-13, 1992, pp. 154-160.

53. Tentative agreement between Lac Minerals Ltd., Hemlo Division and Noranda Inc., Hemlo Division, February 6, 1985, pp 1-6.

54. Golder Associates, Report No. 871-1380, **NFold Modelling Page-Williams Mine, Ontario**. Report to LAC Minerals Limited, August 1987, pp 1-7.

55. Golder Associates, Report No. 872-1471, **Report on Rock Mechanics and Ground Control Aspects of Proposed Mining Adjacent to Property Boundary by Hemlo Gold and Teck-Corona Operating Company**. January, 1988, pp 1-13.

56. Hurley, P., **Mine Design at the Golden Giant Mine, Noranda Inc., Hemlo Division**. Paper presented at the CIM District Three Meeting, Timmins, October 3-5, 1985, pp 11.

57. Pieterse, K.R., **Production Lead Time Reduction, An Approach, Part 1**. Presented at the CIM Eighth Underground Operators Conference, Elliot Lake, February 23-25, 1987.

58. Grant, D.R., Potvin, Y., Rocque, P., **Three dimensional stress analysis techniques applied to mine design at Golden Giant Mine**, First Canadian Symposium on Numerical Modelling Applications in Mining and Geomechanics, Montreal, March, 1993, pp 5-15

59. Personal Communication--David Bronkhorst, Chief Engineer, Williams Mine, August 1991.

60. Golder Associates, Report No. 921-1727, **Comments on Sill Pillar Mining Development, Williams Mine, Marathon, Ontario**, Submitted to Williams Operating Corporation, April, 1993, pp 1-16.

61. Mitchell, R.J., Roettger, J.J., **Analysis and modelling of sill Pillars.** Innovations in Mining Backfill Technology, Hassani, F.P, Scoble, M.J., Yu, T.R., Editors, Balkema, Rotterdam, 1989, pp 53-61.

62. Yu, Y.S., Toews, N.A., and Wong, A.S., **MINTAB Users Guide - A Mining Simulation for Determining the Elastic Response of Strata Surrounding Tabular Mining Excavations (ASCII Version 4.0, 1982)**, CANMET Division Report MRP/MRL 83-25(TR), pp 41.

63. Kazakidis, V.N., **An Integrated Study of Fractured Rock Mass Behaviour Using Structural Mapping, Microseismic Monitoring and Numerical Modelling**, Master's Thesis, Queen's University at Kingston, July 1990, pp 14-61.

64. Golder Associates, Report No. 881-1731, **In-situ Stress Measurements at Paige Williams Mine 9240 Level**, Submitted to Williams Operating Corporation, December, 1988, pp 1-21.

65. Leduc, M., **Rock Mass Behaviour at the David Bell Mine, Hemlo, Ontario**, B.Sc. Thesis, Queen's University at Kingston, 1992, pp 1-63.

66. Course Notes, Mine 325, Rock Mechanics, Archibald, J.F., Department of Mining Engineering, Queen's University, 1994.

67. Smith, J.D. and Pelley, C., **Closure and Stress Distribution Numerical Modelling Study - David Bell 5 Year Production Plan**, Internal report prepared for Teck-Corona Operating Corp., June 15, 1988, pp 1-16.

68. Starfield, A.M., and Crouch, S.L., **Elastic Analysis of Single Seam Extraction**, in New Horizons in Rock Mechanics, Hardy, H.R., and Stefhanko, R., (Eds), ASCE, New York, pp 421-439.

69. Zhang, L., and Mitri, H.S., **3-D modelling of underground mine structures with backfill**, CIM Bulletin, Vol 85/No. 962, July/Aug.,1992, pp 89-95.

70. Sheridan, F. and Assabgui, R., **Numerical Modelling at Teck-Corona Operating Corporation David Bell Mine**, Internal Report, February 28, 1990, pp 20.

71. Pelley, C.W. and Archibald, J.F., **Numerical Modelling of Backfill Failure Response under Variable Strength Parameters**. Proceedings of the 11th Symposium on Engineering Applications of Mechanics, University of Regina, May 11-13, 1992, pp. 121-126.

72. Bos, E., **Barite and its Effect on Stability at the David Bell Mine**, B.Sc. Thesis, Queen's University at Kingston, 1993, pp 41.

73. Pakalnis, R., Cook, K., and Vongspaisal, S., **Sill Pillar Excavation of a Cut and Fill Operation**, First Canadian Symposium on Numerical Modelling Applications in Mining and Geomechanics, Montreal, March, 1993, pp 17-28.

74. Mitri, H.S., and Scoble, M.J., **A numerical procedure for stability analysis of hardrock mine structures**, Mining Science and Technology, 9 (1989), 187-195

75. Mitri, H.S., and Rajaie, H., **Stress Analysis of Rock Mass with Cable Support - A Finite Element Approach**, Proceedings of Specialty Conference: Stresses in Underground Structures, Ottawa, October, 1990, pp 110-119.

76. Quesnel, W., and Chau, P., **Mine Design Validation Utilizing 3-D Finite Element Model, La Mine Doyon**, First Canadian Symposium on Numerical Modelling Applications in Mining and Geomechanics, Montreal, March, 1993, pp 29-39.

77. Assabgui, R., **Fall of Ground - 7B Sublevel**. Internal Report, Teck-Corona Operating Corporation, David Bell Mine, October 1, 1989.

78. Taylor, H.K., **Mine Valuation and Feasibility Studies**, Industry Costs, Northwest Mining Association, Spokane, 1976, pp 1-17.

79. Carlisle, D., **The Economics of a Fund Resource with Particular Reference to Mining**, American Economic Review, September 1954, pp. 575-616.

80. Cooper,R,. Caplin, Robert S., Implementing Activity-Based Cost Management, Institute of Management Accountants, Montvale, NJ, 1992, pp 1-7.

81. The Globe and Mail, November 29 and 30, 1993

82. Wiles, T.D., and Nicolls, D., **Modelling Discontinuous Rockmass in Three Dimensions Using MAP3D,** First Canadian Symposium on Numerical Modelling Applications in Mining and Geomechanics, Montreal, March, 1993, pp 40-49.

83. Fotoohi, K., **Nonlineal Boundary Element Analysis of a Rockmass with Discontinuities**, Ph.D. Thesis, Mcgill University, October, 1993.

APPENDIX A

Table A1. STOPE BLASTING SCHEDULE

Note: The first date is for the slot blast, the second date for the final blast.

BLOCK EXTRACTION NUMBER	BLOCK DESIGNATION	BLAST DATE
1	L71A	Feb-May, 88
2	L77A	May 20-Jun 8, 88
3	L71B	Jun 6-Jul 8, 88
4	K77A	Oct 13-30, 88
5	K77B	Nov 4-12, 88
6	L81A	Nov 23-25, 88
7	K73A	Dec 5-20, 88
8	L85A	Dec 28, 88-Jan 17, 89
9	J73A	Jan 27, 89
10	M85A	Feb 1-18, 89
11	J73B	Feb 21-24, 89
12	L75A	Mar 3-15, 89
13	K81A	Mar 13-14, 89
14	K81B	Mar 24-30, 89
15	K71A	Apr 3-5, 89
16	K85A	Apr 16-18, 89
17	K85B	Apr 25-28, 89
18	L81B	Apr 25- May 9, 89
19	K75A	May 20-25, 89
20	L85B	Jun 4-9, 89
21	M85B	Jun 20-28, 89
22	L73A	Jul 17, 89
23	L77B	Aug 3-18, 89
24	K73B	Aug 5-28, 89
25	M81A	Sep 18-27, 89

Table A1. (Cont)

BLOCK EXTRACTION NUMBER	BLOCK DESIGNATION	BLAST DATE
26	M81B	Sep 18-27, 89
27	L73B	Oct 14, 89
28	L83A	Nov 3, 89
29	L87A	Nov 3, 89
30	K83A	Nov 15-18, 89
31	K83B	Nov 15-18, 89
32	J81A	Nov 27, 89
33	J81B	Dec 4, 89
34	M87A	Dec 14-16, 89
35	L75B	Dec 29, 89-Jan 5, 90
36	M83A	Jan 23-29, 90
37	K75B	Feb 9-16, 90
38	M82C	Feb 27-Mar 8, 90
39	M86C	Mar 18-24, 90
40	K71B	Apr 6-15, 90
41	L72C	Apr 20-May 5, 90
42	L82C	May 15-25, 90
43	K87A	Jun 6-16, 90
44	K87B	Jun 18-28, 90
45	M83B	Jun 22-Jul 2, 90
46	J71A	Jul 5-17, 90
47	J71B	Jul 19-Sep 11, 90
48	K72C	Jul 25-Aug 11, 90
49	L76C	Aug 19-Sep 3, 90
50	M82D	Sep 4-30, 90

Table A1. (Cont)

BLOCK EXTRACTION NUMBER	BLOCK DESIGNATION	BLAST DATE
51	M87B	Oct 10-16, 90
52	L72D	Oct 23-28, 90
53	L86C	Oct 30-Dec 19, 90
54	K76C	Nov 13- 23, 90
55	L76D	Dec 3-13, 90
56	M86D	Dec 20, 90-Jan 29, 91
57	K72D	Jan 3-13, 91
58	K82C	Feb 1-23, 91
59	L74C	Mar 5-12, 91
60	K76D	Mar 20-23, 91
61	L83B	Apr 2-12, 91
62	M84C	Apr 20-May 15, 91
63	J72C	May 27-29, 91
64	L78C	Jun 11-21, 91
65	J72D	Jun 23-28, 91
66	M88C	July 5-16, 91
67	J76C	Jul 18-26, 91
68	J76D	Jul 26-Aug 6, 91
69	L87B	Aug 3-14, 91
70	K4,6 (K61A)[1]	Aug 21-28
71	K86C	Sep 2-24, 91
72	L84C	Sep 19-26, 91
73	K86D	Oct 6-12, 91
74	K74C	Oct 7-17, 91
75	K78C	Oct 22-Nov 1, 91

Note[1]: The stope numbering system at the mine changed. The old numbers given in brackets correspond to the numbers shown in the Figures in Chapter 6.

Table A1. (Cont)

BLOCK EXTRACTION NUMBER	BLOCK DESIGNATION	BLAST DATE
76	Note[2]	
77	M84D	Nov 10-18, 91
78	L88C	Nov 24-Dec 2, 91
79	K2,6 (K65A)	Dec 6-21, 91
80	L2,6 (L65A)	Dec 28, 91- Jan 31, 92
81	J74C	Jan 16-27, 92
82	J4,6 (J61A)	Jan 28-Feb 16, 92
83	K4,8C (K82D)	Feb 19-Mar 21, 92
84		
85	L1,7C (L78D)	Mar 2-30, 92
86	L4,6 (L61A)	Mar 10-16, 92
87	K3,7C (K74D)	Mar 26- Jun 6, 92
88	L4,8C (L82D)	Apr 3-15, 92
89		
90	J2,6 (J65A)	Apr 11-25, 92
91	K1,7C (K78D)	May 7-Jun 3, 92
92	K3,8B (K84C)	May 20-Jun 5, 92
93	L3,7C (L74D)	Jun 8, 92
94	K3,8C (k84D)	Jun 18-20, 92
95		
96		
97	L2,8C (L86D)	Jun 1-8, 92
98		
99		
100	J3,7C (J74D)	Jul 22-Aug 3, 92

Note[2]: Where the stope number is not shown, the stope was either above or to the east of the modelled area.