Deep Underground Hard-Rock Mining - Issues, Strategies, and Alternatives

by

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Abstract

Underground hard-rock mining is an important segment of the Canadian mining industry. Its survival, however, is being threatened by both external and internal forces. In the external front, low cost producers such as local open-pit mines and foreign operators keep raising their production rates and lowering the price of hard-rock mining commodities. Internally, ever-increasing depth of mining of existing underground operations and the depth of the new discoveries mean higher capital and operating costs.

This research has identified five factors that most critically affect the profitability of deep underground hard-rock mining: vertical ore and waste transport; horizontal ore and waste transport; ventilation; mine development; and personnel and materials distribution. Each of such factors has been analyzed, theoretically and through two case studies, in order to determine their relative strategic significance. Solutions to the problems currently being faced by deep mine operators have been discussed as well.

Strategic analyses indicate that deep mines can be designed and planned so that their profitability is not seriously affected by such factors. Solutions to ventilation and vertical and horizontal transport problems are strongly influenced by technological innovation (air conditioning, automation, slurry pumping, etc.), reducing the ability of the operator to strategically address them. On the other hand, hard-rock mine development is a critical phase of the ore production process that cannot be carried out by automated or continuous equipment. It requires careful design and planning and, thus, has a very high strategic component. Decisions made at the development stage have profound effect on the economics of an operation over its entire productive life. The case studies included in this thesis emphasize the importance of properly evaluating mine development alternatives, particularly in the areas of equipment selection and opening design (e.g., cross-sectional area and inter-level spacing). Given the sizeable resources and long periods devoted to mine development, it also has a significant economic impact by itself, and usually constitutes the limiting factor in production rate determination.

The economic evaluation of the mining scenarios constructed as part of the second case study clearly indicates that production cost by itself is not a sound parameter for the evaluation of the economic performance of a mining operation. It also showed the benefits of using discounted cash flow methods techniques and demonstrated the existence of a mine development configuration which results in a production plan that maximizes the return on the investment.

Finally, the need for and main characteristics of a computer-aided underground mine design and planning tool were critically discussed in this thesis. The development of the second case study, which required the use of AutoCAD, spreadsheets, and project management programs to manipulate large quantities of data and involved calculation-intensive tasks, highlighted the complexities and interdisciplinary nature of underground mine evaluation. This type of analysis would be very beneficial to the industry, but the time and resources demanded due to the lack of proper computer tools make it impractical to carry it out routinely.

Acknowledgements

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I would like to acknowledge the financial support of the Mining Research Directorate (MRD, now CAMIRO) for the initial development of this project.

Many of the ideas presented in this thesis were informally (but intensely) discussed with fellow graduate students at Queen's University. Although we devoted most of our time and energy to issues other than our own research projects, such discussions had a profound effect on my understanding of the difficulties facing the future development of the mining industry. Thus, at the risk of missing an important/controversial contribution, I would like to thank Aaron Mak, Baqun Ding, Darren Koningen, David DeGagné, Henry Heidrich, Juan Camus, Mike Lewis, Mohamed Kolahdoozan, Paulo Franca, Rob Lee, Steve Williams, Todd Harvey, William Arrocha, and Yaohong Jiang.

Last but not least, I thank my wife Carola for her support, encouragement, and patience. It is evident that she realized better than I did how important this project was to me.

Dedicated to

Carola Adriana

and

Amanda Lucía

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1. Introduction

The mining industry worldwide has always been characterized by its dynamic and global nature. The same forces that brought the Europeans into the Americas at the end of the 15^{th} Century and developed the Chilean copper industry in the early 1800's[1] are now carving the shape of what might be the mining areas of the future in countries such as Argentina, Ghana, and Tanzania. Not surprisingly, mining recently has been defined as an *"environment which can only be charitably described as turbulent, unpredictable, uncertain and uncontrollable."* (Richardson, 1993).

The response from major North American mining corporations to the latest trend in exploration and mine development has been varied. Some have moved their main exploration offices overseas, into countries such as Chile (i.e., Falconbridge and TVX Gold), whereas other firms actively have begun seeking foreign partners and invested heavily in new foreign operations (i.e., Barrick Gold, Noranda, and Placer Dome). This is not to say that the majors have forgotten their roots: new mines are still developed and current operations expanded,[2] but larger portions of their budgets are routinely invested elsewhere (Crowson, 1995; Anonymous, 1999, p. 3).

This thesis addresses the challenges facing North American underground mine operators as their mines become deeper and the profit margins shrink. The analysis is carried out from an operational point of view and the option to divest, although perfectly valid to the corporation as a whole, is not further investigated. The implicit assumption made is that underground mines will have to remain viable business opportunities if they are to survive the current re-structuring of the international mining community.

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1 British capital started large-scale copper mining in Chile in the early 1800's but the industry collapsed in 1827. Nonetheless, production increased gradually and in 1850, Chile became the leading copper producing country in the world. The exhaustion of high-grade ore resulted in a rapid decline in copper output, and the industry collapsed again during the late 1890's. American capital rebuilt it twenty years later (Culver and Reinhart, 1985; Pareja, 1992). Similar fluctuations have been experienced by the Cornish mining industry in the UK; the North American tungsten industry (destroyed by low-cost producers from China); and the European iron-ore industry (severely affected by low-cost producer CVRD from Brazil).

2 About 50% of non-ferrous North American mining companies explore and develop mining properties in Canada and 48% are still active in the U.S.A. (Silver, 1996).

1.1 Underground Mining in Canada

Given the intensity of the exploration programs carried out in Canada during the past twenty years, it can be expected that most of the new base and precious metal deposits to be discovered and brought into production in the near future will be medium to high grade underground operations. The discovery of the large nickel-copper deposit at Voisey's Bay, Labrador, being an extraordinary exception, only confirms the rule. A similar situation exists in the United States and most of the developed world. On the other hand, the vast majority of the developing world has not undergone serious or systematic grass-roots exploration. The potential exists for the discovery and eventual development of a number of low-cost, high-grade open pit and shallow underground operations. Furthermore, the relative improvement in the economic and political conditions of some developing countries has opened up entirely new exploration areas in places such as China, Vietnam, and the republics that used to be part of the former USSR.

Within this framework, the competitiveness of existing Canadian underground base and precious metal operations is undermined by the following three important factors:

- ***Depth of the deposits.***

  Most of the deposits are currently located at moderate to high depth (900 to 1500 metres below surface).[3] This not only results in long and expensive exploration and development periods, but also in higher capital and operating costs, and complicated production plans.

- ***Local economic conditions.***

  The relative strength of the Canadian dollar turns imported products, such as ores and metals produced overseas, cheaper[4]. Furthermore, the higher cost of Canadian labour and the associated compensation packages significantly increases the labour component of the total production cost.

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[3] Of the 49 underground hard rock operations that provided shaft data to two mine surveys (Scales, 1998; Scales, 1996), 30 had production shafts deeper than 900 metres. The average depth of such shafts was 1235 m, whereas the average depth of the 19 shallower shafts was 712 m.

[4] Since 1991, Canadian copper, nickel, and zinc producers have benefited from a weaker Canadian dollar and gained in relative competitiveness. However, this gain has been against American producers mainly, since the currencies of other mineral-producing countries such as Australia, Ireland and Sweden, have continued to be much lower relative to the US dollar (Natural Resources Canada, 1994).

- ***Environmental concerns.***

  This factor will become even more important in the future. Most environmental groups oppose the types of mining that are considered disruptive to the environment such as open cast, mountain-top, and large-scale open pit operations. However, radical groups would like to prevent the development of underground mines as well. Evident achievements of the environmental movement are that it takes longer to obtain the exploration, development and mining permits; fewer areas are available for exploration and/or mining; and the number of financial institutions willing to provide the resources required to develop mining projects is being reduced.

In addition to being affected by these factors, **new** underground mining projects in Canada also will have to face the following challenges:

- ***Exploration cost.***

  With very few exceptions, the most accessible and easily identifiable deposits have been discovered already. Currently, mining and exploration companies have to look in more isolated and deeper areas, which can be done only at significantly higher costs. Furthermore, longer exploration programs are required and the chances of making an economic discovery are smaller.

- ***Quality of the deposits.***

  Contrary to what was expected in the mid-seventies, the switch to very low-grade, high-tonnage underground operations, with production rates of up to 100,000 tons/day, never took place (Gentry, 1976; Thomas, 1976). This is not only due to the fact that such operations are not viable under current economic conditions, but also to the many drawbacks inherent to large scale underground mining. Those include difficult and expensive ground and grade control practices, long development stages, lack of flexibility, high capital expenditures, etc. Furthermore, some of the recent deep discoveries, such as Inco's Victor deposit in the Sudbury camp, have very high grade and lack the volume to warrant the development of very high tonnage operations. Nonetheless, if the term "quality" is used as an indicator of the overall attractiveness/profitability of a deposit, it can be argued that new discoveries will be of lower quality than those brought into production within the past decade

- ***Development Phase.***
  The development phase which, as discussed above, is extended due to the depth of the deposits, becomes even longer as a result of the permitting process and the number of environmental studies/reviews required to start any mining activity.

As indicated by the series of studies carried out by the Mining Sector, Natural Resources Canada (Natural Resources Canada, 1994) and the Intergovernmental Working Group on the Mineral Industry (IGWG, 1992) the government and the industry are genuinely concerned about the impact of global competition on the future of the Canadian mineral industry. In the particular case of underground mining, the response from operators and researchers has been technological to a large extent (Paraszczak, 1995; Scoble, 1994; Baiden et al., 1993a). In other words, it is thought that by adopting new technologies the mining cost will be reduced (mainly through the reduction of the labour component), the profitability increased and the operations will become competitive again. This is not entirely wrong: little can the industry as a whole do to change the economic and political conditions of the country or significantly affect (i.e., increase) the price of the commodities produced. The solution, thus, has to come from within the corporations themselves and an important area to improve is the productivity of the equipment and work force.

Nevertheless, the development of purely technical solutions is bound to encounter significant obstacles and their success is not guaranteed (Wagner, 1995; Dietze, 1993; Singh and Hedley, 1981). It is postulated that the response cannot be exclusively technological, i.e., other aspects of the problem that directly affect the operations have to be taken into account as well. A comprehensive strategy that encompasses every facet of the problem and is developed according to corporate and business objectives is required. In this way, issues such as the impact of the new equipment and processes on mine personnel, professional training programs and the production process itself can be better addressed, ideally in an integrated manner (Pareja and Pelley, 1995c; Vagenas and Clément, 1995; Rosengren, 1988).

The survival of a healthy Canadian underground mining industry depends on the coordination of efforts from the several layers of government (Federal, Provincial, and Local), clear vision from the business community, and sound operational practices. The governments can only provide the adequate environment and conditions for the industry to develop. It is up to the operators to apply their experience and knowledge in order to benefit from them.

1.2 Objectives and Scope of the Thesis

The main objective of this research is to identify and study the factors that most significantly affect mining at depth and evaluate their impact on the economic viability of deep underground hard-rock mines. Aspects regarding such *factors* that demand special attention include:

1. the constraints they impose on the mine design and production plan;
2. the changes of their relative significance as the depth of mining increases; and,
3. their respective strategic implications, from an operational point of view.

A secondary objective, which can be considered as a natural consequence of the study of deep mining, is to critically discuss a new approach to the design and planning of deep underground hard-rock mines. The study of the subject of underground mine design and planning inevitably raises the issue of a *computer-assisted system*. Such a system, as envisaged by mine operators, not only integrates the process but also allows the rapid and efficient evaluation of alternatives regarding excavation designs and production plans. This thesis defines and discusses the basic components and tools that must be integral parts of such a system.

Finally, this thesis also investigates the efficiency and adequacy of the cost-collecting methods currently being employed at most mining operations. The purpose in this case is to determine if there is a need for a better system, more suitable for underground production control and mine management in general.

1.3 Methodology

To achieve the objectives outlined in the previous section, the research project followed a series of steps:

a. Background bibliographical research on the subjects of underground mine design and planning, mining sequencing, production scheduling, geotechnical mine design, mine automation and mechanization, and underground mining operations management.

b. Site visits to underground operations, including David Bell Mine, Teck Corp - Homestake Mining Corp.; Kidd Creek Division, Falconbridge Limited; Creighton Mine, Inco Limited; and Campbell Mine, Placer Dome Canada.

c. Compilation and processing of relevant information provided by various operations. The following information was requested:

    - orebody information;
    - detailed cost data; and,
    - mining plans.

d. Development of a model and its corresponding computer-assisted implementation that would be used to test the applicability and validity of the analysis and conclusions reached in the study. This would be carried out through a case study using real data adequately disguised in order to protect its confidentiality.

1.4 Main Factors Considered in the Study

Since the outset of this project, the existence of some factors that most significantly affect the profitability of deep underground mines was observed. Indeed, after interviewing the mines' personnel and discussing the subject with them, several such factors were identified. It was then agreed that the research should focus on them, as they were perceived to become even more critical in the future. Those factors are as follows:[5]

[5] This list only reflects *qualitative* evaluations of the impact of the factors on the profitability of the operations. The objective of the discussions held was to define a starting point for the research.

1. vertical ore/waste transport;
2. lateral (horizontal) ore/waste transport;
3. ventilation;
4. mine development; and,
5. cost of labour and supplies.

It is interesting to note that twenty years ago, Gentry (1976) qualitatively analyzed the development of deep mining techniques and concluded that the most significant problems were to be found in the areas of:

1. material transport (ore and waste);
2. ventilation;
3. underground mine development;
4. underground working environment; and,
5. ground control.

It can be seen that both lists are very similar. There are three main differences: Gentry specifically addresses underground working environment and ground control problems, and labour and supplies are included in this project's list. The reason for not having ground control as a separate factor is that it is implicitly included in at least three of them: vertical and horizontal transport, and mine development. Similarly, the conditions of the working environment directly affect ventilation and the cost of delivering labour and supplies. It has been decided that the best way to study the effects of ground conditions and working environment on the profitability of deep mines is through the analysis of the impact they have on such factors. On the other hand, it is apparent that the issues of labour cost and delivery of supplies are rapidly becoming major concerns in deep underground operations and deserve special attention.

The problem is, it has been twenty years since these factors were first publicly identified as important issues in deep underground mining, and they have not been properly addressed in a comprehensive manner. As will be discussed in a later chapter, this apparent lack of interest only can be explained by the complexity of the issues themselves and the need to rely on special tools and methods for the analyses involved.

1.5 Structure of the Thesis

This thesis is organized in seven chapters. Chapter 1 is the introduction to the subject. It discusses in general terms the future prospects of the Canadian underground mining industry, outlines the objectives of the project; and describes the approach adopted. It also presents the basic background for the research, including the factors identified by the operations involved in the program, which will be the focus of the study.

The objective of Chapter 2 is to establish the need for and main characteristics of the underground mining strategy. The importance of underground mining in general and hard-rock underground mining in particular are highlighted and their features outlined. The term *mining strategy* is expanded to include all aspects directly related to an underground operation, and the relative importance of each of them is evaluated. Strategy development and implementation for deep mines are investigated in more detail, discussing the impact of the characteristics of the ore deposit, and operating style and practices on the above-mentioned factors.

Chapter 3 is focused on the design and planning process of underground hard-rock mines. The main aspects specifically addressed are: the objectives of underground mine design and planning; the impact of technology and technological innovation; and the evolution of mining sequences in deep Canadian hard-rock operations. The interrelationships between the tasks of grade control, underground ore transport and ground control are discussed from a mine design and planning perspective. The concept of an integrated computer-assisted mine design and planning system is introduced and its (required) main features outlined.

Chapter 4 focuses on the particular case of deep underground hard-rock mines. Each of the five *factors* discussed in Chapter 1 is dealt with in detail in order to determine its significance from mine design and operational perspectives. The analysis is based on extensive literature research and general information provided by the sponsors of this research project. Furthermore, the comprehensive/strategic approach is maintained, and the problems and options available in every case are presented and discussed in the light of the background provided in the first three chapters.

The remainder of the thesis consists of two case studies and the general conclusions. The case studies are used to demonstrate the applicability of the concepts introduced and discussed in the first four chapters, and to determine the validity and relative importance of each of the factors presented in Chapter 4.

Chapter 5 is a case study based on an existing deep underground base metal operation that is currently considering several options for deepening the mine in order to extend its productive life. The operator provided a wealth of information that was used to analyze the current cost structure, critically discuss important operating issues such as dilution and mine development, and evaluate some of the alternatives available to the future deep operation.

The traditional nature of the cost-collecting system and cost-management style of the operation that provided the data for the previous case study did not allow the investigation of detailed mine development issues or changes of cost structure with depth. As noted above, they were two of the main objectives of this research project. Thus, it was decided to create an *artificial* case study with the specific purpose of looking into the depth/cost structure and mine development issues. Such a case study, presented in Chapter 6, is based on computer-generated orebody models and mine structures and uses the technique of *scenario analysis* to investigate the effect of different development opening sizes on mining strategy.

The main conclusions and recommendations are included in Chapter 7.

2. A Strategic Approach to Underground Mining

Traditionally, the expression *"mining strategy"* has referred to *"mining sequence"*, i.e., to the interaction between the mining and extraction methods, the mine layout and the extraction sequence (Pelley, 1994). The notion of a formal, integrated underground mining strategy has never received significant attention from the academic and industrial communities. Largely, this is because a theoretical approach to the problem is not applicable, and the methods and tools needed to adequately formulate and develop underground mining strategies are not available. However, the increased erosion of the competitiveness of North American underground operations is threatening their survival and makes it imperative that opportunities for improving overall productivity are identified and pursued (Scoble, 1994). This can only be achieved by carrying out a comprehensive evaluation of the factors involved in the production process, considering the long-term effects of operating decisions.

This Chapter is an extended version of Pareja and Pelley (1995a). It presents the basic framework for the formulation and development of a strategic approach to decision-making in underground hard-rock mining. Establishing such a framework involves the definition of the objectives of the mining strategy and understanding the characteristics of decision-making in mining operations. It also requires the specification of the methods and tools needed to integrate the design, planning and production processes. The development of an underground hard-rock mining strategy is a complex process affected by factors such as the financial capabilities of the operator, the geological and geotechnical characteristics of the ore deposit, the technologies to be employed, and the market conditions. Not recognizing the importance of such factors at the design and planning stage may cause serious problems when the actual mining operations take place.

2.1 Underground Hard-Rock Mining in Canada

Underground hard-rock mining is a significant segment of the Canadian mining industry. In 1996, 97 underground mines (about 65% of the total number of Canadian mines) accounted for 30% of the total tonnage of ore produced in Canada (Figure 1). Out of the 281,000 metric tonnes (*tonnes*) of ore mined every day in underground operations, about 207,000 tonnes (74%) were produced by 79 hard-rock mines (Figure 2). Finally, Figure 3 depicts the importance of bulk mining methods in underground mines: they were used at 61 operations to produce 92% of the total underground hard-rock ore.

The analysis of the data on advanced Canadian mining projects provides a somewhat different picture, more in agreement with current trends in the industry. In 1996, underground mining projects accounted only for 14% of the total ore production capacity scheduled to be brought into operation in the short-term (Figure 4a)[6]. However, underground mining projects still outnumber their open pit counterparts (Figure 4b). Bulk mining will become even more important in the near future: it will be used to mine 87% of the ore involved in new underground projects (Figure 5a). It is also significant that precious metal mining (both bulk and selective) will account for about 49% of new underground hard-rock ore production and 56% of new operations (Figure 5a & b).

Figure 6 allows us to put this brief overview of advanced Canadian mining projects into perspective. The focus of open pit mining and exploration companies is on gold. In fact, 47% of the new open pit projects will account for 73% of the new open-pit production capacity, whereas the corresponding figures for base metal mining are 35% and 24%, respectively. Nonetheless, it is reassuring to see that base metal open pit mining (mainly low-grade copper mining) is still an economically viable option in Canada.[7]

In addition to producing base and precious metals, the main features that distinguish underground hard-rock Canadian mining are as follows:

- mine design and planning are strongly controlled by the geological/geotechnical features of the orebody and surrounding rock mass; and,
- the strength of the rock (usually having an unconfined compressive strength of more than 124 MPa) precludes the use of existing continuous mechanical excavating machines (Bullock, 1994).

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[6] No new underground soft-rock mining projects were reported.

[7] The precarious situation of open pit copper mining in Canada and its strong dependence on high copper prices are highlighted by the temporary closure of Highland Valley and the uncertain future of Gibraltar (both of them located in British Columbia).

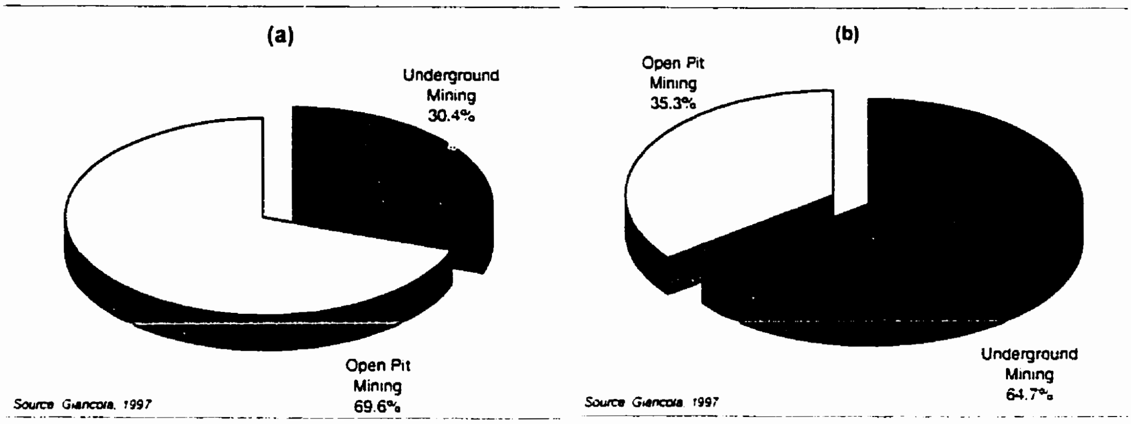

Figure 1: Canadian mining in 1996: (a) tonnage, (b) no. of mines

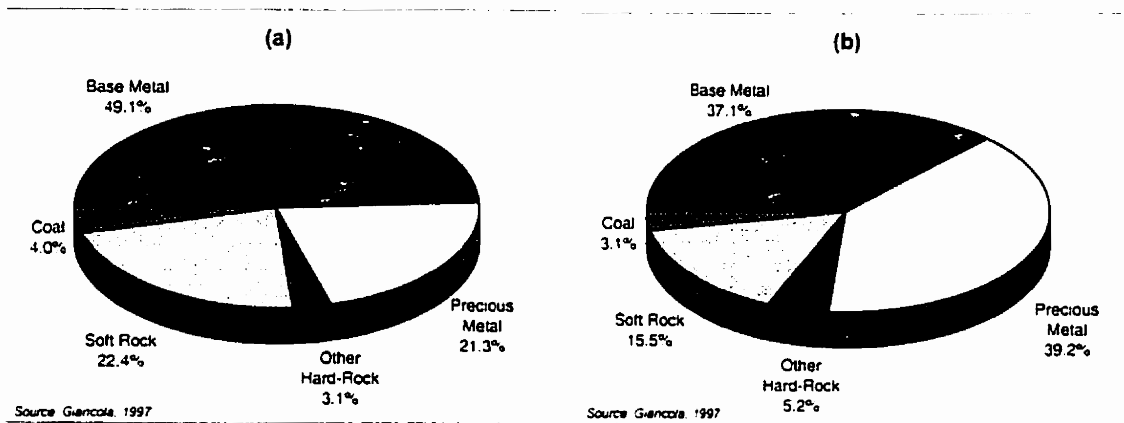

Figure 2: Canadian underground mining in 1996: (a) tonnage, (b) no. of mines

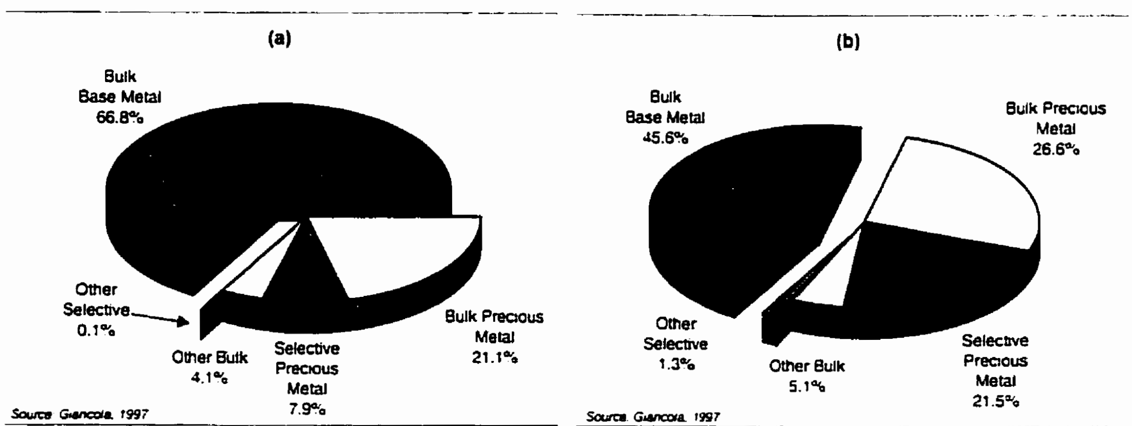

Figure 3: Canadian underground hard-rock mining in 1996: (a) tonnage, (b) no. of mines

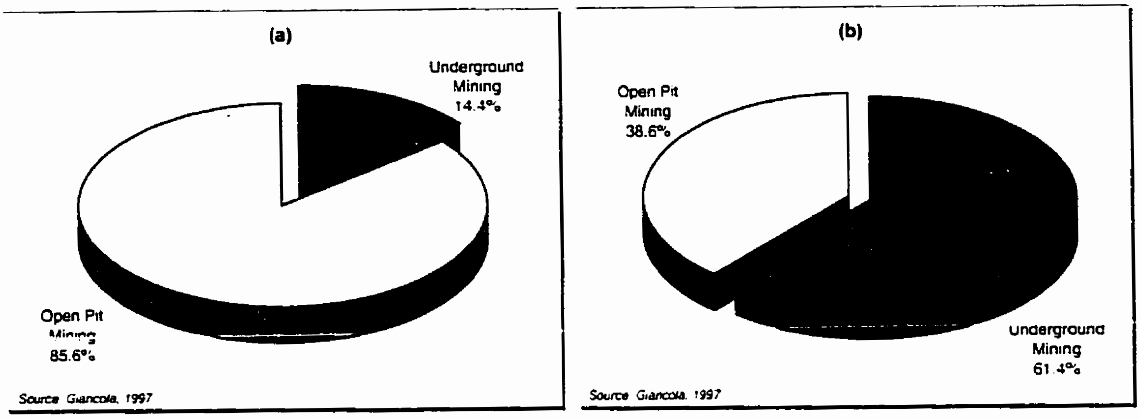

Figure 4: Canadian mining projects in 1996: (a) tonnage, (b) no. of projects

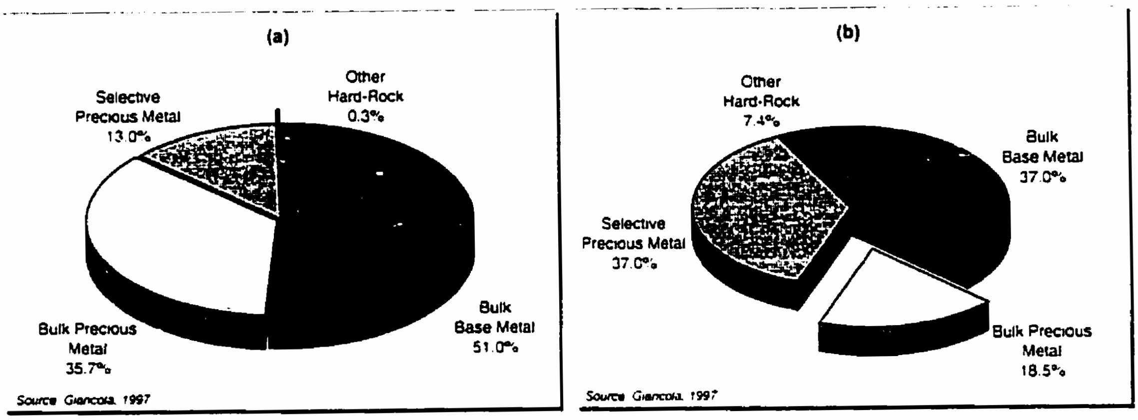

Figure 5: Canadian underground mining projects in 1996: (a) tonnage, (b) no. of projects

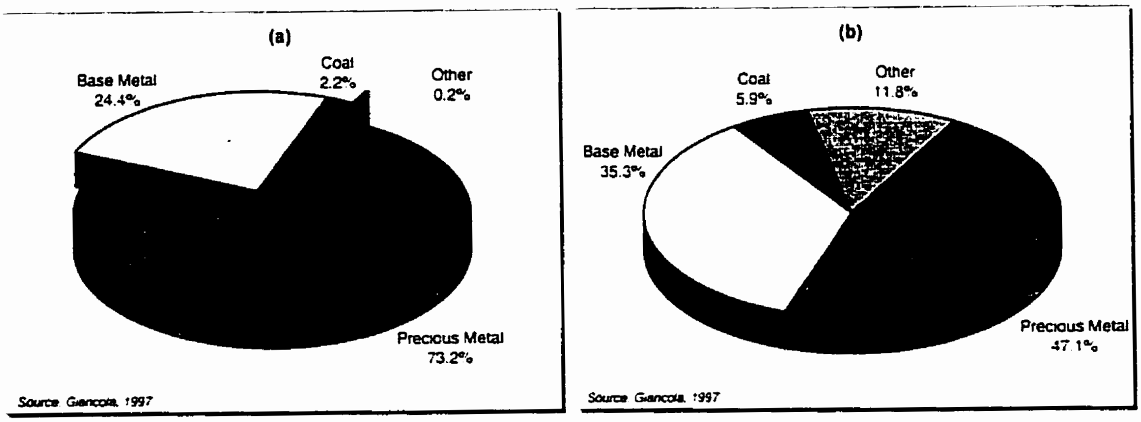

Figure 6: Canadian open pit mining projects in 1996: (a) tonnage, (b) no. of projects

2.2 Strategy: Definition and Levels

The definition of a strategy is multidimensional: it encompasses every activity of a corporation, provides the basic operational guidelines, and facilitates changes or modifications required by the environment and growth. A strategy can be best described by indicating what a firm can realize through its development and application (Hax and Majluf, 1991). In its most general connotation, a strategy supplies a corporation with the following:

- a coherent and integrated pattern of decisions;
- the long-term objectives, action programs, and resource allocation priorities;
- the definition of its competitiveness;
- the proper response to external and internal factors that may affect its competitiveness;
- a channel that differentiates managerial tasks at every level; and,
- a definition of the contribution that it intends to make to those who benefit from its operation.

Decision-making, usually based on strategic planning[8], takes place at several levels within a business organization. At the highest level, top managers deal with decisions that have full corporate scope and cannot be decentralized without risking non-optimal solutions. The objective of corporate strategies is to provide the sense of vision and leadership required by the organization to properly carry out its functions and grow. At the second hierarchical level, business strategies aim at consolidating the long-term competitiveness of a corporation. In mining, they are concerned with the type of commodities to explore for and/or produce; the geographic investment areas; and the development of long-term programs, budgets and management control systems.

This research project is focused on the third and lowest level, which deals with the specific functional requirements of the firm. Functional strategies define the concrete actions that must be

[8] The term "*strategic planning*" refers to the process of providing data, support, and a basis for strategic decision-making. It therefore legitimizes strategic decisions and offers means for controlling their implementation. Strategic planning is usually carried out by planners (or the planning department) whereas actual strategy is made and carried out by management at different levels. See Mintzberg (1994) for a thorough discussion on the subject of strategic planning and strategy formation.

performed in order to implement the business and corporate strategies, and are concerned with each of the functional areas of a corporation. The number, characteristics, and relative significance of such areas vary with the nature of the firm, but usually include finance, human resources, technology, manufacturing/operations, procurement, and marketing. Functional strategies are developed according to guidelines provided by the corporate and business levels and, as is the case of every strategic level, are aimed at addressing the challenges created by the external environment. Coupled with the need to support central strategic decisions that affect several of the functions simultaneously, this results in heavy interdependence among them (Hax and Majluf, 1991).

2.3 The Concept of Mining Strategy

Mining activities have continued for decades without considering the strategic implications of major operating decisions. As indicated by the papers on strategic and financial planning found in Tinsley et al. (1985), mining companies have routinely employed strategic planning to focus their business and corporate objectives and decisions.[9] However, planning and decision-making at the operations level was until recently exclusively based on short-term, tactical considerations (Sloan, 1983). It should be noted that surface mines have historically been more strategically driven than their underground counterparts. Pit operators took advantage of the relative simplicity of surface mine design and planning to develop tools and methods, both empirical and theoretical, that allowed them to implement efficient mining strategies (Breza and Richardson, 1994; Pippenger and Neustal, 1994; Koniaris, 1991).

The inherent complexity of underground mining has precluded the successful development of a similar approach for underground operations. Complicated design, planning, and production processes make it very difficult to develop theoretical solutions to the problems involved. This research postulates that strategic underground mining will only be possible when adequate empirical tools to produce integrated mine designs and production plans are available.

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[9] As noted above, such decisions were typically concerned with the type of commodity to focus on, vertical integration, long-range corporate and business planning, and other non-operational issues (Tinsley et al., 1985).

It is refreshing to see papers such as those by Morrison (1995) and Moss et al. (1995), analyzing and discussing underground mine design, geotechnical, and management topics under strategic frameworks. After years of emphasizing entirely technological responses to complex operational problems, the industry seems to have realized that both technical and non-technical (i.e., managerial) issues must be addressed adequately if mining operations are to confront increased competition successfully. For example, the difference between *effectiveness* and *efficiency*, as noted by Morrison, is pertinent and stresses the importance of the comprehensive approach. It is believed, however, that a mine must carry out a complete assessment of its competitive position and have a sound understanding of its operational capabilities before innovative manufacturing techniques such as *just-in-time* production can be applied effectively. Once an operation is thoroughly optimized, new equipment and methodologies can be adopted rapidly and with a clear knowledge of the type of results and benefits that can be achieved.

An intrinsically complex operation such as an underground mine may be able to succeed in the short- or even medium-term without paying attention to the strategic significance of operating decisions. However, its long-term survival can be severely affected, especially under current metal market conditions and strong overseas competition. In fact, the lack of a mining strategy (formal or emergent) does not necessarily prevent an operation from being profitable, but it may affect it in one or more of the following ways:

- Additional revenues may be lost due to the lack of an integrated program that comprehensively considers the design, planning, and production processes. Improper mining sequencing, for example, may result in an ore feed that does not maximize recovery at the processing facility or might threaten the minable conditions of existing ore reserves.
- The operation may lack the flexibility to:
    a. adapt to market changes;
    b. adopt new technologies; or,
    c. efficiently modify the scale of the operation.
- Technologies and equipment may be inadequately selected, which may not only increase production costs and thus reduce the ability to face the competition, but also limit growth prospects.

Paradoxically, the very nature of underground mining makes it difficult to develop an operations strategy and simultaneously demands an approach that is able to support the decision-making process at the operations level and integrates all aspects of mine design, planning and production. It is necessary, then, to expand the meaning of the expression *"mining strategy"* to include not only mining operations but also the interaction with other functional areas critical to underground mining such as technology, finance and human resources. A mining strategy is based on goals established at the beginning of the design and planning phases, and is developed according to financial, technological and operational constraints identified by the corporation. If properly defined and developed, the strategy provides a logical, consistent and operationally feasible way of carrying out all the interrelated tasks involved at every stage of the mining process.

2.4 Interactions with Other Functional Areas

From a strategic viewpoint, the underground hard-rock mining industry is characterized by the importance of the operations and the complex interactions between operations and other functional areas. Such interactions must be appropriately identified and understood in order to incorporate them explicitly into mining strategy formulation and development. Every functional area requires attention, but finance, technology, and human resources are especially significant. Marketing by itself is not a strategic factor in most underground mines, but market conditions are important and will be discussed.

2.4.1 Finance

Being a capital-intensive activity, underground hard-rock mining is very sensitive to financial management (Caraghiaur, 1985; Robinson, 1985). Most financial decisions are taken at either the corporate or business level, but their operational implications are significant. The impact of financing on mining strategy varies according to the operation (i.e., production capacity, location, products, etc.) and is particularly critical at the exploration and mine design stages. Several key aspects of the mine design, planning and operations are influenced by finance, including:

a. the thoroughness of the exploration and orebody delineation programs, which determine the quality and reliability of the orebody-related information that forms the basis of the mine design and production plan;

b. the length of the mine development stage which, due to the location of the orebodies and difficult ground conditions, is particularly critical in deep operations;

c. the scale of the operation;

d. the technologies to be employed;

e. the ability to phase the development of certain areas of the mine (the so-called *just-in-time* development); and,

f. the specific final mining products (concentrates, refined metals, etc.).

2.4.2 Technology

The selection of the technology and equipment to be used for the various mining operations is a fundamental underground mine design issue. The choice of a particular mining method commits the operation to certain types of technologies, equipment, human resources, and processes that may not lend themselves to short-term variations of the operating parameters. For instance, the current emphasis on bulk mining methods for the future development of existing underground mines implicitly sets the technological framework for many years to come. Once the decisions regarding technology are made and the respective equipment is commissioned, very few changes can be made without incurring additional and sometimes very costly expenditures. On the other hand, if the mine design and production plan allow for the possibility of future technological changes, a seamless transition can be achieved.

Much of the effort currently being devoted to increasing the competitiveness of North American mining operations is focused on the development of technologies that are expected to:

- reduce production costs;
- minimize environmental problems;
- improve working conditions; and,
- improve safety standards.

However, even if such goals are achieved, the impact on the competitiveness of the operation may be minimal (even negative) or only transitory (Skinner, 1985; Stalk and Hout, 1990). Furthermore, in the specific case of underground mines, the analysis of the strategic significance of achieving the above objectives is very difficult. This is because of the need to explicitly consider the complex interrelationships that are common in such mining operations. The effectiveness of technological innovation, thus, can be seriously reduced by the lack of adequate links between R&D and the mining operation (Singh and Hedley, 1981). Similarly, it is impaired by the development of mine designs and production plans that do not consider the long-term effects of the decisions made at this level.

2.4.3 Human Resources

A major concern of mining corporations is recruiting and maintaining qualified personnel. *"Qualified"* is a relative term that depends on technology and the organizational structure of the operation. In underground mining, they heavily depend on the mining method: it dictates not only the specific training and skills that salaried personnel must posses, but also that of line supervisors and management. The principal strategic decisions in human resources management are related to selection and promotion of personnel; performance evaluation; compensation; management development; and employee relations. The objective is to establish policies that motivate the personnel and promote quality and efficiency.

2.4.4 Markets

The mining strategy must provide efficient and operationally feasible responses to market changes. Typical operational questions related to market conditions include when and how to high-grade (a practice not very common in underground mining), the introduction of changes to the mine development schedule, and the production of by-products. With very few exceptions, nickel being one of them, little or no marketing is required in the hard-rock mining industry. Precious and base metals are sold through international commodity exchanges where the price is determined by the market according to worldwide stockpiles, demand, production levels, and global economic conditions.

If concentrates[10] are produced, and if they are *"clean"* or have very high grade and can be blended to allow the treatment of lower grade/contaminated concentrates, there is some room for negotiation with the purchasing smelter. In these cases, the *net smelter return* can be maximized by producing concentrates with the optimum specifications.

In most cases, however, mining operations only can hope to react to changes in market conditions because it is extremely difficult to obtain reliable medium- or long-term market forecasts that can be embedded in the mining strategy.[11] Thus, the volatile nature of the precious and base metal markets, coupled with the fact that commodities are produced, force mining corporations to compete on production cost only: hence the importance of calculating/estimating the actual cost figures accurately. Reliable figures enable the operation to determine its position in relation to the competition and, eventually, face unexpected market conditions successfully.[12]

2.4.5 Discussion

Mining strategy development must be approached integrally: the process has to consider explicitly the overall impact of each functional strategy and take into account the framework provided by the corporate and business levels. The relative importance of each functional area depends on the characteristics of the operation. Some operations are sensitive to finance, such as large mines that employ capital-intensive mechanized bulk mining methods and require extensive

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[10] Canada is an important exporter of base-metal concentrates. In 1992, it exported concentrates containing 321,200 tonnes of copper (mainly to Japan); 716,000 tonnes of zinc (mainly to Germany, Belgium and Spain); and 183,300 tonnes of lead (mainly to Italy, Japan and Germany) (Metallgesellschaft, 1993). Similarly, in 1997 Canada exported concentrates and ores containing 510,000 tonnes of copper (47% of total copper exports) mostly to Japan and China; and 515,000 tonnes of zinc (47% of total zinc exports) mostly to Spain, Belgium, and Germany (ABMS, 1998a; ABMS, 1998b).

[11] Metal price forecasting is a difficult task that requires sound understanding of the economic principles involved **and** how technology, institutions, market structure and government policies and regulations affect international mineral supply and demand (Tilton, 1992).

[12] Porter (1980) identified what he called the *"three generic strategies"* that a firm may pursue in order to improve or maintain its competitive position. They are overall cost leadership, differentiation, and focus, which in reality is just a combination of the first two strategies. As noted in the text, underground hard-rock mines produce commodities (precious and base metals) and, thus, competition is based on cost only, i.e., with very few exceptions, focusing and differentiation are not feasible strategies. The lowest cost producers will have definite advantages regarding issues such as financial planning (healthier cash flows result on safer operations, from a market viewpoint); orebody recovery (lower cut-off grades can be implemented); and weathering market turmoil (they can absorb unexpected drops in commodity prices, at least better than the competition). Maintaining a cost leadership position, however, is not simple and may impose severe burdens on the operation (Porter, 1980).

and complex financial arrangements. Other mines are forced to rely on technology to remain competitive, as is the case of gold operations that develop new ore processing methods to expand their reserve base and treat low-grade or refractory material. Technology is also the key factor in deep mines that must resort to "*unproven*" hoisting technology to bring ore and waste to surface.

Given the high degree of interaction between the functional areas of an underground operation, it is necessary that strategy formulation and development be carried out with adequate coordination among those responsible for each area. This requires that suitable organizational mechanisms be set up so that functional communication and coordination are facilitated and encouraged.

2.5 Strategy Development Issues

In spite of the importance of the functional interactions discussed in the previous section (which cannot be overstated), strategic mining is primarily concerned with operational aspects. They dictate strategy development and decide the applicability of basic tools for decision-making. Critical to mining strategy development is the clear definition of its objectives, the understanding of the mining operations decision-making process, the recognition of the importance of mine design and planning, and the definition of appropriate performance measures.

2.5.1 Objective of the Mining Strategy

The overall objective of the mining strategy is to support the decision making process by facilitating prompt and adequate responses to changes in the design, planning and operation parameters. This general objective, however, must be reflected in more specific or "operational" objectives that depend on the features of the corporation and provide basic guidelines for the various stages of the production process. They must be complemented with the particular goals that other functional areas are to achieve. The operational objectives are to:

1. provide safe working environments;
2. preserve the minable condition of the reserves;
3. maximize the recovery of the ore reserves;

4. minimize the mine development time and cost;
5. minimize the cost per unit of commodity produced;
6. facilitate the operation at the optimum rate;
7. facilitate grade control practices; and,
8. maximize the net present value (NPV) of the investment/project.

As noted by Pelley (1994), it is impossible to develop a mine design and production plan that simultaneously achieves every single optimum condition. In other words, some mines will find it more attractive to maximize ore recovery at the expense of production cost, whereas other operations may be able to maximize the NPV without recovering all the minable reserves. From an operations point of view, the main objective is to reach a compromise between these apparently conflicting goals and produce an operationally feasible mining strategy that is consistent with the corporate and business strategies, supports other functional strategies, and maintains or enhances the long-term competitiveness of the corporation.

2.5.2 Decision-Making at the Operations Level

Since the focus of the mining strategy is on decision-making support, strategy development requires a clear understanding of the decision-making process which, in general, follows a sequence of basic steps:

- identification and clear definition of the problem;
- gathering of the information needed to analyze viable solutions to the problem; and,
- selection and implementation of the most beneficial solution.

The operations level of a firm is faced with very distinct decisions that can be classified as follows (Krajewski and Ritzman, 1990):

a. positioning decisions, that affect the direction of the company;
b. design decisions, related to the production system; and,
c. operating decisions, concerned with the actual operation of the production system.

Very few positioning decisions must be confronted at the mine operations level because most of them depend on the orebody features and the mining process itself. In fact, the products and competitive priorities are determined by the specific commodities contained in the ore to be mined. Similarly, the capital-intensive, high-volume, and linear nature of mining production systems entails a *product-focused* approach.[13]

Design decisions imply long-term commitments, have profound effects on the production system, and are particularly critical to the success of the operation. They not only address *traditional* mining areas such as mine design, mineral processing and support services, but also work-force management, new technology adoption, environmental engineering, and equipment maintenance. Some design decisions are also influenced, if not outright decided, by the type of deposit to be mined. Typical examples are the mine layout, the location of facilities, and some of the processing features. The input from the operations level is always necessary in order to align the design of the mining systems with the overall strategy of the organization, to guarantee a trouble-free implementation of the designs, and to provide long-term guidance to the operation.

Operating decisions are related to the management of the resources involved in the production process. They deal with production plans and detailed short-term schedules. Mining operations managers must decide how to administer personnel, materials, and equipment; how to set up and maintain quality control systems; and how to deal with changes in the operating conditions.

As a direct consequence of the focus on design and operating decisions, the basic instruments for strategic decision-making in mining operations are design, planning, and production scheduling and control tools. In addition to their usual functions, they are expected to provide adequate linkages with the other functional areas and facilitate integrated analyses and solutions. It can be concluded, thus, that the strategic approach to underground hard-rock mining is strongly based on mine design and planning methodologies and tools.

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[13] As opposed to a *process-focussed* operation, where, as the term suggests, the manufacturing facilities are organized around the production process, and positioning is dictated by a particular segment(s) of the market(s) targeted by the company. Process-focused operations typically are flexible multi-product and, thus, have the ability to produce a so-called "*optimum*" mix of products (i.e., a mix of products that maximizes the return on the investment).

2.5.3 Strategic Mine Design and Planning

Underground hard-rock mine design and planning are characterized by their interdisciplinary and iterative features. However, very little effort has been spent in developing the methods and tools needed to integrate the process, and it is usually carried out in stages. As will be discussed in more detail later, the main reason for this is the difficulty in dealing with the information required by the various studies and tasks involved. In principle, an integrated decision-making approach to underground mine design and planning must address the following issues:

- data exchange at all phases must be fast, consistent and error-free;
- the three-dimensionality of spatial data must be explicitly taken into account; and,
- the impact of changes to the design and/or planning parameters must be evaluated in a simple, efficient, and integral manner.

It is critical that the design and planning process includes the evaluation of several alternatives; allows changes in the design parameters as actual conditions vary; and facilitates *what if*, sensitivity and/or risk analyses. It is evident that, due to the complexity of underground hard-rock mining, these tasks only can be efficiently accomplished through the use of computer-assisted methods (Badiozamani, 1992).

The complexity of the data-processing problem stems from the nature of the underground mining process and the ore deposits involved. Unlike surface mining, in which the existence of a unique final pit limit simplifies mine design and planning (Koniaris, 1991), underground mining must rely on trial-and-error methods to determine *optimum* designs and production plans. Open pit mine design and planning are, for all practical purposes, two-dimensional exercises, whereas they are three-dimensional activities in underground situations. Orebody attributes such as physical dimensions, grade and tonnage distributions, depth, and geotechnical parameters determine the three-dimensionality of underground mining. Likewise, key design and planning aspects such as the mining and access methods, mining sequencing and production rate depend on the ore deposit features. They also affect the flexibility of the mine production system and the ability to implement high-grading practices and mine low-grade hangingwalls and/or footwalls.

2.5.4 Providing Support for Operating Decisions

Due to the stringent data-processing requirements and nature of the design and planning process, strategic underground mining is virtually impossible to develop and implement without the help of a dedicated computer-assisted system. Integral parts of the system required to support the underground mine design and planning process are: geological interpretation; ore reserve estimation; mine openings design; mine layout design; mining sequencing; extraction systems design; scheduling; and cost and financial analysis. The system must facilitate the interactive design of the various components of the mine structure as well as the evaluation of the *validity* of such designs in terms of their stability (will this opening collapse?), *economic viability* (is the grade of this stope high enough?) and *practicality* (can this excavation be carried out in this period?). The details of the actual mining plan must be reflected in development, production, and resource utilization schedules for the periods under consideration (Wooller, 1992).

The basis of the system is provided by an integrated database capable of simultaneously handling spatial and non-spatial data, and a computer graphics-based system that allows the representation, manipulation, and analysis of underground excavations and geological features. Adequate design tools and graphical user interfaces should let the engineer/planner carry out the design and planning tasks interactively. Interfaces with other mine design software that cannot be directly incorporated into the system, such as numerical stress analysis programs, must also be provided.

Strategic decision-making is supported by software that implements the approach discussed in this thesis. It must explicitly incorporate the functional interrelationships and those existing between the various areas/studies/tasks involved in underground hard-rock mine design and planning process. It is equally critical to have access to adequate cost and financial data, which not only serves to obtain an accurate picture of the ongoing operation, but also to evaluate the impact of changes in the operating conditions.

2.5.5 Measurements of Strategic Performance

The definition of the quality of the results is an essential part of the strategic approach. Results, or products, must be evaluated in terms of how they improve the ability of the firm to compete. This is not a trivial task, since the evaluation must be carried out in a comprehensive and consistent manner, taking into account the impact on the overall functional strategy. Standard measures of performance in manufacturing are cost, delivery, quality, and flexibility.

As discussed in Section 2.4.4. most hard-rock mining operations must adopt (either formally or implicitly) cost leadership strategies, an inherent characteristic of commodity producers. Delivery and quality are rarely of any significance to hard-rock mines, as long as the standard conditions specified in corresponding sales contracts are met. Operations that sell concentrates containing unusual quantities of by-products or pollutants, which affect the actual value of the product, are the only exceptions. Flexibility, on the other hand, can become a strategic issue, depending on the market conditions, financial capability of the firm, and the orebody characteristics. However, an underground operation cannot be simultaneously flexible and a low-cost producer: the technology, methods, and mine design required by a flexible operation are very different from those demanded by a low-cost producer. Indeed, low-grade, low-cost bulk mines tend to be very rigid operations.

For the reasons outlined in Section 2.4.4, most hard-rock mine operators opt for a strategy aimed at reducing production cost, which protects them from market uncertainties and other risks inherent to the mining business. The success of a mining strategy, thus, is directly related to its effects on the overall production cost expressed in dollars per unit of commodity produced (e.g., dollars per ounce of gold; dollars per pound of copper; etc.).[14] The figure (i.e., the reduced mining cost) by itself is not enough: it must be compared against those obtained by the competition. Equally important is to examine medium- and long-term trends, investigate the effects of the strategy on other functional areas, and evaluate the impact on the other operational objectives.[15]

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[14] The significance of this parameter as a measurement of overall operating performance is obvious: when properly calculated it is an effective indicator of the profitability of an operation and its competitive advantage. However, the mining industry in general still prefers to use the production cost in *dollars per ton of ore* to report, for example, the success of a cost-reduction program.

[15] It is evident that throughout this Chapter there is a strong emphasis on cost reduction as a strategic goal. However, it is important to keep in mind that the objective of a mining operation is to generate a profit, not just pursuing never-ending cost-reduction strategies. In other words, it is not enough to reduce or even eliminate certain cost components such as labour. Accomplishing this would be meaningless, from an overall perspective, if it did not result in an increased profitability and long-term competitiveness.

2.6 Special Features of Deep Underground Mining

All the points discussed in the previous sections of this chapter apply to deep underground hard-rock mines. However, because of their particular characteristics, deep mines demand a more careful approach to their development and operation. In fact, they have in general:

- more complex mine designs and production plans;
- less flexibility; and,
- increased sensitivity to several key factors.[16]

It should be noted that there is a high degree of interdependence between them. For instance, the lack of flexibility of the operations (which makes modifying the production rate or incorporating different types of equipment difficult) is reflected in the mine design and corresponding production plans and schedules. Also, because there are certain factors that more significantly affect these operations, mine design and planning specifically have to address them and determine (i.e., quantify) their importance.

2.6.1 Complexity of the Mine Design and Production Plan

The increased complexity of deep mine design and planning stems from the limited amount of information that is typically available at these stages, and from the need to rely on several additional studies to carry them out. The lack of adequate *hard* data demands that a number of scenarios be investigated in order to determine the *optimum* solution (i.e., the optimum mine design and its corresponding production plan). Usually, most of the parameters to be used in the scenario analysis must be *estimated* due to the lack of sufficient data to *determine* them with acceptable precision or confidence. This introduces additional uncertainty to the process. Also, the number of options to consider within each scenario, and the interrelationships between them, significantly expands.

The mine design and plan reflect several studies in the areas of ore reserve estimation, geotechnical design, mining method selection, mine ventilation, equipment selection, etc. They

[16] The case of deep underground hard-rock mining will be dealt with in more detail in Chapter 4. This section is restricted to discussing these aspects from a strategic viewpoint.

attempt to *estimate*, with varying degrees of confidence, the conditions that would be encountered at depth. There are inherent risks in such estimates, which are also sensitive to certain factors (including depth of mining). It is the task of the team in charge of the design and planning of the operation to consolidate all such studies and propose a solution that represents the optimum, given the information available at the time. It has been pointed out already that the difficulties in handling the information involved in the underground mine design and planning process have precluded the development of a strategic approach in underground hard-rock mining. It is postulated that, *before* attempting to implement strategic mine planning, it is necessary to develop the methodology and tools that allow the manipulation and analysis of such data.

2.6.2 Reduced Flexibility

The long development period and large capital expenditures associated with deep mining significantly reduce the ability to modify the main operating parameters. For example, once the mining sequence is defined and the corresponding excavations are started, it is virtually impossible to switch to a different strategy without compromising the recovery of large percentages of the reserves and increasing the overall mining cost. Similar problems are encountered when trying to introduce equipment that is substantially different (bigger, more technologically advanced) from the one considered in the original design. Depending on the case, some of the openings probably would have to be modified, personnel trained, and the supply of ventilation and energy re-evaluated.

The limits imposed on the mining strategy by decisions taken at the design and planning stages must be clearly identified and understood. In this way, if there is ever the need to change the design and planning parameters in order to adapt to different operational conditions, it can be achieved with minimal disruption of the production process and loss of competitive position. On the other hand, a mine could be designed so that it allows such modifications to occur. There is a compromise between the degree of flexibility that can be provided to the mine and the increased capital and operating cost that it entails.

2.6.3 Sensitivity to Specific Factors

As discussed in Section 1.4, several factors are known to have a considerable impact on deep mines. The list of factors is repeated here for the sake of completeness:

1. vertical ore/waste transport;
2. lateral (horizontal) ore/waste transport;
3. ventilation;
4. mine development; and,
5. distribution/delivery of labour and supplies.

The degree to which an operation is sensitive to any of such factors is a function of the actual features of the mineral deposit, the operator, and the commodity markets. Critical orebody characteristics include depth of the mining area; geotechnical properties of the rock mass (strength, jointing, and stresses); geological features of the orebody (continuity of the mineralization, average grade, grade distribution, shape, and definition of the hangingwall and footwall contacts); and the type of commodity being exploited (either base or precious metal).

As discussed in Section 2.4.1, financial management is particularly important in underground hard-rock mining: the financial strength of the operator determines, for instance, the type of technologies to be used for, and the length of, the development and mining stages. The mining method and the education/training of the labour force also influence choices made at the mine development stage, and affect the mining strategy by determining the flexibility and versatility of the operation. The market plays an important part, since the unit value of the ore, expressed in dollars per tonne, and the definition of ore itself vary with changing commodity prices.

It is interesting to note that the above-mentioned features are interrelated. For instance, there is a strong relationship between the depth of the deposit and the conditions of the rock mass, i.e., the deeper the orebody the more difficult the ground control is. Precious metal deposits tend to have ore of higher unit value, and to be more irregular in nature, than base metal ones. The choice of mining method is not only influenced by the conditions of the rock mass and the geology of the deposit, but also by the financial capability of the corporation.[17]

Empirical evidence provided by the operations visited as part of this research emphasizes the importance of the five factors listed above. In extreme cases, the viability of the entire deep operation can be threatened by poor control over any one of them. The factors will be analyzed in depth in Chapter 4.

2.7 Summary

Modern North American mining corporations cannot afford to operate without a mining strategy. The volatility of the metals markets and increasingly strong competition from foreign low-cost producers make it essential that mining companies clearly know and understand their competitive position, develop efficient responses to changes in the operating conditions, and continuously look for integral ways of improving their overall productivity. This can only be achieved through a strategic approach to mining operations decision-making. Piecemeal solutions to underground mining problems are not acceptable: increased productivity, for instance, is irrelevant if it does not result in a general reduction of production cost.

In spite of the strong interrelationships between the various functional areas, the focus of strategy development is on operations, particularly on decisions regarding mine design and planning. Due to the complex, interdisciplinary, and iterative characteristics of underground mining, strategy development requires a methodology that relies on computer-assisted mine design and planning tools capable of providing decision-making support at the operations level. The goal is to be able to evaluate modifications to operating conditions integrally, taking into account the corporate and business objectives established by top management, and their effects on the long-term competitiveness of the corporation.

[17] In theory, orebody features dictate the choice of mining method (Hartman, 1987). However, the operator's previous experience, availability and cost of technology, and local mining practices play important roles as well. This is demonstrated by the popularity of bulk mining in Canada, drift and fill mining in Nevada, and cut-and-fill stoping in South America.

3. Underground Hard-Rock Mine Design and Planning

The objective of this Chapter is to analyze the development of underground hard-rock mine design and planning strategies.[18] The analysis focuses on aspects of the design and planning process such as mine layouts, extraction systems, and mining sequences. The interrelationships between mine development, grade control, underground ore transport, and ground control are discussed from a design and planning viewpoint. The need for and characteristics of an integrated mine design and planning system that allows the efficient evaluation of alternative mining strategies are addressed.

3.1 Introduction

As discussed in Chapter 2, new technologies have been introduced into the mining industry to augment its economic and financial viability by increasing productivity and efficiency. These already have had significant impacts on the organizational and cost structures of mining corporations (Robinson, 1985). Nonetheless, the evaluation of the overall effect of technological innovation on an underground hard-rock mining operation is a difficult task due to the complex nature of underground mining. Indeed, it is not hard to estimate the direct consequences of modifying a process or replacing a piece of equipment (i.e., lower direct labour costs, more production per hour, etc.). However, it is not easy to determine their impact on other operational aspects such as grade control, ground support requirements, or the mine's long-term profitability.

Singh and Hedley (1981) pointed out that the introduction of mechanized underground mining methods into Canadian hard-rock mines during the 1970's did not result in an increase in overall productivity. They noted that this was due to the lack of adequate links between R&D and the operations, and that effective mine design and planning had to be carried out taking into account the long-term global effects of the decisions made at this level. For instance, a labour-cost reduction program must necessarily consider the fact that increases in productivity (e.g., as

[18] A preliminary version of this Chapter was presented at the Third International Symposium of Mine Mechanization and Automation (Pareja and Pelley, 1995c).

measured in tonnes of ore produced per man-shift) can be completely offset by the relentless increase of unit labour costs (in S/hour).[19]

Currently, underground mines are designed according to the characteristics and requirements of existing technology, thus imposing an extraneous set of restrictions on the design, planning, and production processes. Research projects on underground mining technology, in most cases driven by industry requirements and sponsorship, have focused on specific aspects of the mining process: blasting, continuous excavation, ore and waste transport, etc. Unfortunately, very often they have neglected to consider the overall operating framework under which they are supposed to function. As an example, the stoping machines proposed by Brackebusch (1991) and Planeta et al. (1991) require long, flat stope geometries in order to be reasonably productive. This may be impossible to achieve in deep mines with moderate to high rockburst activity.

In order to take full advantage of the benefits derived from the adoption of new technology, and to avoid conflicting requirements and objectives, a more comprehensive approach to design and

[19] To a large extent, this is confirmed by the fact that labour cost ***as a percentage of total mining cost*** has not changed significantly over the years, as illustrated by data provided by Boshkov and Wright (1973), Dravo Corporation (1974), and Halls (1982), shown in the following tables. Dravo Corporation reported in 1974 that the labour cost of 14 room and pillar U.S. mines was, on average, 52.5% of total mining cost. This percentage is 20% **higher** than the one provided by Boshkov and Wright for the years 1955-1959, and about 13% higher than that estimated by Halls for 1977. Furthermore, Dravo noted that a large block caving mine (full gravity extraction) had a labour cost component equal to 52.1% of total direct cost, close to the 1955-1959 average shown above.

Labour cost as a percentage of total mining cost, 1955-1959 (Boshkov and Wright, 1973)

Mining Method	Labour Cost (% of total mining cost)
Cut-and-fill	56.7
Room-and-pillar (trackless)	43.7
Sublevel stoping	56.9
Sublevel caving	63.3
Block caving	54.2

Labour cost as a percentage of total mining cost

Mining Method	Labour as % of total mining cost by source and year (%)	
	Halls, 1982 Year: 1977	Dravo Corporation, 1974 Year: 1973
Block caving	60.2	26.0 – 52.1
Room-and-pillar (trackless, metal)	46.3	52.5

planning is required. Technology must be developed according to the needs of the operation, and its application must be explicitly incorporated into the mine design and plan. The strategic implications of technological innovation must be also evaluated, so that the long-term competitiveness of the corporation is ensured.

3.2 The Underground Mine Design and Planning Process

The development of an underground mine design and production plan requires a number of steps which, traditionally, have been carried out in more or less independent stages (see Figure 7). One of the reasons for using this disjointed approach, which fails to recognize the definite iterative and interdisciplinary characteristics of the process, is the lack of communication between the different teams involved in the various tasks and projects. Another equally important reason is the difficulty in efficiently storing, retrieving, manipulating and analyzing the information required to carry out such studies and tasks in an integrated manner. Typically, underground mine design and planning relies on large amounts of information that is distributed in three-dimensional space; consists of numerical, textual, and graphical elements; and contains very complex data structures and relationships. It is possible to satisfy the individual information processing requirements of each sub-component of the process, but the implementation of an integrated approach demands special tools and methods that, to the author's knowledge, are not yet available.

3.2.1 Objectives of Underground Mine Design and Planning

Regardless of the design and planning approach adopted, the main objective of developing a mine design and production plan is to facilitate the achievement of the operational goals established according to the mining strategy (see Section 2.5). In practice, the definition of such an objective involves analyzing each of the eight conditions to be optimized and determining which one (or which ones) will be the focus of the strategy.

It has been postulated that the ultimate objective of mine design and planning is the maximization of the NPV of future cash flows (Davis, 1995; Cavender, 1992; Koniaris, 1991; Yi and Sturgul, 1987). It can be argued that this is true only in corporations that are *risk-neutral*, i.e., in firms that consistently are seeking to maximize shareholder value.[20] However, recent research has indicated that most mining companies have objectives *in addition* to maximizing shareholder value, i.e., they are *risk-averse* (Walls and Eggert, 1996). Indeed, depending on the financial position of the corporation, its size and the commodities being mined, other equally important goals include: company survival, maintaining market share, and benefiting all stakeholders of the firm. A design and planning strategy, either explicitly or implicitly, must reconcile all these factors and the operational objectives in order to provide basic guidelines for the construction of the mine structure and the development of production plans and schedules.

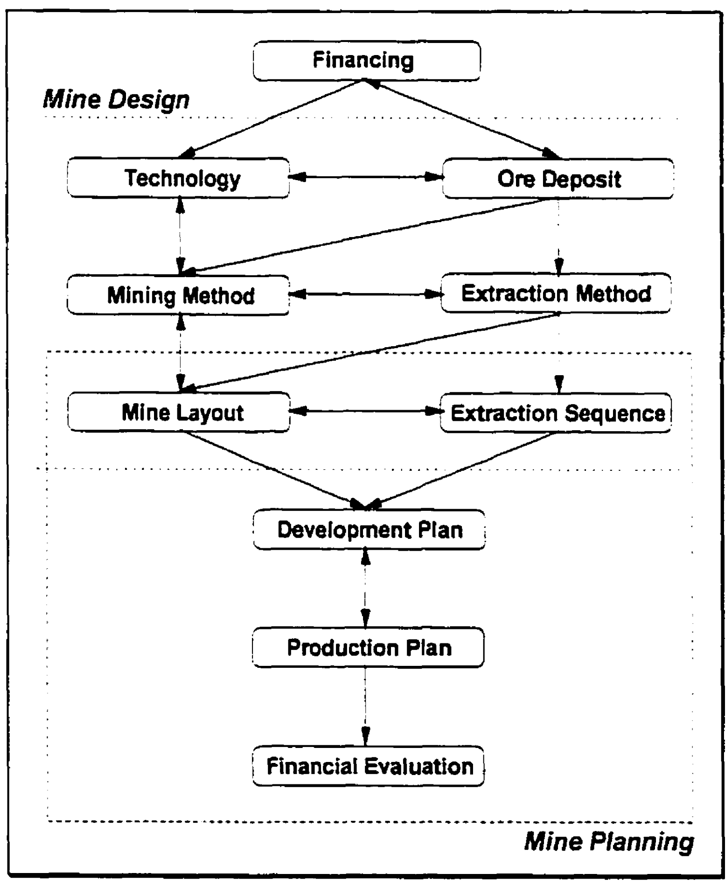

Figure 7: Main components of the underground mine design and planning process

[20] It is generally agreed that the overall objective of **any** firm is to maximize the wealth of its owners (i.e., shareholders). Such a wealth is best represented by the market price of the corporation's common stock. It should be noted that *profit maximization* is not an appropriate goal, since it does not consider, among other things, the duration of the returns or the risks involved (Gentry and O'Neil, 1984).

It should be made clear that there is no doubt the NPV is the best criterion for investment project evaluation (Curry and Weiss, 1993). However, as far as mine design and planning is concerned, the impact on NPV is only one more factor to be considered at the time of deciding the feasibility of a particular option. Detailed mine design and planning are usually done *after* the decision on proceeding with the project has been made (or when the mine is in production). The objective then switches to optimizing the operation as a whole, over the medium and long terms.

3.2.2 Technology

As depicted by Figure 7, the effects of technology propagate through the entire design and planning process. Indeed, by directly influencing critical aspects such as orebody delineation and mining method and extraction system selection, technology determines key features of the mining operation. It should be pointed out, however, that the relationships between technology and the various design and planning components are very complex, and in most cases are characterized by reciprocal influences. This is in stark contrast to what it is usually observed in the manufacturing industry, where there is a one-way relationship between technology and processes and products.[21]

Technology significantly affects several design and planning elements of underground hard-rock mines. For instance, it influences the definition and delineation of orebody limits and ore reserves in two ways. First, it determines the applicability of grade control strategies, (e.g., by facilitating a flexible operation). Secondly, it can increase the minable reserves by allowing the mining of inaccessible ore, or of low-grade ore that was previously considered uneconomical (Wagner and Oberholzer, 1989).

[21] The typical example of this situation is an orebody where there is a high-grade zone and gradational wall rock, i.e., where a continuous transition from high-grade ore to barren waste is observed. Depending on market conditions and location, such an orebody could be successfully developed very differently. For instance, a small or medium-sized operator could build a small, selective mine to extract the high-grade ore only. The mining method, technology, mine design, and mining strategy would be very different from that of a large firm with enough financial resources (and, probably, in-house know-how) to build a large-scale, low-grade, bulk mining operation to recover most of the ore. Thus, technology is determined by the financial capability of the prospective operator. It imposes a restriction on the definition of the ore reserves (through the cut-off grade). At the same time, the features of the resulting minable orebody (tonnage, grade, thickness, shape) will demand (or preclude) the use of certain technologies.

3.2.2.1 Technological Innovation in Mining

New technology introduction is critical to the survival[22] of any industry and mining has benefited extensively from developments in the areas of hard-rock drilling and blasting, ore and waste transport, communications, and ground control. However, there is not a clear procedure to follow for the implementation of new technologies. Previous experience in North America indicates severe start-up problems and long learning/adoption periods in which no real benefits were realized (Singh and Hedley, 1981).

Recent underground hard-rock mine mechanization and automation projects have concentrated on the following aspects of the mining process:

- Communication systems (Hackwood, 1993; Baiden, 1993a; Baiden, 1993b).
- Continuous hard-rock mining systems (Scoble, 1994; Lombardi, 1991; Planeta et al., 1991).
- Tunnel boring machines (Lewis, 1991; Boyd, 1987).
- Automated haulage systems (Whiteway, 1991).
- Rock fragmentation methods (Nantel and Kitzinger, 1990; Anderson and Swanson, 1987).
- Robotics (Konyukh and Becker, 1993; Baiden et al., 1993b).

Other projects were more comprehensive in nature and stressed the importance of coordinating the efforts with other areas of the corporation (Scoble, 1994; Pukkila and Lappalainen, 1993; Baiden et al., 1993a; Udd and Pathak, 1991). In most cases, R&D on automation concentrated on very specific facets of underground mining, which is an acceptable approach given the complexity of the problems involved. However, the projects often disregarded the global significance of the technologies being investigated, and were not related to the operations and overall corporate strategies. It is obvious that some automation projects are driven by the exclusive desire to reduce the direct labour cost component (e.g., Poole et al., 1996). They are ignoring the failures experienced by manufacturing and services companies that followed that

[22] Nickel producers are currently facing the challenge of low-cost Australian mines. Anaconda Nickel, for example, is developing the Murrin Murrin project that is expected to produce 115,000 tonnes of nickel and 9,000 tonnes of cobalt per year from lateritic ore (Anaconda Nickel, 1998). Operating cost before cobalt credits is expected to be US $ 1.10/lb of nickel. The key to the success of this project is the exclusive licensing of Sherritt International's *Acid Pressure Leach Process*, which allows efficient and cost-effective treatment of lateritic nickel-cobalt ores.

approach during the 1980's and 1990's (Skinner, 1985; Abegglen and Stalk, 1985;[23] Stalk and Hout, 1990[24]).

This is not to say that further research is not required, but that the focus and objectives of the programs should be established from operational perspectives, without neglecting the fact that their pervasive effect will be felt at every stage of the production process (Morrison, 1995). Nevertheless, it seems as if research on mine automation has advanced more rapidly than the respective methods and tools required to carry out underground mine design and planning. Such tools and methods are necessary when evaluating the applicability and effectiveness of new equipment and processes.

3.2.2.2 Technological Innovation and Risk

Since the introduction of new technologies is a critical part of the design and planning strategy of any underground mining operation, its impact should be evaluated integrally. It could be argued, then, that mining technologies developed without adequately assessing their overall effects on the design and planning of the operation are seriously risking their viability.

Sometimes it becomes evident that new technology must be adopted by an existing operation or new project. However, once the corresponding decision has been made at the highest level, it should be left up to the operators to decide how to implement it. Previous experience in other industries does not clearly support either rapid or incremental changes (Garvin, 1992). It is

[23] Their research provides evidence that automation and robotics are not the *panaceas* some plant operators thought they would be. In discussing the adoption of *just-in-time* (JIT) systems by Japanese companies, they quoted an executive of a robot manufacturer that said: "*If you want to know what is wrong with your process, try putting a robot in it*" (Abegglen and Stalk, 1985, pp. 116-117). In other words, it is imperative that an operation be running efficiently ***before*** it can be automated. The example of Hitachi was provided, which took three years to implement its version of JIT, and ***then*** embarked on a gradual automation program that was expected to take five more years.

[24] They postulate that a company that wants to improve its long-term competitiveness (and become a *time-based competitor*) needs to carry out a total transformation of its operational practices. With regards to the impact on personnel and labour cost, they stressed that, when compared to traditional cost-cutting measures, the benefits of the comprehensive approach are (Stalk and Hout, 1990, pp. 166-167):

"*... completely out of the range of what is achievable by the following methods:*

- *Cutting direct labour wages through renegotiation or going offshore*
- *Reducing overheads by de-layering management structures and/or narrowing the line of product and services offered*
- *Automation short of the 'peopleless' factory*
- *Obtaining superior economies of scale*"

apparent, though, that careful management of the *change* process and adequate understanding of the issues involved are critical to the success of innovation. Borrowing concepts and terminology from operations research, risk increases as an operation moves away from the current state of affairs regarding product, market, and technology (Garvin, 1992). Underground hard-rock mining is unique in the sense that, for a given operation, neither product nor market changes over the mine life (with very few exceptions, see Chapter 2). On the other hand, it is evident that the introduction of new technology brings into the production process an entirely new set of operating parameters.

For example, deepening projects are intrinsically complex and have to deal with a large number of unknowns such as ground conditions, orebody configuration, ore reserve distribution, and new operating parameters. They have an inherent high-risk component. A mine deepening project involves the development of a new mining process (usually, a new or modified mining method is required). Such a process is applied in a new area, under a different operating environment, and exploiting ore from a new section of the orebody. Paraphrasing Garvin, risk in deep underground mining could be considered as a two-dimensional problem: mining environment and technology. Thus, the introduction of new technology in a deepening project could seriously limit the chances of success. Nonetheless, new technology is usually considered (even demanded) for this type of projects. This is because sometimes it would be impossible to successfully proceed (or justify the investment) without the added benefits of new technology.[25]

3.3 Technology and Underground Mining Sequences

Few areas of underground mining have been more significantly affected by technological change than mine sequencing. To the author's knowledge, the relationship between mining sequence evolution and technology development was first described by Hendricks et al. (1992). Pelley (1994), who focused on the particular case of Ontario underground hard-rock mining, also found a definite link between technology and the evolution of *traditional* and *contemporary* mining

[25] For example, at Kidd Creek this process was illustrated by the progressive move from ramp cut and fill to Avoca and finally to blasthole mining. New methods and equipment were tested in a new (deeper) mining area over an extended period until the "optimum" was reached (McKay and Duke, 1983; Pelley, 1994).

sequences. Pareja and Pelley (1996) and Pelley and Pareja (1996) discussed the economic and operational implications of mining technology development on deep underground mines. They concluded that the serious challenges of deep mining could be overcome only by adopting a strategic approach that incorporates technology, rock mechanics, and mining sequence.

3.3.1 Definition of Mining Sequence

Mining sequence is a general term that refers to the interaction between the mining and extraction methods, the mine layout, and the extraction sequence (Pelley, 1994). In general, the objective of a mining sequence is to extract the mining units as rapidly and efficiently as possible, without affecting the stability of the mine structure, jeopardizing the integrity of the remaining ore reserves, or creating dangerous conditions for personnel and equipment. Its sub-components are so interrelated that it is sometimes difficult to clearly separate them for analytical purposes. A mining sequence describes, in an operational manner, how the mine structure is to be developed and constructed, and how the ore is to be accessed, mined and extracted. In fact, mining sequencing should provide answers to the following questions (Badiozamani, 1992):

a. who? ⟶ mining systems / crews;

b. what? ⟶ sections of the mine / order of mining;

c. when? ⟶ time frame for the exploitation of the ore; and,

d. how fast? ⟶ mining/production rate.

A case study presented by Pelley (1994) highlights the importance of the mining sequence on the design, planning and operation of mines that employ bulk mining methods. It showed that, even though ore recovery and productivity were significantly improved, grade control was negatively affected because standard operating practices such as ore-waste boundary determination were difficult, if not impossible, to carry out under the conditions created by bulk mining.

3.3.2 Evolution of Mining Sequences

The evolution of mining sequences was driven mainly by the need to continue existing operations under more difficult economic and/or mining conditions. Significant (the term *revolutionary* could be used) changes were made possible only by technological breakthroughs and radical modification of well-established operational principles.

In his study of mine sequencing in Ontario, Pelley (1994) distinguished between *traditional* and *contemporary* mining sequences. Traditional mining sequences were developed to exploit both narrow and wide ore using technology available before the advent of large-diameter blasthole drilling equipment in the late 1960's. It should be noted that rockbursting probably was the single most critical factor in the evolution of such sequences.[26] Nonetheless, technological innovation also played an important role in such a process. In fact, the development of hydraulic fill, rockbolts, and cemented hydraulic fill facilitated the return to cut and fill mining with undercut and fill for sill recovery. In the mid-60's, traditional sequences were profoundly affected by the introduction of trackless equipment and mechanized cut and fill, which demanded the simultaneous operation of a series of stopes to increase the productivity of mobile equipment.

Most of the sequences currently used in Canadian underground mines (i.e., the so-called contemporary sequences) were made possible by the increased drilling accuracy of large-diameter in-the-hole drills and precision-machined drill steel (Pelley, 1994). Indeed, they facilitated mining from widely spaced sublevels and resulted in the development of new mining methods such as longhole and vertical crater retreat stoping. Furthermore, the use of remote controlled equipment enabled flexible drawpoint designs and removed the requirement to condition the walls or backs of large open stopes after blasting. Transverse mining sequences were modified to take advantage of the mechanized equipment that was sitting idle in a stope during the drilling or filling cycles. The use of such sequences allowed pre-drilling, delayed filling, and further increased productivity and the productive capacity of individual working locations. Consequently, fewer working places were needed for the same longitudinal area of the mine.

On the other hand, mechanized equipment demands larger openings that are more expensive to develop and maintain, and requires that more service facilities be available. All levels must be

[26] The geotechnical characteristics of the rock mass have always defined the key features of the mining sequence, primarily by imposing limits on the speed of ore extraction, defining the stope geometry, indicating the need to use backfill, and determining the inter-level spacing.

ventilated if equipment is to move to widely disperse mining areas, thus greatly increasing the overall air supply requirement. There is a concentration of mining activity and an earlier interaction between secondary stopes and backfilled primary stopes.

Contemporary sequences include post-pillar cut-and-fill mining; ramp cut-and-fill mining; the tertiary mining sequence (i.e., the so-called *1-5-9* sequence); and single-pass mining sequences such as those used at the Kidd Creek and Creighton mines (Pareja and Pelley, 1996).

3.3.3 Mine Sequencing at Depth

There is little doubt that mining sequencing will be critical to the future of deep bulk underground operations. Two concepts are likely to be relevant to deep mining: single pass sequences and top-down mine development (Pareja and Pelley, 1996).

- ***Single Pass Sequences***

  Single pass sequences provide better management of stress at depth. In these sequences, the vertical dimension of a stope is equal to, or exceeds, its maximum horizontal dimensional dimension. This allows more material to be broken in individual blasts and, since a single drawpoint is required, highly productive material handling systems can be used. This geometry has eliminated the creation of the large horizontal sills of previous sequences that proved difficult to recover and often resulted in rockbursts.

  Single pass mining sequences demand strict 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 at that stage can more seriously affect the economic viability of an operation than the loss of reserves or high cost mining late in the mine's life. Grade control has been made more difficult, especially in low-tonnage mines where a large proportion of the daily production may originate from only a few individual stopes. Early high extraction rates also increase the requirement to monitor ground behaviour so that methods can be modified and properly adapted for use in other areas of the mine.

- ***Top Down Sequences***

Top down sequences are likely to be widely used at depth as, from a development standpoint, a sequence beginning at the top of the orebody and proceeding downwards provides the earliest access to ore. In addition the mining advances towards the main abutment and the stresses are shed to the solid rock mass. Most bulk longitudinal sequences used today, however, progress upward from a main access level. The progression may be to the top of the orebody or to some previously created mining sequence. In the latter case, a major sill pillar is created which requires special methods or a higher level of ground support to complete its extraction. This 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.

Many of these bulk sequences could progress from the top down if access for drilling and filling were created in the backfill of the overlying stopes (Pelley, 1994; Wittchen et al., 1990; Pelley, 1990). This would be less productive than the standard practice, as the cure time of the fill in the previous lift becomes a factor, and access drifts could not be established in advance. Downward sequences should be considered for second stage extraction when mining additional ore below a previously established upward advancing sequence. This would place initial high extraction areas at a mid-mine elevation and allow retreat to occur towards both the upper and lower abutments. The increased costs of the downward retreating sequence would be compensated by reduced early development at depth and by eliminating the high costs associated with sill removal.

3.4 Discussion

There is presently no adequate methodology or tools to design and plan underground hard-rock mines efficiently. Theoretical solutions to the problem are only of academic value and, thus, the approach must be empirical. There is a real need for an integrated, computer-assisted system that facilitates the exchange of information between the various studies and tasks involved; takes into account the three-dimensional nature of the underground mine design exercise; and enables the planner/engineer to investigate changes to the design and planning parameters, including the introduction of new equipment and processes.

The analysis of the design and planning process and a review of the evolution of mining sequences provides insight into the design of the mechanized and automated systems considered critical to the economic viability of the mines of the future. It can be seen that, in most cases, the justification for the ongoing research on automation lies in the area of three of the operational objectives of the mining strategy. In the first place, safety will presumably be increased by reducing the number of workers required underground. Secondly, there is no doubt that a continuous and highly productive hard-rock mining machine will decrease development time. Finally, the reduction in the number of underground workers will increase the productivity per unit of labour input, and decrease the direct labour costs.

It should be noted, however, that reaching these three goals does not necessarily guarantee that the overall mining cost will be decreased or, in general, that the competitiveness of the operation will be significantly improved. These can only be determined through a comprehensive evaluation process that is carried out in the light of the long-term corporate strategy, and considers the global effects of the technology under consideration.

The danger is that the development of large automated systems or continuous mining machines will demand large horizontal stopes to achieve the productivity required to justify the substantial capital investment. Most, if not all, of the continuous mining systems described in the current publications will require a return to a horizontal cut and fill mining cycle even if ore-cutting is provided by a mechanical excavating device. Sequences such as those used at the Falconbridge Mines (Figure 8) or at the Golden Giant mine (Figure 9), which demand a multi-level operation and stopes with high vertical to horizontal dimension ratios, would not be possible using such systems. It is evident that single-pass sequences such as those developed at the Creighton and Kidd Creek mines (see Figure 10 and Figure 11, respectively) are essential to the success of productive mining operations at the highly stressed environment of deep mines.

3.5 Summary

Underground hard-rock mine design and planning is a complex process that requires an integrated approach in order to be carried out effectively. The integration of the various tasks and studies involved demands special tools and methods which are not yet available. More research in this area, thus, should concentrate on the development of computer-assisted design and planning tools which not only facilitate the process, but also make it possible to rapidly and efficiently evaluate mining alternatives.

The continuous adoption of new technology is critical to the survival of the mining industry and has profound effects on the mine design and production plan. Nonetheless, failing to acknowledge the importance of the strategic implications of technological innovation may reduce or even eliminate the possible benefits that would have otherwise been achieved. New technology, therefore, should be designed and developed taking into account its overall impact on the operations, and particularly on the mining sequence.

A review of the pertinent literature indicates that some of the current research in the areas of mechanization and mine automation does not recognize the significance of the mining sequence to underground hard-rock mining. If flat, long stope geometries such as those that created ground control problems in the past are necessary for the new continuous miners, their applicability will be very limited. The same type of objections can be raised about innovative rock fragmentation equipment that requires access to every piece of rock to be broken.

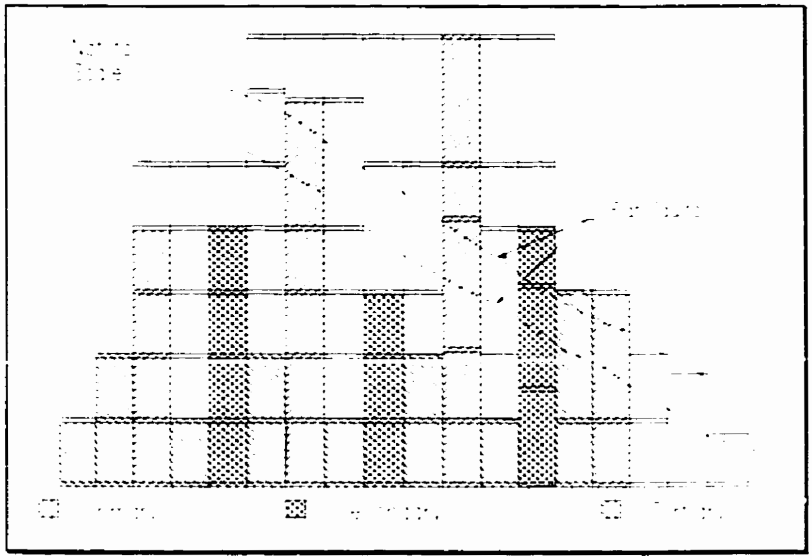

Figure 8: The 1-5-9 mining sequence at Fraser Mine (Potvin and Hudyma, 1989)

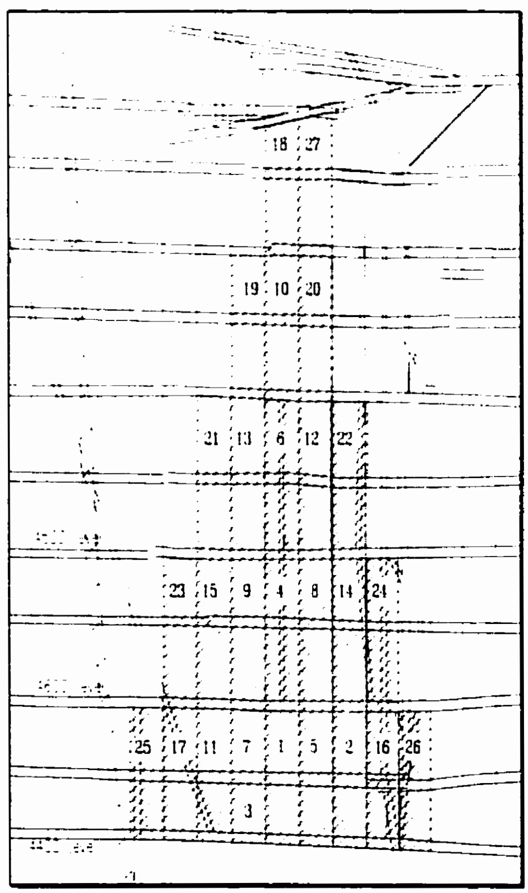

Figure 9: Mining sequence at the Golden Giant Mine (Pelley, 1994)

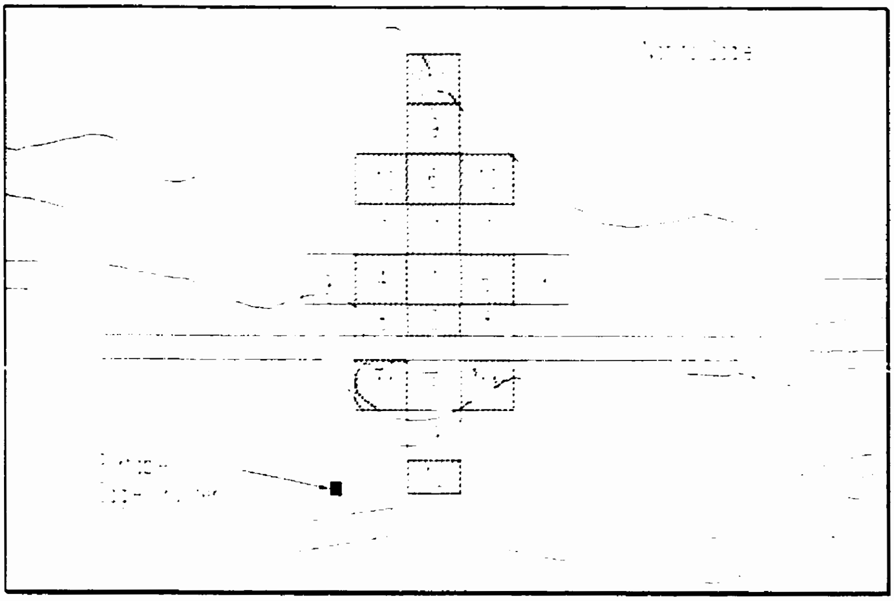

Figure 10: The deep mining sequence at Creighton Mine (Pelley, 1994)

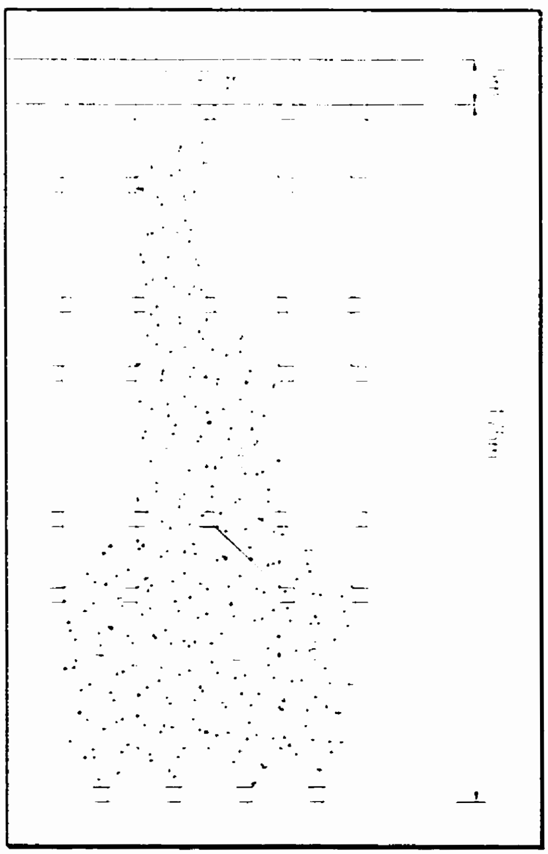

Figure 11: Kidd Creek extraction sequence, longitudinal section (McKay and Duke, 1983)

4. Constraints on Deep Underground Hard-Rock Mining

This Chapter analyzes the particular case of deep underground hard-rock mining using the concepts presented in the preceding chapters. The first five sections are devoted to the five factors identified as the most critical in this type of operations (see Section 1.4). The objective is to determine the impact of each of them on the mining strategy, focusing on aspects such as mine development, mining sequencing, capital and operating costs, and other important operational parameters. The final discussion of the strategic significance of these factors is left to Chapters 5 and 6. In Chapter 5, cost data from a real case study are used for their evaluation. A simulated case study is used in Chapter 6 to illustrate points that could not be analyzed and discussed with the *real* data.

4.1 Vertical Ore/Waste Transport

Deep mines pose serious problems for the design and operation of vertical ore and waste transport systems. The vertical distances involved coupled with difficult ground conditions and increased ventilation requirements demand a careful analysis and understanding of the overall operation before deciding on a specific system. Furthermore, the performance of such a system will depend on the actual design features that, in turn, are the result of technical and strategic considerations.

In general, there are two different methods of transporting ore and waste up to surface or the processing/shipping facilities: through vertical or near-vertical shafts and through inclined openings (inclined shafts and ramps). Skip hoisting, vertical conveyor, and hydraulic transport systems can be installed in vertical or near-vertical shafts, whereas trucks and conveyor belts are usually employed to transport broken muck through ramps. Regardless of the selected vertical transport system, the main issues that have to be addressed by its design include:

- production capacity;
- capital cost;
- operating cost; and,
- operational features.

4.1.1 Skip Hoisting Systems

Skip hoisting probably is the oldest and most common method of vertical ore/waste transport in underground metal mines. In fact, drawings and sketches found in Agricola's *De re metallica* (Agricola, 1950) show primitive hoisting devices that were (supposedly) in operation more than 440 years ago. There is a large body of literature that discusses the principles of skip hoisting systems and their application to the mining industry (e.g., International Conference on Hoisting of Men, Materials and Minerals, 1988; Walker, 1988; Edwards, 1992). A formal analysis of such systems is outside the scope of this thesis and, thus, they will be reviewed only from a design and operational perspective. The focus will be on the comparative advantages of skip hoisting, the identification of the design aspects that are strategically important, and the impact on the mining strategy of deep operations.

A hoisting system has five main components (Edwards, 1992; Edwards, 1988): hoist, conveyances, rope, headframe, and shaft. Only the hoist and the shaft have strategic significance and, not surprisingly, have the largest impact on the long-term operation of the system. Furthermore, their main features determine other important aspects of the production process such as flexibility and maximum production capacity, and are particularly critical to deep mines.[27] On the other hand, the conveyances, rope and headframe are designed primarily according to technical (and, of course, cost) considerations, which to a large extent depend on the main features of the hoist and shaft.

4.1.1.1 Hoists

Strictly speaking, the basic (i.e., purely technical) information required to select and size a hoist include: hoisting distances, production rate (usually, expressed in tonnes/hour), maximum load,

[27] A strategic approach would be to design the system taking into consideration the estimated future requirements of the operation, including hoisting from deeper areas or multiple levels, and increased production rates. However, the pressure to reduce the initial investment (which, of course, improves the NPV of the project at the evaluation stage) usually results in a system that only meets the demands of the original design. As noted by Henningson (1988) a reduction of 10 to 20% on the initial capital cost of a hoisting system may be the most expensive decision ever made. For instance, at Kidd Creek the capacity of the Number 3 shaft hoisting system limits the tonnage that can be provided from that area.

and type of guides. However, if the operation is expected to expand, deepen or otherwise be modified in the future, the following data should be considered also:

- surface plant layout;
- underground layout;
- features of the power distribution system available; and,
- ultimate use of the shaft.

The specification of both the surface plant and underground layouts requires tight interaction with hoist design, even if no further modifications to the system are expected. An iterative approach should be adopted and enforced, so that the final design meets the requirements of the entire operation. The last item (i.e., the shaft's ultimate use) is not easy to define since it demands a clearer understanding of the future prospects of the mine than is possible to obtain at the early stages of an underground project. This is particularly true in the case of the ore reserve estimates and changes in ground conditions at depth. Nonetheless, a careful analysis of the proposed operation can provide a sound estimate of the long-term hoisting requirements. Then, it is necessary to identify the major differences, if any, between the original design and the one that incorporates the features needed to sustain the expansion and/or deepening.

A hoist designed to accommodate operational changes that will **not** take place within the first 7 to 10 years of operation may not be "*optimum*" in the traditional sense. This is because it may not result in the lowest cost per ton of material hoisted during the first years of the operation. However, it will be able to provide continuous service for the entire productive life of the mine.

Edwards (1992) provided a complete list of the various types of hoists. A comparative analysis of the main features of the various hoist types, as applied to deep mining, has been summarized as follows (see Fairbairn, 1988; and Edwards, 1992):

- Drum-type hoists (single and double drum)
  - ➢ Ideal for hoisting high payloads from relatively shallow depths.
  - ➢ Suited for hoisting from multiple levels (mainly double drum hoists).
  - ➢ Limited by the ultimate strength of the single rope used to carry the load.
  - ➢ More expensive than friction hoists, both to install and operate.

➢ The associated peak and average power demand is higher.

- Multiple-drum, Blair-type hoists, which have two ropes attached to each conveyance, can be used to hoist from depths that exceed that of single-rope and friction hoists.
- Friction hoists

  ➢ Due to the use of multiple ropes, they have a smaller drum diameter and lower initial capital cost. As the rope size increases, however, the drum diameter increases rapidly.

  ➢ For a depth range of about 500 to 1500 metres, multi-rope friction hoists have a higher production rate, mainly because they can carry heavier loads.

  ➢ For the same production capacity, the peak power demand is lower than that of a drum-type hoist. They also have a lower overall power consumption.

  ➢ Due to their design, there is a limitation on the ultimate hoisting depth.

4.1.1.2 Shafts

It has been proposed that the design and specification of a shaft requires the previous identification and description of the following nine items (Edwards, 1988):

1. purpose
2. size
3. configuration
4. depth and location
5. inclination

Strategically significant

6. ground support
7. internal arrangement
8. collar
9. shaft sinking

As indicated in the list, only the first five items have some strategic importance. The other four are chiefly determined either by the features of the orebody and surrounding rock mass (e.g., ground support), by the selected hoisting system (e.g., internal shaft arrangement and shaft collar), or by the above-mentioned five strategic items.

In these times of tight budgets and competitive metal markets, there is increasing pressure to design *multi-purpose* shafts. Such shafts can be used for exploration, services (personnel, equipment, and supplies handling), ventilation (either downcast or upcast flow), production (both ore and waste transport) and escape or emergency use. In deep mines, however, it may not be possible to design and equip a shaft that can satisfy all these requirements. While most shafts become critical components of the ventilation circuit of a mine, their use for the transport of personnel, supplies and pieces of equipment will be limited by the ground conditions and cost considerations. In fact, a small excavation with circular cross-section may be the best option to withstand high horizontal stresses and, at the same time, reduce both construction time and cost through the use of a boring machine. However, this may not leave enough space to accommodate an additional service hoist. If a new blind shaft is considered for an expansion into deeper areas (or if a new shaft is sunk from surface in an existing mine), an additional internal ramp may be needed for personnel, materials, and equipment transport. A skip-cage combination would limit the production rate due to the reduced ore/waste hoisting hours. If an existing shaft is being deepened, the options depend on the current shaft configuration and uses.

The *size* of a shaft must be determined taking into account the fact that in most mines it is a critical component of the mine ventilation system, representing the largest single source of frictional resistance. Such a resistance is a function of the length of the shaft, its area and perimeter. The cross-sectional area and perimeter must balance the need to meet production requirements, the changing ground conditions (usually deteriorating with depth), shaft sinking method, and the type of support to be employed.

The most common shaft *configurations* are rectangular and circular. Elliptical shafts are not as popular, but they could become attractive options if efficient continuous excavating machines (that can easily achieve this shape) are developed for this purpose. The comparative advantages and disadvantages of each configuration are as follows:

- Rectangular shaft
  - ➢ Makes more efficient use of available space.

- ➢ Not likely to be used at depth, due to poor performance under stress.[28]
- Circular shaft
  - ➢ More resistant to lateral ground pressure.
  - ➢ Offers the least resistance to ventilation air flow.[29]
- Elliptical shaft
  - ➢ Offers the same advantages than the circular shaft.
  - ➢ This is the most efficient shape, and taking advantage of the space could be improved if the shape of conveyances were redesigned.
  - ➢ Resistance to lateral ground pressure can be further improved if the elliptical shape is properly oriented (according to the principal horizontal stresses).
  - ➢ More difficult to sink, either conventionally or with continuous excavators.

The *depth* and location of a shaft are determined according to the:

1. geometric features of the orebody, including its location, dip and horizontal extent;
2. location of underground ore and waste handling facilities;
3. mine surface layout;
4. stability of the shaft pillar; and,
5. future planned expansion/deepening of the shaft.

The mine operator has no control over the geometric features of the orebody. However, these constitute the most important factors in determining the optimum depth and location of the shaft. On the other hand, the location and main features of the mine surface layout as well as those of the underground ore and waste handling facilities should be established so that the mine design and operation becomes optimized. The stability of the shaft pillar and shaft bottom depend on the quality of the geological and geotechnical information available at the design stage. Problems such as those encountered at the Hemlo camp (Pelley, 1994) should be avoided at all times. An adequate pillar and a stable shaft bottom also facilitate the future deepening of the operation.

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28 However, a rectangular shaft was the choice at Macassa #3 shaft, and was used to depth at the Lakeshore Mine.

29 This also depends on type of lining/support (concrete, shotcrete), and the internal arrangement of the shaft, including the shape and spacing of sets or buntons and guides.

4.1.1.3 Objectives of Hoisting System Design

As in the case of the mining strategy (discussed in Section 2.5.1), there are several *operational* objectives that must be achieved (or *optimized*) by the hoist design. They are:

1. minimize initial capital investment;
2. minimize operating costs;
3. minimize construction time;
4. maximize reliability;
5. maximize efficiency;
6. maximize flexibility; and,
7. facilitate mine development and the elaboration and execution of the production plan.

The conflicting nature of these objectives is obvious. For instance, it is virtually impossible to produce a hoisting system design that minimizes initial investment and operating costs and, simultaneously, maximizes flexibility. The additional flexibility would certainly result in additional expenditures and some operational inefficiencies. Furthermore, the design problem is complicated by the fact that, with the exception of the first three items of the list, it is very difficult to evaluate quantitatively the impact of achieving such objectives. Reliability, efficiency, flexibility, and optimization of mine development and production are all concepts that cannot be assigned a single hard number and have to be determined based on experience and judgement. The existence of a mining strategy would certainly help in deciding the importance of each objective and in focusing the design process.

In new deep mining projects, additional issues are the long period of time required for shaft sinking and the significantly higher hoisting costs involved (on a per-tonne basis). In such cases, capital cost may not be the critical parameter to be minimized. In fact, a more intensive shaft sinking stage, or a larger or more efficient shaft/hoist configuration may, in spite of higher initial capital cost, produce the highest return on the investment through a shorter construction phase and/or lower operating costs.

4.1.1.4 Summary and Discussion

From a strategic point of view, the main advantages of selecting a hoisting system for the vertical transport of ore and waste are the following:

- hoisting systems have been used in the metal mining industry for several decades and are very familiar to both operating and maintenance personnel (i.e., it is a *proven technology*);
- it can be easily adapted to the particular conditions of the job (i.e., hoist type, shaft shape, size and inclination, etc.);
- it results in relatively low operating costs in terms of \$/tonne of material hoisted; and,
- it is not restricted to ore and waste transport only and is normally used for personnel, supplies and equipment handling also.

On the other hand, hoisting systems have some drawbacks, particularly when a deep operation is in mind:

- they lack flexibility: once the system is designed, built and installed, very few changes can be made to modify the hoisting operation without incurring large and time-consuming investments;
- deepening an existing hoist (a particular case of the previous point) becomes extremely expensive due to the loss in production and the strict safety regulations regarding this practice (specifications of the bulkhead and operating guidelines). It is also time-consuming and inevitably interferes with the on-going operation; and,
- peak power demands are sometimes high and require a surplus of installed power capacity that is not used frequently.

4.1.2 Vertical Conveyors

Although they have been used for decades in the coal mining industry, vertical conveyors are still extremely rare in hard-rock metal mines. At least two manufacturers of bulk transportation equipment have been trying for some time to introduce their technologies into underground mining projects: *Trellex AB* (Paelke, 1988) and *Continental Conveyor & Equipment Company* (Dos Santos, 1993b). Trellex sells the *Flexowell* system, which is designed to transport materials vertically. Continental manufactures *HACs*, High Angle Conveyors, which can be designed to operate at 90° angles, but are mainly used at shallower inclinations (around 40°). There are many other systems that can be adapted for mining applications as well (Roberts, 1994).

4.1.2.1 The *Flexowell®* System

Flexowell belts consist of a cross-rigidified belt with flexible corrugated sidewalls. The added flexibility of the sidewalls allows them to run over the head and tail pulleys, and to be guided through any angle. Cross cleats spanning between the corrugated sidewalls form a "pocket" that actually carries the material. The base belt is reinforced with either a fabric or a metal woven material (depending on the characteristics of the conveyed material) in order to provide cross stabilization. Sidewall height is determined according to the particle size of the material and required production capacity.

Several vertical and sub-vertical conveying systems have been developed based on the Flexowell conveyor belt (Paelke, 1987). The Flexowell belt can carry material at any angle up to 90° (vertical). When the conveying angle exceeds 70°, no intermediate structure is necessary between the head and tail stations. The main advantages of Flexowell systems include (Paelke, 1982):

- require little space for installation (relatively speaking);
- provide a continuous transportation process, from loading to elevation and discharge;
- fairly economic and reliable: a single motor is employed;
- power consumption is low, roughly equal to the work done by lifting the material only;
- can be integrated with existing installations and facilities;
- large elevating heights can be achieved in single lifts; and,
- flexible and versatile, these systems are particularly suitable for underground mining, where extensive modifications and adaptations are usually required.

A Flexowell system in operation at the New York Water Tunnel project since August 1994 is North America's highest single lift vertical conveyor (172.0 metres). It is designed to lift 500 tonnes/hour of granite/limestone with an average density of 1.81 tonnes/m^3 and maximum size of 150 mm (Paelke, 1994). An even higher Flexowell system (208-metre high) will be later installed in the same project (Paelke, 1995). A Flexowell system at Placer Dome's Porgera Mine in Papua

New Guinea is the world's highest single lift vertical conveyor (203.0 m). Installed in 1992 inside a 3-metre finished diameter raise-bored shaft, the system lifts gold ore with a density of 1.65 tonnes/m^3 and maximum size of 200 mm at a rate of 350 tonnes/hour. Two similar systems for existing mines are being designed: one at a depth of 400 m and with a conveying rate of 1,360 tonnes/hour and the other at 2,720 tonnes/hour from a depth of 243 m (Paelke, 1993).

4.1.2.2 The *HAC*® System

HAC systems are improved versions of the "sandwich belt" concept (Dos Santos and Frizzell, 1983). This approach to vertical and sub-vertical conveying uses two ordinary rubber belts that "sandwich" the conveyed material. Additional force applied on the belt provides pressure to the material to develop enough friction at the material-to-belt and material-to-material interfaces to prevent sliding back.

Main advantages of HACs include (Dos Santos, 1993b):

- simple approach: conventional conveyor hardware is employed;
- very high capacity: high conveying speeds can be achieved;
- high lifts and conveying angles: single lifts beyond 300 metres (much higher with steel cord belts) and angles up to 90° are possible;
- flexible design and operation: the system can be easily adapted to changes in operational requirements;
- easy maintenance and repairs: smooth surface belts facilitate cleaning and provide easy access for repairs; and,
- the operation is virtually spillage-free.

The world's highest single lift high angle conveyor (a HAC with 174.8 m of vertical lift at an inclination of 41°) operates at Island Creep Corp., Virginia handling coal refuse at a rate of 450 tonnes/hour. The highest single vertical lift installation of this type is at Turris Coal Co., Illinois where it carries 1,360 tonnes of coal per hour (Dos Santos, 1993b). Other applications include continuous ship loading and unloading, elevation from underground hoppers and tunnels, midstream transfer and blending, preparation plants, etc. (Dos Santos, 1987).

4.1.2.3 Vertical Conveyors vs. Conventional Hoisting Systems

Currently, there is only one vertical conveyor system in operation at an underground hard-rock mine (Paelke, 1993). Thus, a comparison between such a system and a standard skip-hoist system would have to be carried out at the theoretical level. Furthermore, the conclusions drawn from such an exercise would be only of a qualitative nature, since there is not enough actual data to support "*first principles*" calculations.[30]

A somewhat detailed comparison between a sandwich belt system and skip hoisting was carried out by the manufacturer of HACs (Dos Santos, 1993a). The study is particularly relevant to the present project since it is supposed to demonstrate the applicability of HACs to deep nickel/copper underground hard-rock mining in North America. The case study focuses on a mine producing 9,000 tonne/day of nickel/copper ore having a bulk density of 2,000 kg/m^3 and a maximum particle size of 17.8 cm. Ore would have to be elevated from a depth of 1,402 m vertically to surface. A conventional skip hoist system is compared with two variations of HAC systems. The author of the paper indicated that, based on system availability, utilization, and "*experience*", the skip hoist system was designed at a production rate of 545 tonne/hour, whereas the HAC systems were designed to elevate ore at a rate of 635 tonne/hour. Apparently, vertical conveyor systems have lower availability than skip hoists.

The design of the conventional skip hoist system called for a 7.0-metre diameter, concrete-lined shaft, sunk to a depth of 1,445 metres. A 4.9-m diameter, four-rope, 6,000-hp friction hoist was equipped with two 20-tonne skips designed to travel at 1,000 m/min. The system also included a 3.5-m diameter, four-rope, 1,150-hp service cage hoist, and an auxiliary 2.5-m diameter, 450-hp cage hoist. The first HAC system was also installed in a 7.0-metre diameter concrete-lined shaft, sunk to a depth of 1,396 m (HACs receive the material directly from conveyors and require no bunkering). It consisted of eight 165-m main HACs and seven connecting HACs, all of them

[30] In the mining industry, the estimation of operating costs using "*first principles*" is equivalent to what Thuesen and Fabrycky (1984) call "*cost estimating by engineering procedures*" and Gentry and O'Neil (1984) call "*detailed cost breakdown method*". As the expression implies, it involves using primary data (engineering drawings, cycle times, labour schedules and rates, materials and supplies consumption rates, etc.) to produce detailed estimates of every major operating cost item. These are then added into the operating cost for the entire project. This method can result in very accurate estimates, but their quality is heavily dependent on that of the basic data and assumptions made. This is true even when such estimates are based on historical data.

running at 152 m/min, a speed that, supposedly, extends the life of the components. Total drive power of the HACs was 5,360 hp. The service cage and auxiliary cage were in this case similar to those of the skip hoist system. The second HAC system was installed in a 5.0-m, concrete-lined shaft sunk to a depth of 1,396 m. Four 3.7-m diameter, 160-m long raises were also required to provide HAC to HAC transfer. The service cage and auxiliary cage were also similar to those of the skip hoist system.

Total excavation, construction, and capital costs were US $ 50.0 million for the skip-hoist system, US $ 50.8 million for the first HAC system, and US $ 41.8 million for the second HAC. Total operating and maintenance costs were US $ 1.97 million per year for the skip-hoist system, and US $ 1.78 million per year and US $ 1.73 million per year for the first and second HAC systems, respectively. The author concluded that the simple economic analysis carried out demonstrated the cost advantages of HACs. He also hinted that economies of scale could be quite significant (Dos Santos, 1993a). Based on the above figures, the use of the second HAC system would result in a 16.4% reduction on total capital costs and a 12.2% reduction on operating costs. However, it is believed that they alone would not justify the switch to a technology that is unproven, and has the potential of intrinsic efficiency and reliability problems.

4.1.2.4 Summary and Discussion

The main advantages of vertical conveyors, regardless of the lift system, are their lower capital costs and the ability to hoist material continuously. Indeed, for a depth of about 500 m, the capital cost of a vertical conveyor is about 50% of that of an equivalent traditional skip hoist (Moser, 1990). However, because these are relatively new applications for the technologies, there are not reliable figures for the corresponding operating costs. Skip-hoisting systems require single loading points (typically, at the bottom of the shaft) to be efficient, whereas HACs have the ability to load material at several levels. This not only reduces the need to have a system of orepasses to bring the material to the loading point, but also could lead to significantly lower initial capital requirements (the shaft would be deepened as the mine progresses downward and additional HACs would be added as required). Finally, vertical conveyors can be operated in a continuous manner, thus providing the capability to automate the loading and discharging stages.

If they were available, continuous hard-rock mining machines would be the ideal matches for such systems.

Drawbacks of vertical conveyors include the limited size of the material that can be hoisted, the fact that they can be used only to transport broken rock (not personnel or supplies), and their *unproven* nature.

4.1.3 Hydraulic Transport

As discussed by Walker (1988, pp. 361-362), there are three main reasons why hydraulic transportation of solids should be seriously considered by deep mining operators:

- the sheer potential productivity of a hydraulic transport system;
- hydraulic systems are very flexible and relatively simple to install and maintain; and,
- although they do have limitations in terms of maximum particle size and depth of operation, these are less restrictive than that of existing hoisting systems, and could be circumvented easily with proper mine design.

Hydraulic hoisting was first attempted in Canada by Falconbridge in the 1970's (Bekkers, 1997, p. 44). Crushed ore with a SG of 3.2 and maximum particle size of 76 mm was introduced into a water stream, and the 16% by volume slurry was pumped through a 25.4-cm pipeline over an elevation of 730 m. Operational problems plagued the system and it was not developed further. The concept was investigated again during the 1980's but results have not been published.

Positive displacement pumps have been used in the mining industry for handling sludge and concrete aggregates for many years. In the underground asbestos mines in Quebec, up to 7,650 m^3 of cement were placed annually using Schwing pumps. The pumps were used to place low slump material (material of up to 68% solids) up to 457 m horizontally and 52 m vertically.

In the 1980's, Siemag, a German manufacturer of hoisting and fluid transportation systems, developed the *three-chamber pipe feeder* system for hydraulic transportation of coal and ore (Kortenbusch, 1982). The general capabilities of the system are as follows (Siemag, 1995):

- ➢ system capacities of up to 500 tonne/hour;

- vertical lifts of up to 1,500 metres without the use of intermediate pump stations;
- horizontal distances of up to 20,000 metres;
- grain size of up to 100 mm and solids with SG of up to 4.5; and,
- hoisting pressures of up to 250 bar.

An installation in a coal mine reportedly hoists raw coal with a maximum grain size of 30 mm from a depth of 507 metres at a rate of 50 tonne/hour. Total installed power in this case is 680 kW and the hoisting pressure approximately 70 bar. Another system is used as a heat exchanger at a 1,000-m deep mine, where the circulation of 800 m^3/h of water results in 15 MW of cooling capacity (Siemag, 1995). Worsley (1990) described a similar system to be implemented at a South African gold mine, and suggested the operation could take advantage of water "losses" in the system in order to hydraulically hoist about 1,000 tonnes of ore per day. Hindmarch (1990) discussed the use of three-chamber pipe feeder systems to integrate air conditioning, backfill, water supply, dirty water pumping, and ore hoisting in deep mines.

Initial capital investments are in most cases lower than those required by traditional hoisting methods. Furthermore, hydraulic transport can generate significant operating cost savings in the areas of refrigeration, muck hoisting, backfill distribution, and pumping of clean and dirty water. It is important to note that the entire operation can benefit also from a continuous and integrated transport system that runs with minimum human intervention and lends itself to automation.

Hydraulic transportation has the potential of solving many of the economic and operational problems associated with deep mines. However, several practical issues must be addressed before it can completely replace production rope-hoisting systems. For instance, it would require the use of underground secondary crushers to achieve the required degree of fragmentation, particularly in bulk mines.[31] This possibility, unfortunately, has not received much attention, although its economic justification would be enhanced significantly by carrying out some degree of ore

[31] At the Cigar Lake uranium mine in Saskatchewan, plans call for the ore to be excavated using boring machines and fed into grinding mills before being pumped to the surface. It could be argued that neither the strength of uranium ore nor the daily production rate at Cigar Lake compare to that of gold or base metal mines. Nonetheless, a similar mining approach, i.e., one that results in "stopes" with high vertical to horizontal dimension ratios (see Section 3.4) and finely broken muck, could provide considerable advantages if successfully designed for deep hard-rock mining.

selection or concentration before hoisting.[32] Similarly, serious coordination problems could arise if hydraulic transport were coupled with traditional drill-blast-muck mining systems. This would be particularly true if an installation were to be used for more than one function (such as the integrated backfill and hydrohoisting system proposed by Hindmarch, 1990).

4.1.4 Ramps

Depending on the vertical distances involved, trucking ore and waste up a ramp to surface (or to the processing and/or shipping facility) can be a costly option. Furthermore, very few deep mines can afford to drive ramps from surface to the areas under current exploitation.[33] On the other hand, ramps provide flexibility to the operation, can be used to transport supplies and personnel, and are attractive for expansions into deeper areas and accessing/connecting working levels internally. They have been described as the lowest cost solution for ore transport to a depth of about 500 m (Moser, 1990) and were used successfully to a depth of 425 metres at the Trout Lake Mine in Manitoba. This option is presently being considered for the approximately 300-metre deepening project at the Creighton Mine in Sudbury.

4.1.5 Conveyor Belts

In deep mines, conveyor belts have limited applicability for the vertical transportation of ore and waste material. This is due to their shallow inclination (in normal conditions, the maximum inclination of standard conveyor belts is about 22°), which makes driving long access openings necessary in order to reach the discharge area. They are rarely used to transport materials over

[32] A major disadvantage of underground comminution processes, (and probably an explanation for the status quo in this regard) is the large amounts of heat that they generate. In fact, Bond (1985) noted that the mechanical efficiency of comminution is close to 1.0%, and that almost all the energy input required to break the rock appears as heat. This would certainly be highly undesirable in deep mines, which already have serious problems dealing with heat generated by the rock mass, diesel and electric equipment, underground water, etc. (see Section 4.3).

[33] The author is aware of only two such exceptions in Canada. At the Kidd Creek Mine, Falconbridge Limited, the cash flow provided by the open pit, which exploited the upper part of the orebody, made it possible to construct a ramp system to the bottom of the underground reserves. Subsequent extensions of such a ramp have allowed access to the bottom of the current mining areas, at the 6200-foot level. The ramp is used mostly for transporting materials and, occasionally, personnel. Ore and waste are skip-hoisted to surface through various vertical shafts. Another exception is the Williams Mine, Williams Operating Corporation, which has an internal ramp that accesses all levels for movement of materials and personnel, and extends to the bottom of the active mining areas. Although not directly connected to surface, the ramp nonetheless provides an escape route by joining to the Golden Giant mine internal ramp and then to the original ramps in the upper "A" zone, which eventually lead to a surface portal.

large vertical distances. Therefore, they do not offer any real advantage (strategic or otherwise) over the other systems discussed in this section.

4.2 Horizontal Ore/Waste Transport

This activity is defined as getting the ore from the drawpoint to the underground crusher. Historically, this task was accomplished by transporting on individual levels and dumping into an orepass feeding a crusher located near the bottom of the mine. The presence of a crusher on an intermediate level usually reflected the existence of an initial stage of mining. This was a very labour-intensive method, although level tramming crews were also responsible for distribution of supplies to the stopes. It is still carried out at certain low tonnage operations and is still employed at the Campbell Mine for the ore above 27 level, with trains loaded mainly from chutes.

At larger operations, the trend has been to install a single large-tonnage haulage system near the bottom of the mine or beneath several levels in one area of the mine. The system is fed by orepasses located regularly along the orebody to reduce the distance between drawpoints and orepasses (on the mucking levels). The lateral transport to the shaft has generally been provided by tracked systems such as those in operation at Kidd Creek and David Bell mines. One exception is the Williams Mine, in Hemlo which has four orepasses located along the orebody, each serviced by a fixed Eagle crusher installation that feeds a conveyor to transport the material to a skip loading bin. Trucks could be used also on the main transport levels.

4.2.1 LHD-Based Systems

In most underground hard-rock mines, the ore is mucked from the drawpoint to the orepasses using remote-controlled **Load-Haul-Dump** (LHD) units. Recent research efforts have concentrated on reducing mucking costs by tele-operating mobile equipment and providing the possibility of one operator being able to supervise simultaneously more than one piece of mobile transport equipment (Poole et al., 1998). This would certainly increase productivity and reduce the labour component of the operating cost. However, LHDs are not very efficient for transporting long distances (Walker, 1988, p. 245). Indeed, LHDs have weight-to-payload ratios of almost 3:1, thus requiring a high-energy input per tonne of material transported (see Figure 12). It could be argued that the efforts spent trying to *optimize* the operation of LHDs and reduce the associated labour cost could be channelled to projects focused on alternative mucking/lateral transport systems. Such systems, still at the research stage, have the potential of not only lowering the mucking/transport cost, but also radically changing the design of hard-rock mines and making them more competitive.

An additional disadvantage of LHDs is the heat and toxic fumes produced by their diesel engines. This issue could be somewhat reduced by switching to electric scooptrams. Electric LHDs, however, would require the installation of high-maintenance trolley lines or the use of cable reels, drastically reducing their flexibility (one of the main reasons to use LHDs in the first place).

In deep operations, the difficulty in maintaining orepasses (which can collapse very rapidly in highly stressed environments) reduces the efficiency and increases the operating cost of LHD-based lateral transport systems. This is because, if ground control problems result in mine designs with widely spaced orepasses, or if they have to be located outside of the stresses areas adjacent to the orebody, haulage distances will become longer. In extreme cases, the reduction of labour costs that could be achieved through tele-operation (or automation) of LHDs may only compensate the increase in production cost resulting from longer haulage distances and/or complex orepass designs. Future overall costs in such cases may be comparable to that of present operations, i.e., no real improvement in the competitive position may be realized.

The prospect of increased haulage distances and hotter mining environments, coupled with the inherent inefficiency of LHDs, should focus the attention of industry and research centres on radically modifying the lateral transport system process. Considerable research effort has already concentrated on converting it into a continuous process. Existing continuous loaders, however, have had difficulties in dealing with the typical fragmentation of the muck found in stope drawpoints of bulk mines. Their productivity can be severely affected by large boulders that require secondary breakage. Design problems have included the mismatch of the capacities of the continuous mining system components. For instance, the loader and conveyor may have much higher throughput capacities than the crusher, which then becomes the bottleneck of the entire mucking/transport system. Low system availability is also common due to the inherent complexity of the design and operation of such systems.[34]

4.2.2 General Discussion - Continuous Systems

It is believed that the greatest potential for a comprehensive solution to the horizontal transport problem in deep mines is in continuous loading and haulage systems. Their main advantages would include:

- Low cost operation (in $/tonne of material transported) due to reduced power requirements, better mechanical efficiency (lower operating weight to payload ratios), and lower operating labour cost.
- Increased productivity of single stopes, since such systems can transport more tonnes per hour than LHD-based ones.
- Improved ground control which, in turn, results in additional direct operating cost reductions. This would come from two main sources: first, the mucking phase of the mining cycle would be shortened, allowing prompt placement of backfill, hopefully before serious ground movement takes place. Second, the dimensions of the openings required to transport, install, and operate these systems are usually smaller than that of the haulageways used by LHDs and mine trucks. Therefore, the former would not only be cheaper to excavate, support, and maintain, but also would cause less disturbance to the rockmass.
- Reduced ventilation requirements due to the use of smaller, more efficient electric motors, and the elimination of diesel-powered equipment for mucking and haulage.

On the other hand, only a slow migration to continuous systems should be expected in the future, mostly because of the following:

- High capital cost. In addition, this would probably preclude their use on individual levels (the desired configuration, from an operational perspective) except in very thick and/or rich orebodies.[35]

[34] The above-mentioned research efforts resulted in the development of continuous loading equipment, including the oscilip loader, the Voest Alpine loader, and the Atlas Copco Haagloader. In most cases, though, trucks were loaded by these devices for transport to the orepasses, instead of taking advantage of the continuous nature of the loading process and switching to a more efficient (and cheaper) continuous transport system. Inco developed the mobile, low profile Eagle crusher to reduce the size of the muck in order to make it more suitable for conveying. Mobile belt benders were developed also to accommodate the routing to orepasses (Pelley, 1996).

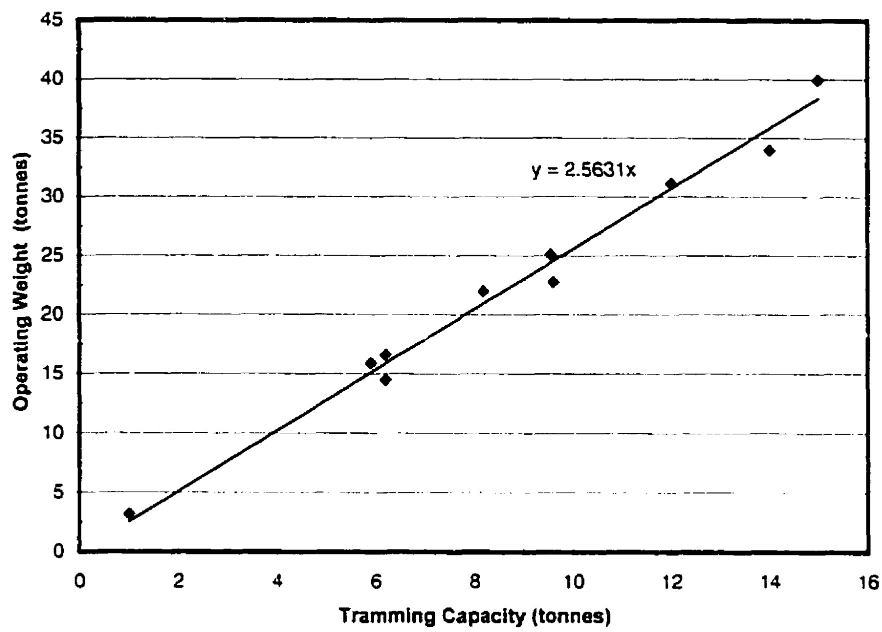

Figure 12: Tramming capacity vs. operating weight - Tamrock LHDs

- For continuous systems to be able to operate in bulk mining environments, muck fragmentation would have to be significantly improved. Otherwise, equipment utilization would be very low, negating any potential benefits from the new technology.
- The inflexibility of the resulting mine designs and production plans, which imposes serious constraints on the mining strategy. Considerable percentages of production capacity can be tied up to each system, jeopardizing the ability of the operation to meet production requirements in case of equipment failure, and making grade control very difficult. Although the systems previously tested in Canadian mines were described as "*mobile*", they were not nearly as mobile as LHDs and mine trucks. Thus, the response time to unexpected operating conditions (including the need to move to a different stope or level) can be very long.
- The full benefit from the introduction of this technology can be realized only in new mines (or sections of a mine) that are designed and planned according to the operating requirements of continuous loading/transport systems.

[35] An alternate solution that could justify thee use of continuous loaders to muck individual stopes would be to increase stope heights. This could have the additional advantage of facilitating ground control practices, as seen at the Kidd Creek mine (see Figure 11). Unless it is carried out for limited tonnages, the practice of excavating orepasses in backfilled stopes so that the continuous loaders do not have to be moved between levels must be avoided, as it usually generates dilution and ground instability.

4.3 Ventilation

Not surprisingly, the industrial sponsors of this research project noted that ventilation had already become a major issue at their deep operations. The problem, however, is not new. Several decades ago, Spalding (1949, p. 238) commented about ventilation issues in deep mining environments in no uncertain terms:

> *"Of all the factors which affect mining operations, high rock temperature is the one most often likely to limit the depth to which those operations can be extended. The science of ventilation is therefore rapidly becoming the most important branch of deep mining. The old rough-and-ready methods of ventilation control no longer afford sufficiently accurate results, and efficiency in ventilation is becoming, if possible, more important than any other branch of mining."*

Similarly, in discussing the importance of considering ventilation as part of an air conditioning system,[36] Hartman et al. (1982, p. 4) pointed out the significance of increased depth of mining:

> *"Although today's technology is vastly improved, environmental challenges underground still abound. Depth, the worst, sets the ultimate limits to rock pressure and temperature in all mining. Not only do rock pressures rise inexorably with depth, but temperatures also, with subsequent deterioration of the atmosphere. ...*
>
> *At great depths, ventilation requirements and costs eventually climb to unsustainable levels. To preserve mine atmospheric quality under these intense heat conditions, ventilation at great depths must be supplemented by air conditioning."*

It is evident, therefore, that the cost of ventilating mines significantly increases with depth. Two main factors play critical roles in both the cost and effectiveness of ventilation in deep mines: the airways and heat.

[36] Strictly speaking, the title of this section should be *Air Conditioning* or *Ventilation and Air Conditioning*. In fact, as noted by Hartman et al. (1982, p.3), ventilation is concerned with the control of air movement, its amount, and direction. Therefore, it is only one of the processes of air conditioning, which deals with the simultaneous control of the quality, quantity, temperature, and humidity of the air. However, in order to follow terminology that is common in Canadian mines, the term *ventilation* will be used in the remainder of this thesis to refer to both ventilation and air conditioning issues.

4.3.1 Airway Design

Power must be supplied to air in order for it to overcome the head losses that take place in the airways and flow. Power is proportional to both volume of air (*quantity*) and head loss. In turn, head loss is proportional to the square power of volume of air. Thus, the quantity of air should be minimized to reduce power requirements and ventilation cost. This involves proper determination of the ventilation needs of every area of the mine as well as adequate leakage control.

Hartman et al. (1997, pp. 432-451) noted that head losses depend on four airway characteristics: size and shape, surface roughness, length, and shock loss. Unfortunately, with the exception of the main ventilation airways (i.e., ventilation shafts and adits), which are designed with the specific purpose of providing efficient, low cost ventilation, little thought is usually given to the ventilation function of most underground openings. An effort should be made at the design and planning stages to strike a balance between the desirable operational features of a particular excavation (such as a ramp with a flat bottom) and the need to optimize its ventilation characteristics. From a ventilation standpoint, an ideal airway would have the largest feasible hydraulic radius (i.e., its cross-section should be as circular as possible), the shortest length, and would be as straight and continuous (i.e., without changes in shape and/or area) as possible.

A strategic approach to excavation design would consider the impact of each opening on the entire ventilation network. The network can be made more efficient, and the corresponding ventilation cost reduced, by minimizing the total length of the airways, reducing the number of airways, and rearranging the existing airways (Hartman et al., 1997, p. 451). These are the basic tools that the operator may use to improve ventilation.

4.3.2 Heat

The main potential sources of heat in underground mines are (Hartman et al., 1982): autocompression, wall rock heat, underground water, machinery and lights, human metabolism, oxidation, blasting, rock movement, and pipelines. The first four are major sources and capable of creating intolerable environmental conditions. Figure 13 shows the average heat sources of seven "*hot*" North American underground mines. Figure 14 presents the heat sources at Inco's

Creighton Mine in 1993. It can be seen that the relative importance of the sources varies widely. In general, the impact of a source will depend on:

- location of the mine (elevation, latitude, weather, etc.);
- depth of mining;
- productivity and efficiency of equipment (the more productive and efficient the equipment, the less heat is generated for a given production rate);
- speed of mine development (faster rates will result in more heat added to the mine environment over shorter periods, with high heat flows at the working faces); and,
- excavation size (larger rock walls will result in faster rates of heat exchange).

4.3.3 Air Conditioning

The need to use air conditioning in underground mines can arise as a result of rock temperature or autocompression. If the *critical rock temperature* of 40.6°C is reached, air conditioning becomes essential, particularly in very hot and poorly ventilated areas of the mine. Such a temperature can be reached at depths considered as shallow by Canadian standards: at 400 m in Tintic, Utah; at 600 m in Superior, Arizona; and at 800 m at Broken Hill, Australia (Hartman et al., 1997, p. 591).

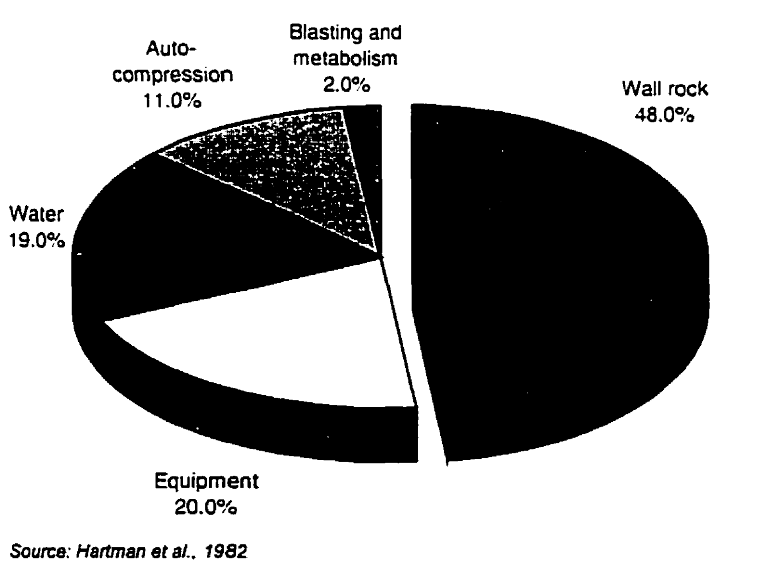

Figure 13: Average heat sources for seven *hot* North American mines

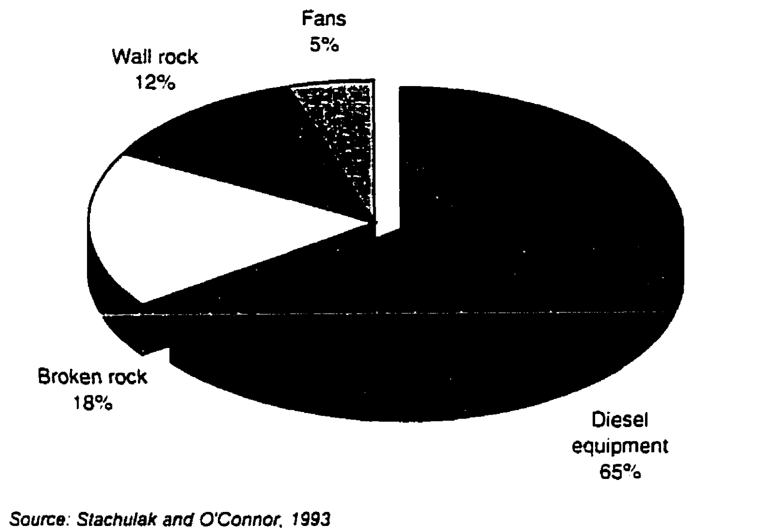

Figure 14: Heat sources at Creighton Mine in 1993

A *critical depth* is determined by the effects of autocompression.[37] At such a depth, ventilation air, heated due to autocompression, has no remaining cooling potential and air conditioning is required. The influence of autocompression is relentless and compelling (Hartman et al., 1997).

Regardless of the relative significance of the other heat sources which, to a certain extent, can be manipulated and controlled by the mine operator, rock temperature and/or autocompression will eventually dictate the need for air conditioning. Figure 15 presents the average heat-removal mechanisms for the "*hot*" mines shown in Figure 13. It can be seen that air conditioning alone removed 25% of the total amount of heat generated in the working places of such mines.

As illustrated by the use of water for air conditioning purposes in deep South African mines, the approach to the problem must be comprehensive.[38] In fact, chilled service water not only can be

[37] As ventilation enters the mine and flows downward, it is compressed and heated. The temperature of the air increases due to potential energy being converted to thermal energy. Usually, such a temperature increase cannot be determined accurately because of the non-adiabatic airflow (Hartman et al., 1997).

[38] Cool water can be used as chilled water service and/or distributed to heat exchangers (Ramsden, et al., 1990). In the former case, there is a dual cooling effect. First, heat is transferred from the ventilation air to the chilled water in the pipe network (usually in the intake airways). Second, depending on the water temperature, there is additional heat exchange at the face, where water is used to spray the muck and the newly exposed wall rock. In the latter case, chilled water does not enter the workings and the heat exchange takes place in the air conditioning unit.

used for refrigeration, but also to power hydraulic drills (Ashmole and Du Plessis, 1990), cut slots for mechanical excavation (Fenn and Marlowe, 1990), and hydrohoisting (Hindmarch. 1990).

4.3.4 Discussion

Unless significant changes are made to the way in which underground mines are designed and operated, the cost of providing them with adequate ventilation will certainly increase in both relative and absolute terms. This is due to the ever-increasing head losses (i.e., longer airways) and air conditioning requirements (i.e., more heat to be removed from working places) associated with deep mines. The importance of addressing both issues at the mine design and planning stages cannot be overemphasized. Largely, this is caused by the complex interactions between a number of design and planning aspects, such as:

- excavation design, that must consider/balance both operational and ventilation needs;
- equipment selection, typically driven by the desire to reduce direct capital and operating costs;
- mining method selection: mostly dictated by orebody characteristics and the requirements of the technologies available to the operator; and,
- production rate, which should maximize the NPV of the operation ***and*** also meet other financial and corporate objectives.

Reduction of head losses can enhance the efficiency of a ventilation system and improve its economics. In deep mines, the most severe restrictions in this regard are imposed by the ore deposit itself. It dictates the location and depth of the main ventilation shaft(s) and, through the mining method, determines the type, number, location, and dimensions of development and production openings. The final decision on the size and shape of main airways is based on financial considerations, i.e., on the initial cost involved in excavating and supporting them.[39] Most development and production openings are temporary, and are designed based on ground control and operational considerations (i.e., shape and size). From a ventilation standpoint, however, they are extremely important, since most of the actual work takes place in them (e.g.,

[39] Maintenance costs, as well as associated operating ventilation costs are indeed important. However, they are spread over a longer period, and only become critical burdens once the mine is in full operation. Thus, their impact on NPV calculations made at the time construction decisions are taken is not as significant.

topsills, bottomsills, drawpoints, sublevels, crosscuts, etc.). Thus, their ventilation characteristics are critical to providing adequate working environments in an efficient and cost-effective manner.

Mine operators have very little control over three of the four major heat sources in deep mines: wall rock,[40] autocompression,[41] and underground water.[42] In practice, of all three sources, only the reduction of the heat generated by broken rock has the potential of significantly decreasing ventilation and air conditioning requirements. This is because up to 60% of the total rock heat inflow can be due to rock broken by the various mining activities (Hartman et al., 1997). The operator, then, can minimize rock heat inflow by reducing the volume of broken rock produced, and/or by transporting the muck to surface as rapidly as possible. Again, the mining method, production rate, and geological/geotechnical features of the orebody are controlling the applicability of any potential corrective measure. Nonetheless, the amount of muck produced can be reduced by minimizing dilution and limiting mine development (length and cross-sectional dimensions of excavations). Since dilution can be very high in operations that use bulk mining methods (up to 35% according to Hartman, 1987) adequate control could result in greatly reduced ventilation requirements, in addition to other economic and operational benefits.[43]

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[40] Wall rock heat can be minimized by reducing the length of the airways, the temperature differential between rock and air, and moisture. Airway length reduction was already discussed in this section, but it is pertinent to note that it has a double effect: lower head losses and wall heat flow. Matthews (1989) noted that heat flow from rock walls is the major heat load component (up to 50%) in the stoping areas of deep mines. In most cases, this is due to heat picked up by the air as it ventilates mined-out areas. He proposed the use of backfill to improve the effectiveness of ventilation systems in deep mines. Computer simulations showed that, when backfill is used, reductions of up to 20% in the overall refrigeration requirements can be realized, and that up to 35% less ventilation air may be needed. Maintaining air temperature close to that of wall rock is a realistic option only when the latter is relatively low, not a common occurrence in deep mines. Moisture (and water) reduction is discussed in Footnote 42.

[41] The impact of autocompression also depends on the amount of air flowing through the mine (less air, less heat due to autocompression). The effects of autocompression on the cooling load can only be eliminated by locating the air conditioning plant underground (Hartman et al., 1997). It is interesting to note that Stachulak and O'Connor (1993) did not include autocompression as a significant source of heat at Creighton mine (see Figure 14). Also, Hemp and Deglon, (1980) stated that "... *it is not strictly correct to consider autocompression as a heat source*".

[42] There are two main sources of underground water in mines: ground water (naturally occurring) and mine water. Ground water, which can be a significant source of heat if it has a geothermal origin, can be minimized by grouting, by isolating it, and by collecting it and transporting it away from the working areas of the mine. Obviously, the success of such measures will depend on the specific characteristics of the water flow, its relationship to the orebody and adjacent geological structures (faults, joints, etc.), and the volumes involved. Mine or service water, used for drilling, dust control, and backfill operations, should always be minimized. As noted above, the used of chilled water to provide air conditioning, introduced by the South Africans, is gaining popularity in deep mines.

[43] Such as lower mining cost due to higher grades, production rates, and equipment utilization. Consumption rates of materials and supplies would also be lower on a per-tonne basis (both at the mine and mill).

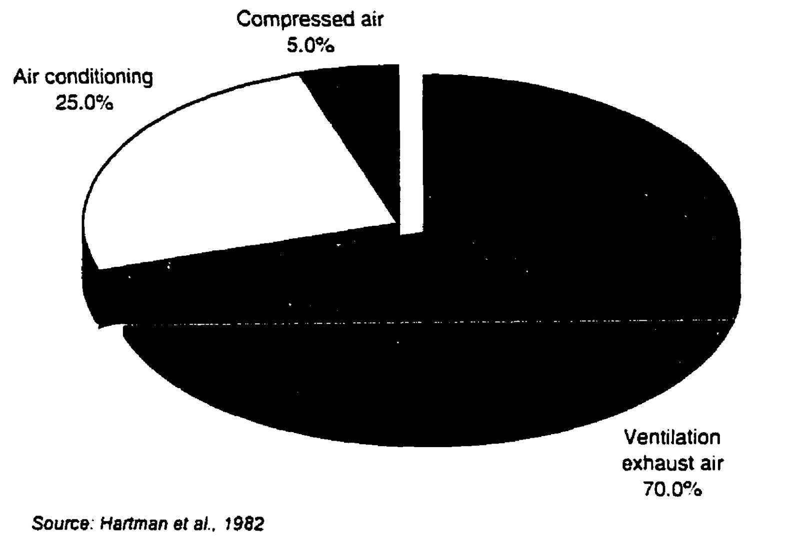

Figure 15: Average heat removal mechanisms for seven *hot* North American mines

The release of heat from broken muck takes place over a certain period, which is a function of the thermal conductivity of the rock, the difference in temperature between the air and rock, and the presence of water. By rapidly removing the "*hot*" muck from its sources (stopes or faces), an important source of heat could be taken from the main working places and, in certain cases, from the total cooling load of the mine. Obviously, the latter depends on the speed of muck transport to surface. Rapid extraction of broken rock is difficult to achieve in old, deep mines, since the transport systems were, most probably, initially designed to meet the needs of a shallow operation. The analysis of this option must consider the mining method, mining sequence, and backfill practices (if used). Methods such as shrinkage or VCR, which require up to 60% of the muck to remain in the stopes before the final drawdown, would facilitate the transfer of broken rock heat to the mine environment (through mine water and/or air flowing through the muck).

Machinery and lights constitute the last major source of heat in mines. As seen in Figure 14, the impact of equipment on the total heat generated in hard-rock mines (especially in large, bulk mining operations) can be quite large: 70% of total heat generated at Creighton Mine. The mine operator, however, does have some control over the problem and adequate measures can be taken, if required, to reduce its effects. Almost all the energy consumption of underground equipment (electrical, compressed air, and diesel) adds heat to the mine air (Hartman et al., 1997). Calculations of the heat liberated by equipment are, therefore, quite straightforward, and several alternatives or scenarios, from a ventilation viewpoint, can be evaluated properly and rapidly.

Usually, one the first considerations is to minimize the number of diesel units operating underground. Electric equipment not only has efficient motors, but also has the advantage of not emitting any toxic fumes (which, in the case of diesel-powered units, require additional ventilation air to dilute them). Internal power transfer losses of equipment are dissipated as a flow of heat. Walker (1988, p. 233) noted that power loss heat dissipated by diesel locomotives is at least 15 times greater than that of battery locomotives of similar operating characteristics. Furthermore, Gundersen (1990) estimated that, for 100-kW duty LHD units, a diesel machine liberates three times as much heat as a similar electric machine. The main difference, however, is in the amount of *surplus* heat generated: as a result of its poor efficiency, the surplus heat of a diesel engine is twenty times that of an electric motor of the same nominal duty. It is estimated that, when ventilation requirements are considered, the hourly energy costs involved in operating a diesel LHD are four times higher than that of an electric machine (Gundersen, 1990). Unfortunately, electric mobile equipment such as LHDs and trucks are not nearly as flexible and versatile as their diesel counterparts.[44] Nonetheless, the following factors must be taken into account during the equipment selection process:

- ***Equipment size***

  The main advantage of the use of smaller equipment is that energy and the corresponding ventilation/air conditioning requirements are less concentrated. Total energy consumption, however, would be higher, since smaller equipment has lower productive capacity per unit of energy. Moreover, if smaller excavations were used to take advantage of the reduced equipment dimensions, higher head losses would result.[45]

[44] Electric LHDs can realize high productivities in cut-and-fill stopes. However, electric mobile mucking and haulage equipment has limited applicability in bulk mines, with the important exception of trolley-powered LHDs and mine trucks. The latter can compete with other vertical transport systems for depths of up to 300 m (Walker, 1988).

[45] On the other hand, smaller excavations would somewhat delay the inflow of rock heat into due to the smaller surface areas, a benefit that should not be overlooked. Since less development muck is produced, there could de also a reduction in the amount of heat generated by broken rock.

- ***Continuous vs. traditional mining systems***

  From a ventilation point of view, the principal advantage of continuous mining systems (excavating machines, transport systems, etc.) is that they are powered by electric motors. Thus, their use would result in drastically less heat generated by machinery, and virtually no emission of toxic fumes. Furthermore, their smaller cross-sectional dimensions would allow the use of smaller excavations for access and transport (see discussion above).[46] On the other hand, the final decision in this regard is chiefly based on efficiency, productivity, and, ultimately, on total production cost.[47] It should be noted that continuous mining systems still have to demonstrate their applicability to hard-rock environments (see Sections 4.1 and 4.2).

- ***Efficiency and productivity of the equipment***

  Regardless of the type of equipment, their efficiency and productivity must be considered in the selection process. This is in order to minimize the total energy utilized by underground equipment, which, as seen above, has a critical impact on ventilation air quality and temperature. Stachulak and O'Connor (1993) observed an increased diesel power consumption (expressed as horsepower of equipment employed per tonnes/day of production) with increased depth of mining. A mobile equipment fleet that minimizes the hp/tpd ratio could be a starting point for equipment selection and ventilation requirement studies. On the other hand, the energy requirements per unit of productive capacity of locomotives and conveyors, typically powered by electric motors, are much lower than that of trackless diesel equipment. Even when mine trucks and LHDs are being contemplated, there are additional issues to be taken into account. Indeed, as seen in Figure 16 trucks have much lower operating weight-to-payload ratios (compare to Figure 12). Thus, LHDs should be limited to short haul distances: a maximum of 150 m, one-way, in normal conditions (Walker, 1988).

[46] The benefits of mine development excavations of smaller cross-sectional dimensions and lighter mine development equipment must be balanced against the higher ventilation cost resulting from increased head losses.

[47] This thesis postulates that the minimization of production cost, although a simple and straightforward concept, is not the only objective of the mining strategy (see Section 2.5.1). Unfortunately, the implementation of a more integrated approach to decision-making in underground mines is difficult and requires a degree of interaction between senior management, operations, and engineering not common in the industry. However, for specific cases such as equipment selection, the comprehensive estimation of the impact on total production can be an effective tool.

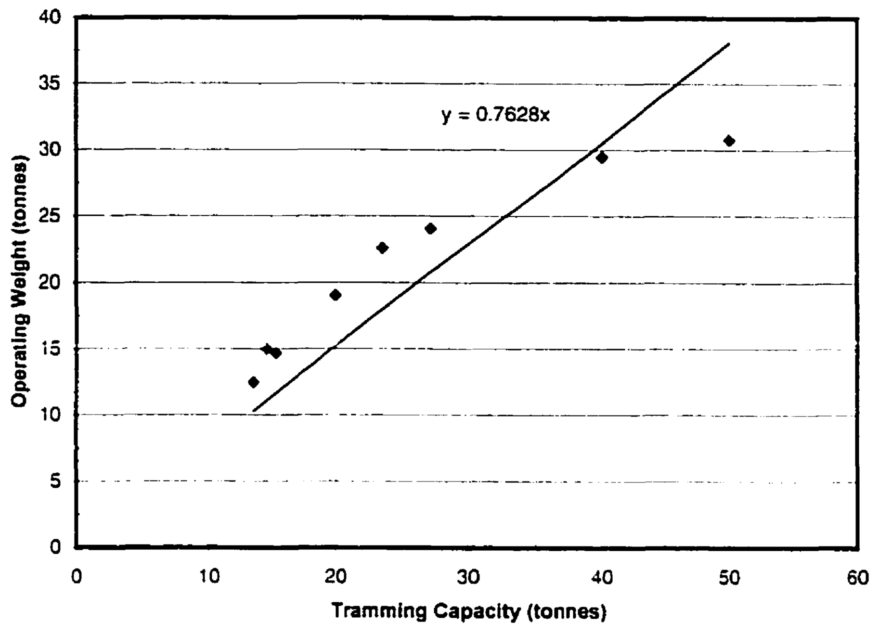

Figure 16: Tramming capacity vs. operating weight - Tamrock mine trucks

The impact of technologies that could change drastically the design and operation of deep underground mines must be considered. It has been suggested that automation and tele-operation of mobile equipment could result in the absence of humans in underground mines. and that this. in turn, will "... *change the needs for ventilation. heating or cooling of air.* ... " (Vagenas et al.. 1997). The ventilation needs of a person-less mine would be certainly different. but it is not clear how this could result in less strict air specifications and. thus. *lower* ventilation cost. For instance. diesel engines lose efficiency in high ambient temperatures, they require oxygen for internal combustion. and need clean air. free of dust and other particles, to avoid constantly blocked intake filters. If tele-operation of mobile equipment is being considered. it is obvious that clear video pictures would be required by the surface operator in order to control the machine.

Extensive research projects sponsored by the largest Canadian underground mine operators have been aimed at automating the operation of LHDs. (Poole et al.. 1998; Piché and Gaultier. 1996). If, as the literature indicates, large pieces of diesel equipment running on rubber tires can be automated or tele-operated, why not replace them with electrically powered steel bins running on tracks? They could be loaded and dumped automatically and, as discussed above, would significantly improve ventilation and the economics of the operation. In fact, they would produce less heat, demand no combustion air, use even less energy due to the reduced rolling resistance, and could run in smaller openings because of the precision of tracked guidance systems. However, some flexibility and the ability to operate in several levels with the same equipment would be lost, and a separate loading device would be needed (at the drawpoints).

4.4 Mine Development

Mine development is a major constraint on deep hard-rock mining operations. The reason for this is twofold. First, the pre-production development stage takes much longer to complete, as a direct consequence of both the depth of the ore deposit and the difficult working conditions (i.e., stresses, rock temperature, groundwater, etc.) encountered at such a depth. A long development phase negatively affects the net present value of a proposed mining operation, since sizeable capital investments must be made prior to achieving any financial return. It should be noted that this is a problem experienced by all deep mines, regardless of scale of operation and mining method.[48] For example, Figure 17 and Figure 18 plot construction time versus shaft and ramp depth, respectively, for mining rates ranging from 4,500 to 18,100 tonnes/day. It can be seen that the bottom of a 900-metre deep orebody could be reached in about 110 to 130 weeks with a fully equipped shaft, or in about 140 to 150 weeks with a 15% ramp. Since only limited lateral development, and virtually no production work, can be carried out before the main access facility is operational, its construction becomes part of the critical path of a project. It is interesting to note that shaft construction time increases only by 16% when production rate increases by 300%.

Second, even at the production stage of a mine, large amounts of development waste rock are generated. Indeed, in some hard-rock operations up to 45% of the muck hoisted to surface is development waste (Scales, 1998, p. 33; Scales, 1996, p. 40). While this is not a unique feature of deep mines, the higher cost associated with mucking, transporting, crushing, and hoisting development waste, as well as the restrictions imposed on the production system, have more

[48] This is because speed of development does not vary significantly with the size (i.e., cross-sectional area) of the development openings. Only when radically different technologies that do not rely on the drill-blast-muck-support cycle, are employed, a drastic improvement in development speed can be achieved.

serious consequences in such operations. As discussed in Sections 4.1 and 4.2, vertical and horizontal muck transport are expensive and time-consuming tasks critical to the efficiency of the entire production system. They constitute bottlenecks in the mining process and, particularly when a hoisting system is used, impose a definite upper limit to a mine's throughput capacity.

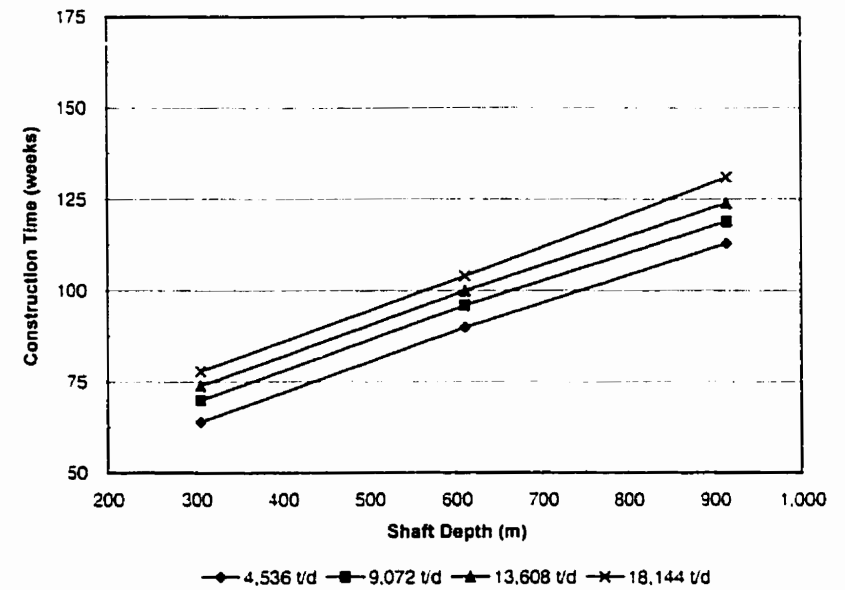

Figure 17: Shaft construction time vs. depth

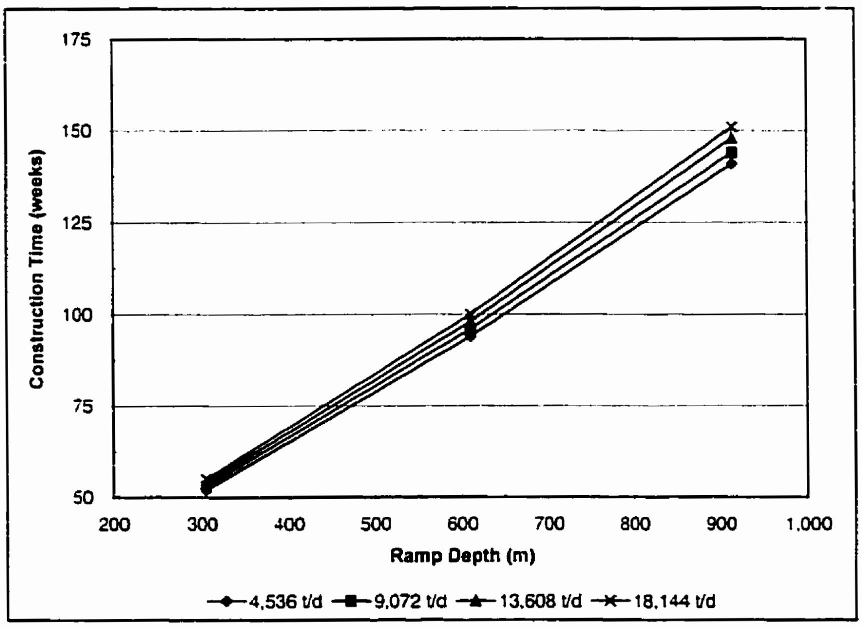

Figure 18: Ramp construction time vs. depth

The problem of disposing of development waste persists for the entire production stage of a mine. Thus, as depth of mining increases and the production system loses flexibility, it has the potential of severely affecting the operation and its economic performance. The analysis of ongoing waste development production must consider waste that is hoisted to surface for disposal and waste that is disposed of underground. The cost involved in transporting waste to empty stopes as rockfill is relatively low, at least when compared to the cost of hoisting it to surface. Furthermore, it has the additional benefit of providing some ground support and improving the ventilation system. However, in bulk mines this practice requires additional scheduling, may impose further restrictions to an already complex production plan, and can reduce the flexibility of the operation.

Although the cost of hauling development waste and its effect on the mine's cost structure can be easily estimated it is difficult to quantify the impact of development waste transport on the efficiency, productivity, and flexibility of a mine. After all, mine development is a fundamental stage of the production process. Nonetheless, it is evident that deep mines must strive to reduce the amount of pre-production and ongoing mine development. However, it is not clear how this can be best achieved, or how a general development strategy/policy can be established. The first step in the process of optimizing mine development is to define its objectives and establish its relationship to the specific needs of the operation. An obvious way of minimizing the amount of development waste is by increasing the lift height (vertical distance between contiguous levels). This is investigated in the case studies included in this thesis (Chapters 5 and 6).

4.4.1 Mine Development and Design

The overall objective of mine development is to open a mineral deposit for exploitation. In underground mines, development precedes exploitation. However, both activities coexist throughout a mine's life due to financial restrictions that prevent the development of the entire mine at once (prior to production), and the evolutionary nature of underground mining, which makes it impractical to develop it completely in advance (Hartman, 1987). Mine development provides access to the ore deposit, allowing entry of personnel, equipment, supplies, power, water, and ventilating air. It also provides extraction routes for the ore and waste produced.

As noted by Young (1916, p. 395), "... *development should be planned to render accessible the maximum quantity of ore for a minimum volume of development workings*". In other words, while it is clear that the function of mine development is important, the mine operator must minimize the amount of time and resources allocated to this activity. This is true even in cases where extensive in-ore development is needed, such as in VCR stoping and caving mining.

Design decisions made at the pre-production development stage have a crucial impact on the operation, profitability, and, eventually, competitiveness of a mine. Therefore, prior to committing a mine to a certain development pattern that, most certainly, will continue for the rest of the mine's life, several mine design and planning aspects must be carefully considered and determined. Such aspects include the mining method, production rate and mine life, main access and extraction openings, inter-level spacing, and level development layout.

The relationships between the mining method and mining sequence in deep hard-rock mines were discussed at length in Section 3.3. Similarly, main access and extraction openings as well as level layout were dealt with in Sections 4.1, 4.2, and 4.3. Furthermore, they are largely site-specific and, from a mine development viewpoint, they must be analyzed and evaluated in the light of the local mining conditions and requirements. An issue related to those aspects that has a large strategic potential is the size (i.e., cross-sectional area) of the development openings. It will be discussed, together with production rate and inter-level spacing, in the remainder of this section. The importance and need of new development technology, which could radically change the development process and reduce all associated costs, will also be addressed.

4.4.2 Size of Development Openings

From ground control and mining cost perspectives, it is obvious that it would be advantageous to minimize the size of the development openings. This logic, however, fails to consider that such openings do have a purpose, and that they are integral part of the mine's access, supply, and ventilation networks. A comprehensive approach to mine design and planning must evaluate the effects of development opening size on the overall production process and long-term competitiveness of the operation, not just cost.

4.4.2.1 Opening Size Determination

The two most important parameters in determining the optimum size of access and extraction openings are mobile equipment dimensions and ventilation requirements.[49] Mobile equipment dimensions are chiefly dictated by the mining method: both production and development needs must be met by the equipment selected. Additionally, there are regulations that stipulate minimum clearances between the vehicles and haulageways walls and backs. The Ontario Mining Act, for example, indicates a minimum of 1.5 m (5 feet) between any vehicle and the haulageway walls (i.e., a drift must be at least 1.5 m wider than a vehicle, thus leaving a 0.75-m clearance on each side). However, it does not address the issue of minimum vertical clearance.[50]

Equipment selection demands a compromise between capital investment, ventilation and cooling requirements, fleet size, mechanical availability, and production cost (in dollars per ton of material transported). Acquisition, ventilation, and cooling costs are usually higher with larger equipment, but actual figures would depend on the total number of units required. Although larger equipment usually results in lower production costs, this ultimately is a function of the location of the active mining areas, haulage distances involved, and production rate. Fleet size should be carefully determined. A small fleet of larger units reduces the complexity of maintenance schedules and operating labour requirements, but restricts the flexibility of the production plan by assigning larger percentages of the production capacity to individual units. Grade control could be adversely affected if a smaller fleet were in operation, as well as the ability to meet production targets when equipment failures take place (Stevens and Acuña, 1982).

As discussed in Section 4.3, deep mine ventilation requirements are determined by the amount of heat to be removed, and the need to dilute and remove contaminated air and dust. The optimum dimensions of the airway (raise, drift, or ramp) will depend on the excavation and ventilation

[49] It could be argued that ground condition should also be included as an additional decisive factor. However, it does not have to be listed separately since it already controls mining method selection and optimum production rate determination (aspects that, in turn, decide equipment dimensions and ventilation requirements). Furthermore, the wide range of general design tools and support systems available makes it possible to achieve the required excavation dimensions in almost every type of ground condition. As expected, there is an additional cost to deal with deteriorating ground.

[50] In fact, it only indicates that the operator should be able to sit erect.

(operating) costs. There are, of course, maximum and minimum speed limits that must be observed in openings where either pedestrian or motorized traffic is expected.[51]

4.4.2.2 Excavation Size and Cost

The immediate effect of a reduction (increase) in the cross-sectional area of a development opening is a reduction (increase) in the volume of muck produced and the unit cost of advancing it (in $/m of excavation). The impact on volume is more significant than that on cost.

For instance, the excavation of a 4.0 m x 5.0 m drift produces, in theory, 20.0 m^3 of muck per metre of advance, whereas a 3.5 m x 4.5 m drift (whose dimensions are only 0.5 m shorter) would result in 15.8 m^3/metre: a 21% reduction in muck volume. The respective decrease in development cost would not be as pronounced. The actual cost differential would depend on operational practices, crew size, bonus regime, depth, availability of equipment, and mucking and haulage distances. Figure 19 presents a breakdown of development cost for various excavation sizes. The figures, calculated using *first principles*, were provided by a deep, base-metal, bulk-mining operation. It can be seen that the unit cost of a 13.4-m^2 drift is about $1,900/metre, whereas that of a 24.5-m^2 drift is about $2,400/metre. Although the area of the larger excavation is about 83% bigger, it only would cost about 26% more to advance it.

Figure 19 illustrates other points about the cost of excavating development openings. First, labour and equipment cost remain virtually constant, regardless of area of excavation. In the case of labour, this is because, for the range of excavation sizes shown in the graph, the same number of workers was employed to drive them (six people per day). Similarly, although smaller LHDs and trucks were considered to calculate the unit cost of the smaller excavations, the increase in hours of operation are offset by their lower hourly operating and maintenance costs. On the other hand, the cost of supplies shows an increasing trend, which is explained by the higher consumption rates of drilling steel, explosives, and support materials of the larger openings. Indeed, the higher

[51] Larger openings result in less resistance to airflow, but air velocity is reduced. Airflow speed must exceed that of vehicles travelling through a haulage route (in the same direction as the airflow), otherwise the concentration of dust, fumes, and heat generated by such a vehicle will continue to increase. This is particularly critical in main ramps, where loaded trucks and other vehicles operate simultaneously.

supplies cost of the 26.0-m^2 drift is mostly due to the use of shotcrete. This highlights the discontinuous nature of this "*cost function*". An increase in the cross-sectional area of a development opening will result only in a slight increment in the corresponding excavation cost, unless such an increase demands different operational practices (e.g., more personnel, much larger equipment, more ground support due to the larger span, etc.).

4.4.2.3 Discussion

Several interrelated factors must be accounted for in development opening sizing. The effect on capital and operating cost is an essential consideration. However, the dimensions of development openings determine and limit an operation's productive capacity, flexibility, and long-term planning. This makes it imperative that an integrated approach to excavation sizing be adopted. The use of smaller development openings cannot be justified on excavation cost savings or other short-term considerations. The impact on production cost, profitability, and the competitiveness of the operation must be evaluated in order to make sure that the excavations are properly sized.

In deep operations, the determination of the *optimum* dimensions for mine development and extraction openings must consider the effects on the mining strategy. Special attention must be paid to the following:

- new ventilation and air conditioning requirements;
- labour requirements;[52]
- production of waste rock, including direct (i.e., horizontal and vertical transportation) and indirect (i.e., reduction in hoisting capacity) costs;
- dilution and ore recovery;
- ground control;
- flexibility of the operation (smaller production units could facilitate grade control, blending and just-in-time changes to production schedules); and,
- overall economics of the operation (the ability to reduce dilution may in fact increase profitability by maximizing the use of resources on revenue-generating activities).

[52] The labour component of an underground mine can always be increased, but only to a point beyond which the resulting operating cost is too high, regardless of the operational benefits that could be achieved by the new mining strategy. This imposes a lower limit to the size of the equipment.

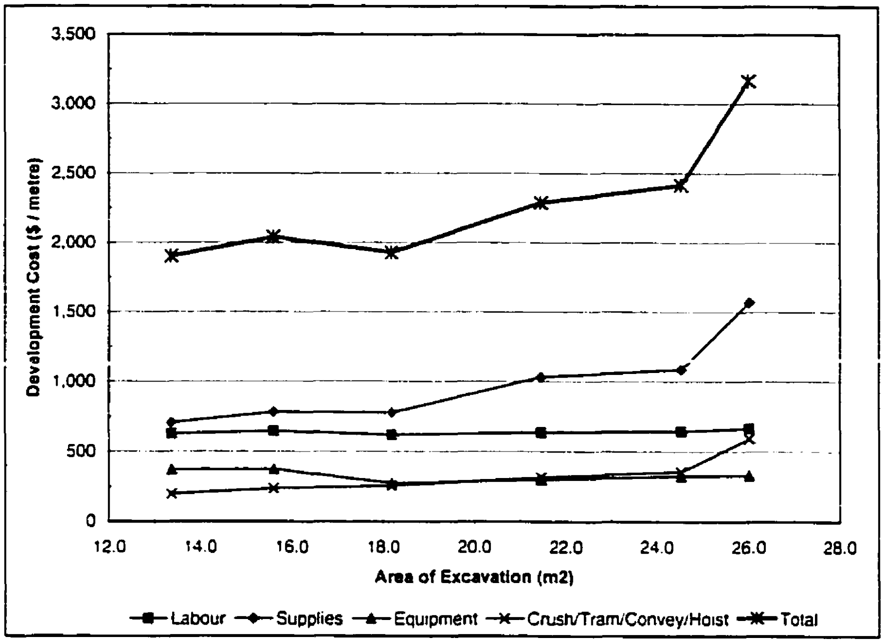

Figure 19: Breakdown of development cost for various excavation sizes

4.4.3 Production Rate

The production rate of a mine is a function of the ore reserves (tonnage and grade of proven plus probable reserves), unit value of ore, financing capabilities of the owner/operator, and the mining cost expressed in dollars per unit of commodity produced. In the specific case of deep hard-rock underground mines, two additional parameters (or, more properly, two additional *limiting factors*) are mine development and mining sequence. The pre-production mine development phase, as previously noted, can be rather long and, depending on the cost of capital and inflation rate, it can eliminate any economies of scale to be achieved by higher production rates. The mining sequence, which in deep mines is dictated by ground control considerations, may impose restrictions on the speed of ongoing development and ore extraction. In the case of existing operations, the ability to increase the production rate is governed by the production capacity of equipment and facilities, mainly in the areas of ore processing and extraction. In fact, the hoisting system is usually the limiting factor when the expansion of a deep mine is being considered.

4.4.3.1 Production Rate Determination - Rules of Thumb

Over the years, a number of rules of thumb have been developed to determine the so-called *optimum production rate*. They all aim at balancing ore reserves, capital expenditures, and expected revenues (Smith, 1997; Smith, 1996). The most common are Taylor's Law, the seven-year minimum productive mine life, the rule that the cash flow must be sufficient to repay the capital cost twice, and the maximization of net present value of expected cash flows.

- ***Taylor's Law***

  Taylor's Law (O'Hara and Suboleski, 1992; Taylor, 1986) is expressed as:

$$T = \frac{4.88 * T_r^{0.75}}{D_{yr}}$$

where:

T :	tonnage of ore mined per operating day (in tonnes)
T_r :	total ore reserves, diluted (in tonnes)
D_{yr} :	number of days per year of operation at full capacity. Usually, D_{yr} is equal to 250 (using a five-day week) or 350 (seven-day week)

As noted by Taylor himself, his formulae are not directly applicable to deep underground mines. He correctly pointed out that, in very deep mines, "*a long and strongly sequential series of varied underground activities precedes the actual production of ore*" (Taylor, 1986). This refers to the long development phase and the need to build the mine structure before any actual mining activity takes place in the stopes. The net effect is that a somewhat high proportion of the ore reserves adds to mine life without significantly increasing production rate. Thus, if applied to a deep deposit, Taylor's formulae would result in production rates higher than operationally feasible ones.

- ***Seven-year minimum mine life***

  The seven-year minimum mine life criterion has its origins in the need to re-pay the initial capital investment, maximize the recovery of the ore reserves and, equally important, provide for the proper development of the mine. However, the conditions under which these factors start playing their envisioned roles are usually related to smaller open pit deposits. This criterion also may rule out the selection of unreasonable high production rates in the design of deep underground mines (in which mine development critically affects the pre-production stage). Its application is discouraged, however, since there is no sound justification for this rule of thumb[53] and more suitable criteria have been developed.

- ***Maximization of net present value of expected cash flow***

  In contrast with the previous two rules of thumb, which call for simple calculations based on the ore reserves to calculate so called *optimum* production rates, this criterion requires the development of several scenarios to investigate the project's performance under various rates. Each mining scenario would include estimates of yearly production plans, revenues, operating costs, and capital expenditures. This is not difficult to do for open pit mines[54]. In fact, most of the published case studies that analyze this option fall into this category. As discussed in Sections 2.5 and 2.6, the significantly more complex nature of underground operations (and of the corresponding design and planning processes) makes the development of such scenarios extremely time-consuming, at least with current tools and methodologies.

  The application of the maximization of the net present value criterion appears as a straightforward solution to the determination of the optimum production rate of a mine.[55] On the other hand, it has been reported that this method consistently results in very short mine lives, is insensitive to increased minable reserves, and may not yield an operationally optimum production rate (Smith, 1997; Taylor, 1986). Nonetheless, the process of thoroughly analyzing several mining scenarios is always very constructive and must be undertaken at the late feasibility stage as part of a comprehensive sensitivity analysis. If properly estimated, i.e., if operational restrictions are taken into account, a production rate that maximizes the net present value of expected cash flows could be considered as the upper limit of the range of

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[53] The author is not aware of any study (empirical or theoretical) that demonstrates that a seven-year mine life results in increased profitability, enhances the economics, or otherwise benefits the operation of underground mines.

[54] Koniaris (1991) suggests that these analyzes should be carried out on a regular basis while developing medium and long term plans for open pit mines.

[55] After all, this method is fully in agreement with the school of thought that postulates that the maximization of the net present value of expected cash flows is ***the*** objective of any mining venture.

possible rates for the project. A more conservative estimate of such an upper limit is the value obtained using the seven-year rule.[56]

- ***Cash flow must be sufficient to repay the capital cost at least twice***

  This rule of thumb, a "*banker's rule*" according to Smith (1997), has its origins on the "coverage ratios" required by financial institutions that provide loans to mine developers. If a project satisfies this criterion, it is very likely that such ratios will be met. In general, the optimum production rate determined using this concept is lower than that obtained with other criteria, with the exception of very small, shallow deposits in which the seven-year rule may result in unreasonably low production rates. For economically feasible projects, usually there are several production rates will meet this criterion. In such cases, the operator has a wide range of rates that could be investigated in order to determine the one that also complies with operational requirements and restrictions. As in the case of the maximization of the NPV, this rule requires the development of several complete mining scenarios, which may or may not be a reasonable proposition at the feasibility stage of a project.

4.4.3.2 Mine Development and Production Rate

The speed of mine development is an important factor that restricts the production rate of deep underground mines (Taylor, 1986). As illustrated by Figure 17 and Figure 18, new deep mining projects should expect extremely long pre-production development phases. This can be a strong incentive to design an operation with the highest possible production rate, which would mitigate the negative impact of the first several years without any positive cash flow. Unless sound mine design and planning are part of the production rate determination process, this may lead to rates that are not operationally feasible and, thus, to serious financial problems. The trouble is, very rarely the quality of the data available at the feasibility stage of deep mines allows the evaluation of several mining scenarios, as required by the more elaborate criteria. This is further complicated by the complexity of underground mine design and planning and the lack of efficient tools to carry them out (see Section 2.6). It could be argued that, given the poor quality of the data, there

[56] O'Hara (1992) has noted that if the ore reserves can sustain the mining rate for seven years or more, an increase in mining rate will result in a higher discounted cash flow.

is no point in using time-consuming sophisticated techniques and tools to produce approximate results only. However, there is a real danger of committing the operation to unattainable mining rates, or to low rates that prevent it from achieving a better economic performance. The length of the pre-production development period turns deep mining projects very sensitive to delays in reaching the proposed production rate due, in most cases, to ground control problems.

In existing operations, ongoing mine development is needed to bring new mining areas into production. Depending on the mining method, it mostly consists of horizontal openings and short orepasses excavated both in-ore and waste rock. The intensity of ongoing development is dictated by the mining sequence and the geometric characteristics of the orebody, and is usually expressed in terms of metres of development per tonne of ore brought into production. In practice, the number of development headings that can be advanced simultaneously is limited and, therefore, this constitutes an additional restriction to achieving higher production rates.

4.4.4 Inter-Level Spacing

The inter-level spacing, i.e., the vertical distance or interval between levels, is a function of several factors such as orebody geometry, geotechnical characteristics of the rock, mining method, and financial considerations (Hartman, 1987). In his analysis of underground hard-rock mining practices in Ontario, Pelley (1994) shows mine layouts with inter-level spacings ranging from 38 m at Lakeshore, Kirkland Lake to 90 m at Kidd Creek, Timmins.[57] A review of mining methods and description of operations compiled by Hustrulid (1982, pp. 227 – 997) indicates that the interval between main levels can vary from 45 to 122 m in sublevel stoping mines; from 30 to 200 m in shrinkage stoping; and from 30 to 75 m in cut-and-fill mining.

Lewis and Clark (1964, p. 416) noted that "*The question of selecting the interval to be used between levels is basically one of determining the distance that gives the least cost per ton*

[57] Research carried out by Pelley (1994, pp. 54 – 140) shows that there has been a slow but persistent evolution of inter-level spacing in Ontario hard-rock mines over the past few years. In Sudbury and Kidd Creek, operators have traditionally used 60-metre spacing of main levels, whereas at Campbell a 45-metre inter-level spacing has been strictly enforced. At the Hemlo mines, the inter-level spacing was expanded to 100 metres with 25-metre intervals between mucking sublevels at the David Bell and Williams operations, and 33-metre mucking sublevels at Golden Giant. Campbell is now considering 30-metre spacing for development below the 30 Level (Pelley, 1996).

or (sic) *ore mined for the method of mining chosen.*" The effect on mining cost is very important and evident: increased spacing results in fewer levels, which in turn reduces mine development and level maintenance costs, accelerates the mining process, and decreases financial requirements. Furthermore, it is relatively simple to evaluate, from a mining cost perspective, several level spacing alternatives. This is illustrated by Figure 20, which plots the mining cost (in dollars per tonne) vs. the level interval for a shallow shrinkage mining operation exploiting a narrow, steeply dipping scheelite vein. The main criticism to this kind of analysis is concerned with the unit of measure employed: instead of mining cost in **dollars per tonne of ore**, the vertical axis should plot mining cost in **dollars per pound of tungsten**. This is not a trivial observation, since the inter-level spacing has a profound effect on ore recovery and dilution, which determine ore grade and, eventually, the profitability of the operation.

On the other hand, direct production cost is not the only issue to be considered: the relationship between inter-level spacing and the mining strategy is also critical. This implies evaluating the long-term impact of several configurations on the entire operation, both from economic and operational viewpoints. Main operational aspects include ease of orebody delineation, time required to bring a new stope into production, ground support systems required for mined-out stopes, accessibility for equipment and personnel, equipment size, and overall flexibility. Unfortunately, and unlike the mining cost, such issues cannot be evaluated with "*hard*" numbers. As in the case of mining method selection (or, most probably, in conjunction with such a process[58]), several mining scenarios would have to be developed in order to test theoretically the economic performance of the operation under different level intervals. Each scenario would be optimized in terms of type of equipment, development opening size, and mining sequence.

In Canadian hard-rock mines, trackless mining and sublevel stoping require the use of ramps to connect all active mining areas in order to allow the movement of equipment, personnel, and supplies. This has certainly increased the efficiency and flexibility of the operations. However, are the higher efficiency and flexibility justified in the light of the increased costs of excavation at

[58] Lewis and Clark (1964, p. 417) correctly pointed out that most of the issues that influence the determination of the "optimum" inter-level spacing are related to the characteristics of the orebody and rock mass. Since such features determine the choice of mining method, both processes are closely related.

depth? To some extent, fewer levels (and less horizontal development) can balance the additional cost of having to excavate the main service ramps, without affecting flexibility.

The fact that direct mining cost improvement is not the only important consideration when determining the inter-level spacing of deep operations is illustrated by changes experienced at the mines in Hemlo, Ontario. Over the years, various sublevel intervals together with mining and extraction sequences have been tried at each of the three mines (Pelley, 1994, pp. 141 – 160). Indeed, 25-metre levels were used at Williams, 25 and 33-metre sublevels have been used at Golden Giant, whereas at David Bell 100-metre level intervals were used at the shallower cut-and-fill stopes and 25 and 50-metre sublevels at the blasthole stopes (see Footnote 57). The main justification for the changes can be found in enhanced dilution and ground control. Shorter distances between sublevels resulted in increased drilling accuracy and lower dilution because of reduced stope sidewall spans. Furthermore, the subsequent reduction in rockburst activity, which in some cases resulted in entire stopes being lost, improved safety and ore recovery.

4.4.5 Development Technology

The issues previously discussed in this section have prompted mining companies and manufacturers of underground mining equipment to spend significant amounts of research money into the quest for alternative mine development methods and technology (Gertsch, 1994). In addition to the expected benefits noted in Section 2.4.2, which apply to the introduction of new technologies in any area of the mining process, the justification for innovative mine development technology can be found in:

- increased speed of development;
- lower unit development costs (in $ per metre of excavation); and,
- reduced production of waste.

Achieving one or more of such objectives could result also in decreased direct operating costs, but as discussed in Section 3.2, the benefits of new technologies must be evaluated in the context of the overall mining strategy. It is especially critical to analyze the effects on the next phase of the production process, i.e., actual mining. The following issues must be considered:

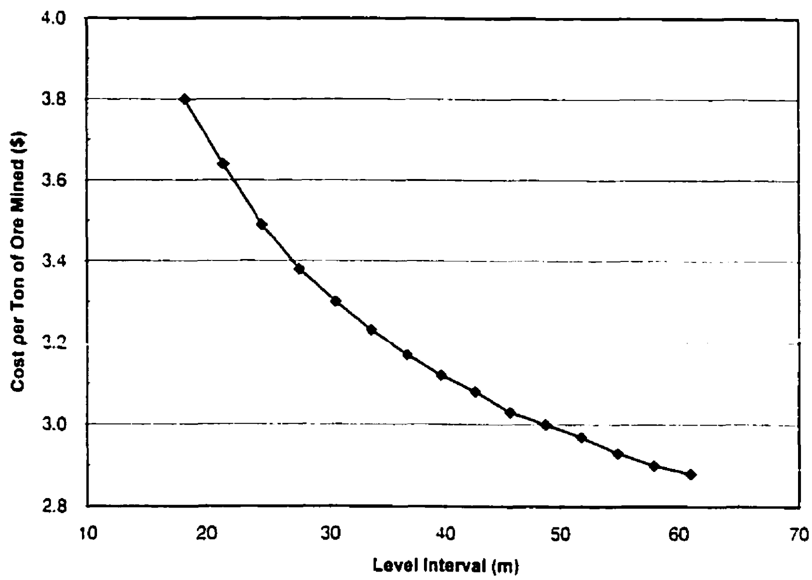

Figure 20: Effect of inter-level spacing on mining cost – shallow shrinkage mine

- ***Shape of the excavation***

The shape of the development opening must allow transport systems to take full advantage of the available space. At the same time, it must meet both support/geotechnical design and ventilation requirements. Balancing such demands is not an easy task. For instance, a sub-horizontal opening with circular cross-section, which could be excavated with a *tunnel-boring machine* (TBM), has the ideal shape from ground support and ventilation viewpoints, but would require additional work to allow the use of existing transport systems. Not surprisingly, industry and research have placed a strong emphasis on developing openings with profiles that support such systems. While this may be adequate in the short-term, it could be argued that the long-term solution to the mine development problem, a major issue that affects the competitiveness of deep underground mines, requires also an innovative approach to haulage. It is imperative, for example, that continuous transport systems, regardless of their type of operation, take full advantage of the circular shape of tunnel-bored openings.

- ***Stability of excavation***

Both the excavation technique and the shape of the opening have a significant impact on its stability. Mechanical excavation systems would provide smooth and less damaged walls that not only have less ventilation resistance, but also could be amenable to innovative and more efficient support systems such as spray-on linings. However, traditional drill-and-blast excavation systems result in a damaged area around the openings that, when supported and kept in place, provides a destressed region which is highly desirable in the stressed environments typical of deep hard-rock mines. Such an area most probably will not be present in mechanically excavated openings. On the other hand, the ability to excavate rounded and horseshoe shaped openings, which are more efficient from a stress management viewpoint, is certainly an advantage of mechanical excavators (Bullock, 1994).

- ***Maintenance***

  Traditional mine development methods result in openings that require high levels of maintenance during most of their operating life.[59] An improvement in this area would require the adoption of different transport systems, i.e., a departure from LHD- and mine truck-based systems, which demand extensive haulageway maintenance. Another important aspect is ground support maintenance, an area that could be enhanced by designing the shape of the openings with this aspect in mind.

- ***Ventilation characteristics***

  The issue of ventilation has been dealt with extensively in Section 4.3. However, it is pertinent to note that, unless expensive pre-splitting or smooth blasting techniques are employed, traditional development methods result in excavations that have poor to fair ventilation attributes. Mechanical excavation techniques, on the other hand, have the potential of improving simultaneously the ventilation **and** support characteristics of development openings, two issues critical to the long-term profitability of deep underground operations. Furthermore, if coupled with continuous material transport systems designed to take full advantage of the available space, the amount of waste and development time could be reduced drastically by minimizing the cross-sectional area of the excavations.

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[59] As discussed in the *real* case study presented in this thesis (see Chapter 5) underground maintenance and support can account for up to 12% of total mining cost. Major components of such a cost item are ground support and haulageway maintenance costs.

For several years the mining industry has been trying to develop procedures and technologies that facilitate the rapid excavation of single levels (Pelley, 1996). The perception is that, by concentrating production capacity in fewer levels, development and ground control problems can be solved, and the costs associated with ventilation, personnel, and supplies distribution can be reduced (Moss et al., 1995).[60]

Rapid excavation and exploitation of a level demands timely availability of access openings for the following level in order to maintain the established mining sequence and production rate. This may be feasible in an operation that exploits a large orebody with only a few stopes in production at a time. It may be impractical, however, in a narrow-vein mine that requires the simultaneous exploitation of several stopes located in many levels. Deep bulk mines such as Kidd Creek sometimes require mining sequences that span more than one level (see Figure 11). In those cases, rapid level development would demand a re-evaluation of the mining sequence in order to match the speed of mine development without having to maintain several levels in stand-by. Obviously, rapid development also requires rapid removal of the muck produced (ore or waste). This may be the reason for the failure of in-backfill development at Inco mines (Pelley, 1990).

Innovative underground excavation technologies "*... have the potential to excavate hard rock when the application is right*" (Bullock, 1994). The trouble is, underground mine development is a dynamic activity, with varying working conditions and applications even in a single operation. This is why existing mechanical excavating machines are unable to fully replace traditional drill-and-blast methods (which do have the ability to adjust to changing conditions and requirements) in underground hard-rock mines. However, it is believed that, given the fact that most mines have already optimized traditional development methods to the maximum, the future competitiveness of this segment of the industry depends largely on new technology.

[60] In fact, at least in theory, *just-in-time* (*JIT*) development can have an enormous beneficial effect on the profitability of an underground operation. This is mostly because the significant financial burden associated with advanced development is lifted. Since there are fewer levels to maintain and ventilate, fewer development crews are required. The amount of development waste is also minimized, and the efficiency of the operation, particularly that of extraction systems, is greatly improved. It should be noted, however, that rapid development and JIT development are not necessarily the same. The former refers to development speed, whereas the latter is focused on the ability to excavate development openings as they are required, with minimal lead time.

4.5 Delivering Personnel and Supplies

This is the *softest* of the five factors affecting the profitability of deep underground hard-rock mines. Although it is relatively simple to define the scope of this factor, it is difficult to generate a hard, reliable estimate, i.e., one that expresses its true cost in dollars per pound of metal produced. Its calculation and evaluation, in conjunction with other mining cost items, is further complicated by the lack of appropriate data. Indeed, typical cost collecting and mine management systems routinely keep track of the usage of service equipment (service hoists, scissors lifts, personnel carriers, service trucks, etc.). However, they rarely specify the specific areas of the mine that are serviced, the time involved in such activities, or, in the case of mobile equipment, the transport routes followed. Furthermore, in most cases only partial information is collected, preventing the calculation of the actual cost involved.[61] It is believed that the main reason for the lack of accurate personnel and material transport cost figures is related to the fact that, when not properly calculated, it is only a minor percentage of the total production cost. Thus, there is little incentive to spend the resources needed to adequately track it down.[62]

It can be argued that even the actual cost *per se* is not as significant as the impact that these activities have on the mining operation. As discussed with the sponsors of this research project, there is the perception that the cost of delivering personnel, materials, and supplies to underground working areas can become an issue in deep hard-rock mining due to the following:

- The long haulage distances involved in deep mines, which increase the cost and time needed to distribute personnel and materials to the corresponding work sites. This has a direct effect on the cost structure of deep mines. However, the actual costs would be probably small, at least when compared with other components of the mining cost. Deep mine operators must be able to allocate such costs accurately, since they can have

[61] As an example, data provided for the first case study show that the operation does keep track of a cost item called *underground personnel and materials transport* (see Chapter 5). However, careful analysis of the data indicates that the item is composed mostly of the cost of ***maintaining and repairing*** scissors lifts (41.0%) and personnel carriers (20.2%), as well as moving heavy equipment (21.2%). In other words, the operating costs involved in such activities, i.e., the cost of labour and fuel oil, are not recorded. Furthermore, the cost of hoisting personnel and supplies (significant in a deep operation) is not known. Not surprisingly, this item only represents 0.9 % of the total production cost at this mine.

[62] In fact, one of the operations studied as part of this research project does not keep a separate cost item for personnel and materials transportation.

a direct and significant impact on the total production cost of isolated areas of the mine. This is particularly true when both financial and operating costs are considered.[63]

- The decreased efficiency on the utilization of the resources of the operation (personnel, materials, and equipment) resulting from the additional time required to deliver them to the actual workplace. Longer transportation routes increase also the risk of delays and interference with other production activities, and demand stricter scheduling. The indirect consequences of longer delivery times are not easy to determine in an ongoing operation. First, it requires a thorough understanding of the production process and the interactions between the various components of the system in order to establish the differences between the optimum (or designed) and actual operating conditions. Second, the mine management system would have to collect data that allow the actual determination and quantitative evaluation of such differences. In the particular case of personnel transport, the sensitive issues of efficiency and productivity arise, which can only be dealt with in conjunction with other labour relations' topics.

Lack of adequate data, public domain or otherwise, does not allow a detailed analysis and evaluation of this particular factor. However, based on the limited information provided by the sponsors and previous experience, it is possible to discuss (quantitatively) its significance and impact on deep hard-rock mining.

4.5.1 Personnel

The actual cost of transporting personnel in an underground mine is small. Even in deep operations such as those studied in Chapters 5 and 6, this cost is only tracked down partially, or as part of another cost item (in which case it is impossible to determine is actual worth). As noted above, however, the investment required for setting up the facilities and equipment needed to carry out this activity can be quite significant: as a minimum, they include a dedicated hoisting system and mobile transport equipment. Nonetheless, the corresponding operating cost is typically a small percentage of the total mining cost.

Personnel must be transported to and from the working areas according to either a pre-determined schedule or particular requirements of the job. Failure to do so can have serious consequences to

[63] The investment required to set up personnel and material transport systems can be significant. In a deep hard-rock mine, such an investment would include, as a minimum, a service hoist and supply and personnel trucks and cars.

the operation, since personnel are required to carry out most production and service activities. Therefore, the efficiency of the personnel transport system and its ability to deliver personnel as required are critical to the efficiency and productivity of the entire ore-production system. Deep hard-rock mines still rely heavily on personnel for directly operating most production and support equipment, supervising equipment that runs more or less on its own, and coordinating and supervising the progress of the work. The problem is that even if existing transport systems work as planned, it still takes a long time for personnel to reach their respective work sites at the beginning of every shift. Similarly, extra time is needed for them to go to surface at the end of a shift. Thus, the actual available working time for **both** personnel and equipment that requires the presence of personnel to run can be drastically reduced. In fact, deep mine operators use experience-based factors to estimate the effective number of operating hours in a shift.[64]

4.5.2 Critical Materials and Supplies

With very few exceptions (the Kidd Creek Mine being one of them), all materials and personnel must enter and exit deep mines through the main shaft. In spite of its inherent long-term economic and operational disadvantages, financial restrictions and time limitations demand the use of a single, multi-purpose shaft for transporting muck, personnel, and materials in deep mines. Thus, the logistical aspects of the transport operation have to be specifically addressed in order to optimize shaft utilization. Napier and Stones (1990) noted that, in very deep and large-scale mines, up to 60% of the shaft time (cage operation) is dedicated to the transportation of materials (the remainder is used for personnel transport, shaft and rope and examinations, and breaks). Appropriate loading facilities and schedules, together with suitable containers and communication systems can increase the productivity of the delivery system, and reduce overall production costs by minimizing delays and maximizing the utilization of both personnel and equipment.

[64] For example, when calculating operating costs for project evaluation purposes, the operation studied in Chapter 5 reduces the effective working time of personnel by up to two hours and one half (in an eight-hour shift). This effectively reduces the productivity of personnel by 31.3%. It should be noted that a mandatory 30-minute break at mid-shift and 5-minute breaks for every 60 minutes of work are included in those 2.5 hours. Thus, it is assumed that about 1.5 hours are needed for personnel to reach their work sites, get ready to work, and return to surface.

The following items are considered as constituting the largest classes of materials or supplies required to support a modern underground hard-rock mine where sublevel stoping (or one of its variations) is the predominant mining method.

- *Explosives*

  In mine development, the explosive has to be delivered in appropriate quantities at the required time if the drill/blast/muck cycle is to be completed timely, and the productivity of the crews maintained. The explosive cost and the cost of transporting it to the development face are not critical if delays in creating the opening can negatively affect the operation (i.e., if they result in failure to meet production plans). As discussed in Section 4.4, alternate development technologies can potentially replace traditional drill/blast/muck methods. Such technologies would eliminate the reliance on explosives for mine development, and drastically could reduce overall materials distribution cost and related problems.

  For production blasting, the explosives must be delivered to the drilling horizon. In large mechanized mines, ramps connecting such horizons with the main access routes allow the distribution of explosives by specialized mechanized handling devices. If sublevels that do not allow the access of mobile equipment to the drilling horizons are used, the blastholes can be loaded from the level above through hose systems. Unlike the case of development, where rounds are fired almost every shift, typically production blasts involve large quantities of explosives, they are more carefully designed, and there is more time to plan and accomplish them. Thus, although the actual transportation cost is more significant, a one- or two-day departure from the originally scheduled date would rarely constitute a major problem. Finally, it is not expected that alternate rock-breakage technologies will replace explosives for ore production in the near future.[65]

[65] There are three main reasons for this. First, chemical energy delivery, as provided by explosives, is presently more efficient than mechanical energy delivery and results in lower per-tonne production costs. Second, although explosives generate significant amounts of heat when they detonate, such heat does not have a significant impact on the operation, since most of it is removed in conjunction with the gases also produced by the blast. Conversely, mechanical excavators generate less heat, but it is released while there are personnel and other equipment operating simultaneously, thus drastically increasing ventilation requirements in the working areas. Finally, continuous excavating machines require the horizontal slicing of ore, as carried out in non-bulk mining operations, a move not considered wise in highly stressed ground (see Section 3.3).

- ***Fuels and Maintenance Supplies***

The use of bulk mining methods and the virtual elimination of cut-and-fill stopes requiring captive equipment have simplified the transportation and delivery of maintenance supplies, fuel, and lubricants used by mobile diesel equipment. Maintenance areas are concentrated and underground warehousing and better communication systems have greatly reduced the down time resulting from waiting for parts. This is not likely to change in a deeper mine.

Changes in operating practices such as using smaller or more sophisticated mucking and haulage equipment would not drastically affect consumption rates, since these are directly related to actual work accomplished. A reduction in the number of mobile diesel units and a respective increase in the use of electric equipment would certainly alter diesel consumption patterns. As in the case of explosives for mine development, prompt delivery of these supplies is more critical to the operation than the actual cost of transporting them.

- ***Ground Support Materials***

These materials usually constitute major cost items in bulk mines. Furthermore, as mines mature and the working places become spread over larger areas in different levels, the cost of maintaining and supporting the excavations, as well as the corresponding materials transport and delivery costs, increase drastically. As in the case of explosives, it is pertinent to distinguish between development and production requirements. Prompt support of mine development openings is necessary, from both geotechnical and operational perspectives. Delays in installing the system(s) may not only affect the stability of the excavation (and increase the total cost of supporting it) but also can have a negative effect on the development and production programs. Systems used to support stopes in deep mines must be installed readily also, but without the urgency typical of mine development. Furthermore, with the exception of backfill, which in bulk mines is put in place after an entire block is mined, stope support systems such as cablebolts are typically installed well ahead of production activities. Thus, within certain limits, delays are not critical to the production cycle.

As mines become deeper, more complex (and, unfortunately, more expensive) support systems are needed. In particular, there has been a significant increase in the use of cable bolts, especially in secondary and tertiary blasthole stoping areas. The need for intensive support systems for both development and production openings is not likely to change with depth. The only potential relief might come from smaller development excavations that have lower consumption rates of support materials (on a per-metre basis), or from the use of lighter and easier to apply innovative support systems such as spray-on liners.

4.5.3 Alternatives

This review of the process of delivering personnel and materials to the underground work sites indicates that the actual transportation cost, although important, most likely will not be a critical factor in deep hard-rock mines. On the other hand, the financial cost involved in setting up such services can be high, and is a function of the depth of mining, mining method, and production rate. However, the question of affecting other aspects of the operation, including its overall efficiency, is already a major concern and its effects are expected to expand further as the depth of mining and the corresponding transportation distances and times increase.

It is firmly believed that this particular aspect of deep mining can benefit greatly from thoughtful development and introduction of innovative technologies. Indeed, automation and tele-operation have the potential of dramatically increasing equipment utilization and efficiency. For example, the productivity of longhole drills has been enhanced by tele-operation, which allows a single operator to be in charge of up to three drills simultaneously (Poole et al., 1998). The need for a comprehensive approach is highlighted by the fact that, although drill availability increased to 85%, utilization rose to 64% only. This was attributed to reliability and maintenance problems which, of course, were accentuated by the higher drilling rates. Also, machine set up and coordination problems were more noticeable due to the use of traditional planning and scheduling methods that did not account for the improved availability. In spite of such setbacks (which can be corrected through better operations management), the ability to allow a drill to continue operating in the absence of the operator or during a shift change constitutes a major gain in productivity. The next step is to identify other areas or activities that can be enhanced through similar procedures, and develop and introduce the required technologies.

The impact of automation on this particular factor is not easy to evaluate, mostly because it would involve a new technology that would have not been used previously in deep underground hard-rock environments. In fact, such an evaluation would heavily rely on subjective estimates of, and speculations about, equipment and personnel performance. For example, if equipment efficiency were improved by automation, this could lead to a reduction in the number of units required by the operation which, in turn, may result in decreased personnel and materials requirements. Fuel consumption rates may not change greatly as (within certain ranges) they are functions of work accomplished, not of the number of units. Another example could be the switch to smaller access and extraction openings that would result in the use of smaller mobile equipment: higher unit consumption rates and efficiencies would probably balance out.

4.6 Summary and General Discussion

Chapter 4 has reviewed pertinent literature and provided a theoretical discussion on the five factors constraining the operation and future development of deep underground hard-rock mines. This section summarizes briefly the main points raised during the analysis and theoretical evaluation of every factor. It also discusses the issues involved in the practical assessment of the relative importance of such factors, which is highly dependent on the particular features of a mine/project. This establishes the general framework for the three case studies that make up the final chapters of this thesis.

4.6.1 Summary - The Factors

Hoisting systems[66] used in deep Canadian mines are approaching rapidly their maximum hoisting depths.[67] Short of lobbying the government in order to modify existing regulations and reduce the

[66] The vast majority of the hoists used in Canadian hard-rock mines are single and double drum hoists. Indeed, Lazurko et al. (1988) reported that of 169 hoists operating in Ontario in 1987, 145 (86%) were drum hoists, and 24 (14%) friction hoists. Although multiple-drum, Blair-type hoists are the most suitable for deep hoisting, they have never been installed in a North American mine (Edwards, 1992).

[67] Shaw and Chaddock (1990) pointed out that, with current technology (i.e., double-rope, multiple-drum, Blair type hoists) the maximum depth for large-tonnage (6,000 tonnes per day) single-lift hoisting systems would be 2,850 m.

rope factors of safety,[68] the industry has two main options to provide long-term solutions to the vertical ore and waste transport problem:

- Install multiple drum, Blair-type hoists.

  Depending on hoisting rates, this would allow mining to depths of up to 3,300 m with single-lift shafts (assuming a static factor of safety of 5.0; see Greenway, 1990).

- Turn to *innovative* systems, such as vertical conveyors and slurry pumps.

  The advantages and drawbacks of innovative vertical transport technology were discussed in Section 4.1. The author is not aware of any hard-rock mine currently using non-traditional systems for ore/waste transport. However, they could drastically change existing mining systems, not just the transport component, and reduce (maybe eliminate) the depth and tonnage limitations imposed by rope hoisting.[69]

It must be remembered that, regardless of the type of hoisting system in place, improving the usage of the installed capacity is of the utmost importance. An essential aspect in this regard is the minimization of the amount of waste that must be hoisted, which can be achieved by efficient dilution control and underground disposal of development waste.

Horizontal transport of ore and waste is an important but often neglected component of the underground haulage system. In fact, although shaft hoisting systems can consistently achieve utilization rates of 90% to 95%, horizontal transport systems sometimes operate at rates as low as 25% to 30% (Ebersöhn and Visser, 1990). There is, therefore, ample room for improvement even within the limitations of conventional technology (i.e., track and trackless equipment).[70] However,

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[68] As noted by Sykes and Widlake (1990), the rope factor of safety is the most important single influence on the maximum depth of hoisting. In South African mines, where regulations stipulate a minimum static factor of safety of 4.5, it currently limits hoisting depths of large-tonnage mines to between 2,500 and 3,000 m.

[69] Slurry pumping with the three-chamber pipe feeder system, in particular, can address simultaneously the backfill, cooling, mine water pumping, ***and*** ore and waste transport problems (Worsley, 1990; Hindmarch, 1990).

[70] In their study, which was focused on a rail transport system capable of tramming over 640,000 tonnes of muck per month, Ebersöhn and Visser (1990) found out that the low utilization was due to lack of control, poor communications, and unreliability of the track and rolling stock. Furthermore, in discussing upgrading existing technologies to improve the operation of deep South African gold mines, Van Jaarsveld (1990) suggested that rail transportation systems are ideal candidates for such an overhaul. He noted that "***The general state of these systems as found in most mines today is nothing short of abysmal. Great losses are caused, and the level of expertise on most mines regarding rail transportation underground is unacceptably low.***"

onerous ventilation and cooling requirements coupled with very high operating weight to payload ratios make LHD-based systems expensive and inefficient. Electric LHDs are cleaner machines, but suffer from lack of flexibility and limited operating ranges. For similar reasons, and unless it is used in bulk-mining operations spread over large areas, track transport (i.e., with locomotives) is not suitable for level-haulage.

Continuous mining systems capable of fully replacing traditional drilling, blasting, mucking, and transport technology, have not been successfully deployed in hard-rock environments. They can potentially decrease direct production costs and improve productivity, ground control, and ventilation. However, continuous systems also are inflexible and expensive, and require a degree of fragmentation that is not typical of bulk operations. It should be noted that the design and planning of mines using non-traditional horizontal transport systems would have to address the operating requirements of the new technology, different from that of traditional equipment.

Ventilation is the only factor that, on its own merits, can severely restrict the development and operation of a deep hard-rock mine. Underground South African operators have been able to exploit narrow gold reefs[71] to depths of up to 3,600 metres. Viljoen (1990) argues that high thermal conductivities have made mining feasible at great depths in the Witwatersrand Basin. Canadian underground base and precious metal mines, however, not only face rocks with higher thermal gradients, but also make extensive use of bulk mining methods and mobile diesel equipment. They result in stringent air conditioning and ventilation requirements that could impose serious limitations at much shallower depths than in South African mines.[72]

Long pre-production development phases coupled with large tonnages of waste generated by ongoing development impose severe restrictions to deep mining operations and seriously affect their profitability. Properly addressing this factor requires sound understanding of the strong

[71] There are some exceptions to this rule. For example the thickness of the South Deep deposit, currently being developed by Western Areas Gold Mining Company, varies between 2 to 40 metres. The depth of this flat orebody is between 2,500 and 3,300 m below surface and the mining methods considered include room-and-pillar and vertical crater retreat (Tregoning and Barton, 1990).

interrelationships between mine development and mine design and planning. In fact, design issues affected by development include mining method, production rate, mine life, main access and extraction openings, inter-level spacing, and level development layout. The interaction between such issues is complex and demands an integrated approach difficult to implement without proper computer-assisted tools.

Mine development is tightly integrated with vertical and horizontal ground transport. Thus, it can also benefit greatly from the introduction of innovative technology that accelerates the excavation process, reduces the direct development cost, and minimizes the production of waste. Regardless of the type of technology employed, however, the shape (i.e., cross-section) and stability of the resulting excavations, as well as their maintenance requirements and ventilation characteristics, will determine its applicability to deep mining environments. New technology may allow a drastic increase in both the speed of level development and ore extraction. This would certainly demand the use of different mining sequences in order to cope with the higher rates of stress released as a result of the increased (and concentrated) stoping activity in a particular level.

There is no doubt that, in deep mines, the cost of delivering personnel and materials to the working places increases in absolute terms. However, since other cost items increase with depth of mining also, it is likely that its relative significance is either maintained or reduced. On the other hand, this factor has two important effects on deep operations that rely on a single multi-purpose shaft for transportation. First, and because personnel are required for the operation and control of most of the equipment used in a mine, proper coordination and execution of the transportation schedules are critical to accomplishing production targets. Second, even when personnel and materials are delivered as planned, the long transportation routes and the bottleneck created by the single shaft result in ever decreasing personnel and equipment productivity.[73] This

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[72] It is also argued that high thermal gradients, resulting from norite and gabbro formations that act as good insulators, have prevented the same level of deep mining development (below 1,500 metres) in the Bushveld Complex (Viljoen, 1990). Similar rock formations found in the Sudbury area may partially explain ventilation problems encountered at moderate depths (see also geothermal gradient graph in Hartman et al., 1997, p. 591).

[73] Shaw and Chaddock (1990) discussed a 2,850-metre deep South African gold mining project expected to produce 180,000 tonnes of ore per month. They estimated that, on average, each worker would have to spend 90 minutes of every eight-hour shift travelling to and from his/her workplace (about 19% of the shift). It was also noted that the coordination of the various transportation systems involved probably would be complex and difficult to manage.

is an area where technology, for example, by allowing equipment to operate unattended during shift changes or lunch breaks, could greatly enhance the effectiveness of existing production systems.

4.6.2 Practical Assessment of the Constraints

It has been noted already that the relative importance of each of the factors discussed in this chapter is highly site-specific. For instance, mines in Utah and Arizona experienced extremely hot environments at depths of less than 1,000 m, whereas South African mines only reached the same conditions at about 3,000 m below surface (Hartman et al., 1997, p. 591). The stress regime, structural geology, and physical characteristics of the orebody are unique features that dictate important design and planning parameters such as mining method, level development layout, and extraction equipment size. They also are beyond the direct control of the mine operator.

Therefore, the first and most substantial reason for assessing the constraints is to identify their relative significance in the light of the envisaged mining operation. It must be stressed that the assessment is concerned with a particular type of operation: changes in mining method, transport technology, or production rate, for example, may drastically modify the "ranking" of the factors. The identification of the critical ones helps in establishing objectives for mine design and planning, and in effectively addressing them through the mining strategy. In theory, once a factor has been dealt with, it ceases to be an issue anymore, although the measures taken to minimize its effects on the operation will continue to affect it for the entire mining life.[74]

The second reason has to do with the dynamic nature of underground mining: it is necessary to anticipate changes in the way the factors will continue to constrain the operation as it evolves over the years. For instance, even if ventilation were not an issue when a mine was first started, open stopes (without backfill) and/or higher water inflows may render the initial system ineffective and demand a new approach. Similarly, a strategy that called for intensive mining near

[74] For instance, if heat is the most pressing issue in a deep project, once the air conditioning system capable of dealing with it is designed, and the economic feasibility of using such a system is determined, heat becomes a non-issue. This does not mean that air conditioning is not important anymore, but that this particular aspect of the operation has already been optimized, and that another constraint can now be tackled and similarly optimized.

the production shaft and benefited from short horizontal transport routes may have to be radically changed to cope with longer haulage distances after the immediate ore is depleted. Although it is impossible to predict accurately the evolution of a deep mining operation, the exercise will certainly help in understanding its future development and the likely changes in cost structure.

Regardless of its framework (i.e., explicit versus implicit; formal versus emergent; etc.), the application of strategic concepts to underground mining requires careful consideration of the factors identified as critical to deep operations (see Chapters 2 and 3). In other words, studying and monitoring them must be an ongoing exercise. However, the evaluation of the issues affecting deep mining is most likely to take place ***only*** when:

- the possibility of developing a new deep mine is being considered;
- an already deep mining operation is deciding on a deepening project; or,
- an existing deep mine is having economic or operational problems.

In the first case, geological and geotechnical data are likely to come from several exploration programs that, due to the depth of the orebody, in most cases included underground drilling. Unless a similar ore deposit is mined in the area, such data constitute the only source of information for the feasibility study. Thus, the degree of uncertainty regarding the deposit itself and the mining environment is high. Depending on the operator, decisions in the areas of mining methods and technology may not be based on first-hand experience, in which case the risk component concerning those two areas would be also large. The evaluation of the deep mining issues, an important part of the main feasibility study, is based on estimates obtained through *first principles* calculations.

As discussed in Section 3.2.2.2, deepening projects are risky because, in addition to venturing into a completely new area, they usually involve new/modified mining methods and technologies. In this case, however, the evaluation of deep mining issues and the proposed expansion may benefit from the experience gained in the ongoing operation. Such an experience could play two important roles: first, detailed knowledge of the local operating conditions would have been acquired. Regardless of the extent (i.e., depth) of the expansion, the main features of the orebody and surrounding rock mass in the deeper zones would most probably resemble those of the currently active mining areas. In the worst case scenario, any extrapolation of current conditions would be based on sound grounds. The operation would be quite capable, for example, of determining if the methods and equipment in use could be applied in the deep project with reasonable chances of success. Second, a wealth of operating and cost data would have been accumulated, which, in theory, could help the operator in deciding the feasibility of mining the deeper ore. The problem with such data is that not always are collected and stored properly, but with the objective of fulfilling accounting or other non-managerial requirements. Thus, poor decision-making could result if the data were used for evaluating a deep project.

The last case requires careful analysis of the reasons for having such problems. Once it has been determined that they are related to depth of mining only, conditions similar to that of the deepening project would prevail. In other words, the analysis of the current situation, and the eventual solutions to be proposed, would be strongly determined by the existing operating data and conditions. Due to the ongoing economic and/or operational problems and the need to embark on a major investment program, the chances of success in this case would be a function of the financial capability of the mine operator.

4.6.3 Case Studies

The emphasis on the practical aspects of deep hard-rock mining demands the analysis of cases in order to demonstrate the validity and applicability of the concepts presented in this thesis. This was accomplished through two case studies. The first case study is based on actual operational and cost data supplied by a Canadian deep mine which are adequately disguised in order to protect its confidentiality. The second case study was created using an artificial (and very simple) orebody model, operating data calculated using *first principles*, and cost information obtained from mine suppliers, cost surveys, and other publicly available data.

5. Case Study 1: Lynx Brook Deep Mine

This case study investigates strategies for the deepening of the Lynx Brook Mine from the 7200 Level to the 7800 Level. It focuses on several equipment and inter-level spacing options, as well as on operating aspects such as dilution, mining sequence, and production rate. The case study is based on an actual Canadian mining operation. The information has been disguised in order to protect its confidentiality. It should be pointed out that this practice has affected neither the validity nor the applicability of the analysis.

5.1 Introduction

The information that serves as the basis for this case study was furnished by the operator of the Lynx Brook Mine. Two site visits provided opportunities to discuss several important issues with the mine's personnel and, thus, focus the analysis.

The objective of the case study is to determine a mining strategy that optimizes operating conditions and ensures the long-term profitability. The study is based on general development and production plans, as well as operating and capital cost estimates. Instead of working out the details of the mine design and plan, it concentrates on the relative advantages and disadvantages of every option considered, which are then analyzed from strategic viewpoints. It is hoped that the conclusions drawn from the case study demonstrate the applicability of the concepts presented throughout this thesis.

5.2 The Existing Operation

Over the years, technological development and changes in mining conditions led to the adoption and/or modification of several mining methods including undercut-and-fill, mechanized cut and fill, shrinkage, blasthole stoping, and post pillar cut and fill. Currently, mining activity at Lynx Brook is concentrated between the 4500 and 7400 Levels (between 1,350 and 2,220 metres below surface).

5.2.1 Geological and Geotechnical Features

This section is a general introduction to the geological and geotechnical characteristics of the Lynx Brook orebody and surrounding rock mass. Sound understanding of these aspects will ensure that the analysis is properly focused and the results are applicable, even at the conceptual level. Major sources of information have been internal reports as well as electronic spreadsheets and *AutoCAD* drawings provided by the operator.

5.2.1.1 Summary Regional and Local Geology

The base metal ore of Lynx Brook Mine is associated with a large mafic intrusion in a footwall composed of felsic igneous rocks and metavolcanics. The intrusive dips at about 60° NW but can vary locally from 90° to 30°. Diamond drilling has confirmed that the orebody extends to a depth of 3,000 metres. The main source of ore in the region concerned with this research (from the 2,160-metre to the 2,340-metre elevation) is a massive sulphide lens found at the contact, with a large nose of massive sulphide that stretches out into the footwall. The footwall rocks close to the orebody consist of medium-grained granite containing a high proportion of mafic minerals. This rock unit is interspersed with granite gneiss and fine-grained metavolcanic rocks. Since these rocks have been metamorphosed and the contacts fused, the rockmass is relatively homogeneous.

5.2.1.2 Geomechanical Issues

There are three main structural domains: the hangingwall, the sulphide domain (the mineralized zone and the adjacent footwall), and the footwall region. Each of these domains can be considered isotropic. The geomechanical properties of the rock mass at Lynx Brook can be considered as good (Pareja and Pelley, 1996). Furthermore, Baker et al. (1996) carried out Mathews analysis and numerical modelling (Examine 2D program) with the same geomechanical data. They concluded that only the footwalls of 30-metre high stopes would require some support if they were more than 10 metres long. The walls and backs of longer and taller stopes would require ground support, but nothing unusually expensive.

5.2.1.3 Summary Economic Geology

In general, the ore tends to be quite homogeneous within the mineralized sulphide zone. In upper levels, there was a definite zoning pattern with a high-grade footwall and a gradational lower grade hangingwall. At Lynx Brook Deep, the gradational hangingwall has disappeared and the large nose of massive sulphides that used to extend into the footwall at higher elevations has become only a minor feature of the mineralization. The main orebody becomes significantly thinner and elongated with depth, losing its three-dimensional character and adopting a pseudo-tabular nature.

For the purposes of this thesis, it has been assumed that the metal-bearing minerals are uniformly distributed over the entire orebody. This maintains the confidentiality of the data and eliminates a factor not relevant to the present project (variability of ore grades). The orebody outlines have been obtained from AutoCAD drawings provided by the operator and it is assumed that the ore/waste contacts are clearly defined. For instance, Figure 21 shows the main orebody outline at the 7350 Level elevation. It can be seen that the nose in the footwall is still quite significant.

Using average *in situ* grade figures and June 1998 metal prices, the ore of Lynx Brook Deep has an *in situ* value of US $ 193.8/tonne. The average ore and waste specific gravity are 4.2 and 3.0, respectively.

Tonnages for every level can be calculated based on orebody outlines such as the one shown in Figure 21. The formula employed in the calculation procedure is as follows:

$$T_i = \left[\frac{S_{i-1} + S_i}{2} * h_{i,i-1} \right] * \delta$$

where:

- T_i : tonnage of ore at level i (in tonnes)
- S_{i-1} : area of orebody outline at level i-1, i.e., at the level above i (in square metres)
- S_i : area of orebody outline at level i (in square metres)
- $h_{i,\,i-1}$: vertical distance between levels i and i-1 (in metres)
- δ: *in situ* density of the ore (in tonnes per cubic metre)

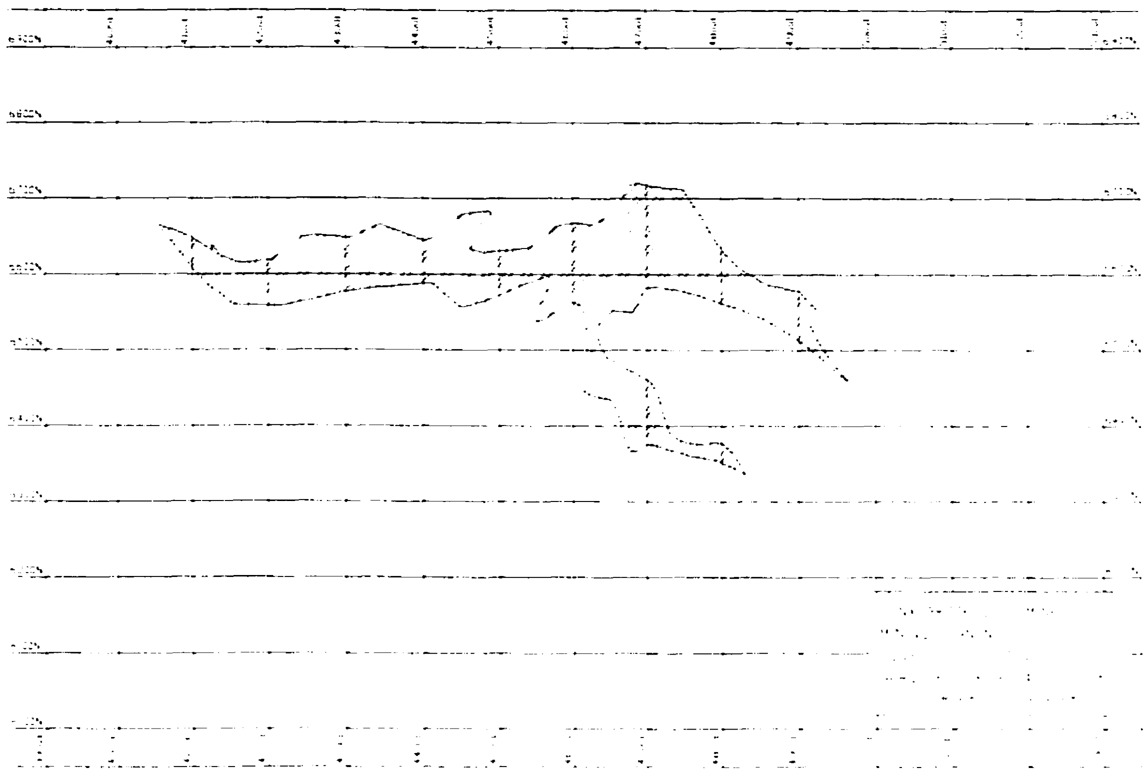

Figure 21: Lynx Brook Deep - Ore outline at 7350 level elevation

The use of the above formula carries two implicit assumptions regarding the continuity and geometry of the mineralization. First, it is assumed that the ore varies continuously between levels (i.e., a point in the upper orebody outline corresponds to a unique point in the lower outline). Second, that such a variation follows a step function: the average grade and horizontal area of the orebody suddenly change halfway between levels. The results are shown in Table 1. It is evident that the orebody is becoming narrower with depth: at the 7800 Level the area of the orebody outline (plan view) is only 49.6% of that of 7400 Level. The operational implications of a narrower orebody are serious and will be specifically addressed in a later section.

5.2.2 Mining

The overview of current mining practices includes mine design and mining method, access and ore transport, ground support, and ventilation. The objective is primarily to gain a thorough understanding of the operating conditions so that the case study is properly focused. Only issues pertinent to the general purpose and scope of the thesis will be addressed.

Table 1. Lynx Brook Mine - Ore reserves for 7250-7800 levels

Level	Area (m^2)	Volume (m^3)	Tonnage (tonnes)
7190	7,018.9	---	---
7250	6,812.7	126,477	531,202
7300	6,697.8	102,950	432,389
7350	7,557.3	108,623	456,219
7400	8,112.3	119,402	501,490
7450	7,958.7	122,461	514,335
7500	7,892.1	120,783	507,288
7550	7,269.2	115,529	485,223
7600	6,567.9	105,439	442,843
7650	6,401.5	98,827	415,073
7700	5,680.0	92,061	386,656
7750	4,685.5	78,985	331,735
7800	4,026.1	66,382	278,806
Total	**86,679.9**	**1,257,919**	**5,283,258**

5.2.2.1 Mine Design and Mining Method

Although cut and fill and undercut-and-fill mining methods were used in the recent past at Lynx Brook Mine, a variation of the well known *vertical crater retreat* (VCR) method (Lang, 1982; Osborne and Baker, 1992), currently accounts for most of the ore produced. In fact, in 1994 about 95% of the ore extracted came from VCR stopes (Ernst & Young, 1995). The evolution of mining methods and corresponding sequencing strategies at Lynx Brook Mine are well documented elsewhere (Pelley, 1994) and will not be discussed here.

Access to the deeper areas of Lynx Brook Mine is obtained through a 2,175-metre deep shaft. Internal ramps connect the deep levels in areas of mining activity. In June 1995, the 7200 Level was the deepest active production level. In general, the VCR method involves drilling sub-vertical blastholes from the topsill level and blasting 2.4 to 3.0-metre thick slices of ore progressively from the bottom of the panels. Depending on the configuration of the orebody, the horizontal cross-section of a panel can vary from 4.5 x 18-m to 12.6 x 18.0-m. Blastholes are 16.5

to 20.0 cm (6½ to 8 in.) in diameter and are loaded either with ANFO or slurry. A typical deck incorporates a 34.1-kg charge (powder factor ranges between 0.40 – 0.60 kg/tonne). Sequential blasting is used for larger VCR blasts in order to reduce vibration and the resulting wall damage.

Hydraulic backfill is used to provide support to stope walls and is required by the mining sequence. Backfill material consists of mill tailings with blast furnace slag as a binding agent (Portland cement is also used). Currently, high-density paste fill is being tested (but has not been formally adopted). The mining sequence dictates that mined out panels are filled before adjacent panels are blasted, and to mine away from backfilled areas to avoid the creation of pillars that could dangerously load up. The general mining sequence at Lynx Brook Mine is as follows:

a. A rectangular panel is first mined in (approximately) the centre of the orebody.
b. After backfilling the first panel, adjacent panels are extracted so that mining *radiates outwards* from it, panel by panel, creating a diamond-shaped slot.

Known problems with this sequence include:

1. Both hangingwall and footwall access are required to expand the slot in four directions. This results in long and more expensive development programs and seriously restricts the productivity of mining areas.
2. The sequence is difficult to follow when the orebody loses its three-dimensional nature (i.e., when it becomes thinner) or there are significant amounts of internal waste that must be left behind.
3. Some ore is lost in *wedges* left at the topsills and bottomsills, resulting in reduced *planned* ore recovery.
4. The level below only can start production after the above-mentioned slot has been established across the entire orebody (from hangingwall to footwall).

5.2.2.2 Mine Access and Ore Transport

Personnel and materials are brought to the active mining areas via the main shaft. There is a drift connection at 7000 Level and a loading pocket at 7080 Level. As mentioned above, several ramps provide access and ore/waste transport routes. The shaft is 6.3 metres in diameter and uses wooden guides. Stations and drift connections have been excavated every 200 metres. The shaft has been equipped with two double drum hoists (5.4 metres in diameter, with face widths of 2.03 metres), a cage, two 11.8-tonne skips, and scapeway and service compartments. The cage motor is rated at 2,800 kW (3750 hp) and the skip motor at 5,220 kW (7000 hp).

Load-haul-dump units are used to muck ore and waste from production areas and development headings. Since ore and waste passes are generally close to dumping areas, low profile trucks are not required for haulage. Although several locomotives are used at Lynx Brook to haul broken rock in the upper levels (down to 7000 Level) only trackless equipment is employed currently in the deeper section (7200 and 7400 levels). This situation will not change in the future as long as loading pocket 7080 is the deepest skip-loading facility at Lynx Brook Deep.

5.2.2.3 Ground Support

In addition to backfill (discussed above), the ground support system at Lynx Brook comprises the following elements:

- ***Rockbolts and screen***
  This is the primary ground support method in development headings and VCR topsills and bottomsills. Several rockbolting patterns have been developed to address expected ground conditions for diverse types and sizes of development headings.

- ***Cable bolting***
  Cable bolts are installed in excavations with spans greater than 7.2 metres or where ground conditions demand additional support. Twin cables are normally installed but the final design and pattern will depend on the specific conditions of the area being supported.

- ***Shotcrete***
  This is the primary support method for development headings through rib pillars driven with backfill in the walls and/or back. It is also a secondary support system when applied on top of bolts and screen in VCR brows.

- ***Other***
  Other support methods, not used systematically at Lynx Brook, include cable lacing, and timber and steel sets.

Rockbolts must be installed to within 0.6 metres of the active face, whereas screen is to be installed within 0.9 metres of the face. This, together with strictly observed distressing practices, significantly affects the speed of mine development since the normal cycle must include one more time-consuming phase.

Mined-out panels/stopes are backfilled before starting mining activities in adjacent panels/stopes. In these cases, the backfill must be reinforced with Portland cement, limed slag, or some other cementing agent (the minimum cement/limed slag to backfill ratio is 1:30). Backfilled panels must cure for at least three days if reinforced with cement and for at least seven days if poured with limed slag, before mining begins immediately adjacent to them.

5.2.2.4 Ventilation

Current installed ventilation capacity at Lynx Brook Mine is 1,200,000 cfm. Most fresh air is drawn from surface through broken rock in an open pit. By acting as a heat exchanger, broken rock warms the air in the winter and cools it in the summer, maintaining it at a temperature of about 3° C all year round. The rock gradient at Lynx Brook is 1° C per 54 metres with a virgin rock temperature of 42° C at 7000 Level (2,100 metres below surface).

Two sets of fans acting in parallel supply air to active mining areas down to 7200 Level. The main exhaust system extends from Level 7150 to surface. Three exhaust fans operating in parallel have been installed on surface at the collar of a 1,500-metre deep shaft. The existing ventilation system is capable of removing about 15,000 BTU/hour (4.4 MW) of heat. About 14% of the heat generated is produced by the compressors installed in the 6000 Level, while diesel equipment, wall rock, water, and other electrical equipment (fans, pumps, crushers, etc.) account for the rest. It is generally accepted that the system was not designed to accommodate the underground compressors. Current ventilation air shortfall has been estimated at about 120,000 CFM.

In spite of the natural heat exchanger, ventilation and air conditioning at Lynx Brook are difficult and expensive due to the multi-level operation and the numerous mined-out workings found in the upper levels. The result is significant air losses, and complex and long ventilation networks.

5.2.3 Analysis of Production Cost Structure

The strategic significance of production cost to base and precious metal-mining companies has been discussed extensively in previous chapters. Adequate grasp (and, eventually, management) of the cost structure of deep underground hard-rock operations is critical to their survival. This section analyzes Lynx Brook Mine's cost data provided by the operator. The objective of the exercise is threefold: first, to gain an understanding of the operation's cost structure; second, to determine the adequacy of current cost collection methods for decision-making; and, finally, to investigate changes in the cost structure as depth of mining increases.

5.2.3.1 Introduction

Main sources of cost information were Ernst & Young's *Underground Mining Performance Benchmarking Questionnaire* (Ernst & Young, 1995), and two internal reports, one providing detailed operating cost information and the other one focused on mine management data. All cost information used in this chapter was disguised to protect its confidentiality.

In agreement with its benchmarking objectives, Ernst & Young's questionnaire was concerned with general aggregated cost and production information. In fact, its layout was designed so that it would fit different underground mining methods and facilitate comparisons. However, some inconsistencies regarding labour cost and its components were identified. For example, it was not clear from the questionnaire if the mine services labour cost and the management, administration, and support labour costs were considered as part of labour cost. Given the degree of aggregation of the cost data, it was assumed that they were part of the labour cost. Thus, the mining cost can be broken down as shown in Table 2. The mining labour cost component is very high (47.2%) and total labour cost (mining plus maintenance labour cost) is 58.8%, a figure higher than the corresponding average for American mines in the late 1950's (see Footnote 19 on page 32).

The inadequacy of Ernst & Young's cost forms is further illustrated by Table 3, which shows a breakdown of the mining labour cost. First, it is debatable that management and administration labour cost (*sic*) should be part of mining labour cost. After all, they are normally considered components of the management/overhead cost. Second, mine services labour cost, accounting for

22.6% of mining labour cost, is suspect. When stope support labour cost is added to this figure, (which is believed to be composed mostly of level upkeep), auxiliary and service tasks are responsible for about 40% of mining labour cost. This is in spite of the fact that management and administration costs are included in the total.

The degree of aggregation and the above-mentioned inconsistencies greatly reduced the usefulness of Ernst & Young's questionnaire. For the purposes of this case study, it was used as a reference only, and to confirm results obtained with the raw data provided by the operator.

5.2.3.2 Main Mining Cost Components

Table 4 is a summary of 1994 labour and production costs for Lynx Brook Mine. Costs included cover the mine production process, from orebody delineation and mine development to primary crushing, hoisting, surface handling and loading, and transportation to the mill. Ore processing costs are not included. The seven largest production cost items, shown in Table 5, account for 74.7% of the total mine production cost (ore delivered at the processing facility). Similarly, 64.6% of the *mining cost* (see Footnote 78 on page 118) is comprised of its six largest components. Only two of the items listed in Table 5 correspond to the actual mining cycle (crushing, hoisting and conveying, and mucking/removal of ore and waste). The largest item, overhead cost, comprises ore transportation to the mill (7.5% of total overhead), amortization and depreciation (50.0% of total overhead), and divisional overhead, (42.5% of total overhead). It accounts for 28.6% of total production cost and, as a comparison, is equal to 40.1% of total mining cost.

Table 6 shows 1994 operating and repairs costs. Operating costs have been broken down into operating labour, maintenance labour and supplies, and other costs. Repair costs have been broken down into labour, materials and supplies, overheads, and other costs. Pertinent operating and repair accounts have been added to facilitate the evaluation of the relative importance of every cost item. Although mine management is the second largest cost item, the reports do not provide details about its composition. In fact, the *detailed* report shows only three entries under this heading, including one for $ 4.7 million.

Table 2. Lynx Brook Mine - Total mining cost data[75]

Item	$	%
Mining labour	26,235,571	47.21
Mining consumables	9,360,500	16.84
Maintenance labour	6,420,700	11.55
Maintenance consumables	4,297,900	7.73
Contractors	1,396,200	2.51
Other[76]	7,867,800	14.16
Total Mining Cost	**55,578,671**	**100.00**

Table 3. Lynx Brook Mine - Breakdown of mining labour cost[77]

Item	$	%
Production drilling	1,020,130	3.89
Production blasting	492,156	1.88
Stope support	4,110,400	15.66
Mucking	2,203,922	8.40
Backfilling	1,072,800	4.09
Ore and waste hauling	1,914,100	7.30
Ore and waste crushing/breaking	556,400	2.12
Ore and waste transport to surface	1,354,785	5.16
Mine exploration drilling	76,900	0.29
Mine development	2,218,800	8.46
Mine services	5,934,607	22.62
Management, administration and support	5,280,571	20.13
Total mining labour cost	**26,235,571**	**100.00**

[75] Source: Ernst Young, 1995.

[76] Other costs include electrical power, dewatering, and ventilation costs.

[77] Source: Ernst Young, 1995.

Table 4. Lynx Brook Mine – Labour and cost summary for 1994

Cost Item	Manshifts		Cost ($ x 1000)		Cost Correction ($ x 1000)		Total Cost	
	Operating	Maintenance	Labour	Other	In	Out	$ x 1000	$/tonne
Mine development & delineation drilling	6,173	171	2,539.9	1,712.6	95.0	191.8	4,155.7	4.71
In-the-hole (production) drilling	2,525	938	1,199.1	1,148.9	96.0	---	2,444.0	2.77
Other mining methods	33	---	13.4	41.7	---	---	55.1	0.06
Blasting and explosives	1,239	2	492.5	1,561.8	---	238.9	1,815.4	2.06
Ore & waste mucking/removal	6,925	3,524	3,575.2	2,144.5	390.6	329.0	5,781.3	6.56
Backfill	3,617	749	1,314.4	2,399.0	111.7	---	3,825.1	4.34
Tramming ore and waste	5,635	1,342	2,357.4	461.6	168.8	---	2,987.8	3.39
Crushing, conveying, hoisting & surface handling	9,513	4,847	4,479.2	1,269.5	551.3	---	6,300.0	7.15
Underground upkeep and support	13,104	1,214	4,519.5	2,211.4	132.7	401.4	6,462.2	7.33
Drainage and pumping	944	886	533.8	579.9	551.9	---	1,665.6	1.89
Ventilation and heating	880	1,049	600.2	1,059.8	133.0	---	1,793.0	2.03
Electric power	28	219	90.5	5,070.6	27.5	---	5,188.6	5.89
Water supply and compressed air	893	127	327.3	419.7	17.8	---	764.8	0.87
Surface services, safety and plant protection	2,241	366	753.1	753.5	696.4	---	2,203.0	2.50
Other equipment repairs	183	1,559	557.4	1,057.1	188.9	529.5	1,273.9	1.45
Mine management	11,894	---	4,702.5	27.8	1,749.8	---	6,480.1	7.35
TOTAL MINING COST	**65,827**	**16,993**	**28,055.4**	**21,919.4**	**4,911.4**	**1,690.6**	**53,195.6**	**60.35**
Overheads, depreciation, R&D, surface transport	16,263	3,322	3,215.8	16,740.1	5,134.7	3,747.5	21,343.1	24.21
TOTAL PRODUCTION COST	**82,090**	**20,315**	**31,271.2**	**38,659.5**	**10,046.1**	**5,438.1**	**74,538.7**	**84.57**

Table 5. Lynx Brook Mine - Seven largest cost items in 1994

Item	Cost		Percentage of Cost of	
			Production	Mining
	$ x 1000	$/tonne	(%)	(%)
Overheads, services, depreciation, R&D, transport	21,343.1	24.21	28.63	N.A.[78]
Mine management	6,480.1	7.35	8.69	12.18
Underground upkeep and support	6,462.2	7.33	8.67	12.15
Crushing, conveying, hoisting & surface handling	6,300.0	7.15	8.45	11.84
Ore & waste mucking/removal	5,781.3	6.56	7.76	10.87
Electric power	5,188.6	5.89	6.96	9.75
Mine development & delineation drilling	4,155.7	4.71	5.58	7.81
Total	**55,711.0**	**63.21**	**74.74**	**64.61**

Underground upkeep and support amounted to $ 6.46 million in 1994 (12.1% of mining cost). It is mostly composed of level upkeep and maintenance, the operation of auxiliary mobile equipment (scissors lifts, personnel carriers, etc.), and rockbolts and screens. A reduction in the number of levels in simultaneous operation, achieved through adequate mine development and extraction sequencing, could reduce greatly this cost item. This should be a high priority for Lynx Brook Deep, since such a cost tends to escalate rapidly with depth due to increasingly poorer ground conditions and reduced availability (i.e., productivity) of equipment and personnel.[79]

Crushing, horizontal ore and waste transport, hoisting, and surface handling constitute the fourth largest item: about $ 6.30 million, or 11.8% of mining cost. As long as a bulk mining method is employed, crushing will be needed in order to reduce the muck to a manageable size. Conveying and hoisting, on the other hand, can be replaced by other bulk transportation methods (see Chapter 4). In the particular case of hoisting, the system does not have much room for productivity improvement. Indeed, even under current conditions (i.e., skipping from 7080 level,

[78] The mining cost accounts for all costs incurred to deliver ore at the surface shipping facility. It does not include division overheads, depreciation, research and development, and surface transportation to the processing facility, items that are included in the total production cost.

[79] At the time the data were compiled, the mine was in the process of recovering pillars left in the upper levels. It could be argued that this distorted somewhat the cost structure. On the other hand, the economic evaluation of pillar recovery must consider the need to maintain simultaneously primary and secondary mining areas.

11.8-tonne skip payloads) the static factor of safety is already close to the minimum of 5.0 required by Ontario legislation.[80] In order to maintain current production levels from deeper ore sources using the existing installations, the safety factor would have to be lowered even further.

The fifth largest cost component in 1994 was ore and waste mucking ($ 5.78 million, 10.9% of total mining cost), which at Lynx Brook is exclusively carried out with LHD units. The importance of considering maintenance and repair costs when addressing the problem of deep mining is underscored by the fact that $3.12 million (54.0% of mucking cost) were spent in repairs. It is possible through technological innovation to reduce (even eliminate) the direct labour component of some unit operations such as mucking and tramming in order to increase overall productivity. However, it is not clear how this would reflect on the corresponding maintenance/repair costs. Would more sophisticated systems require additional maintenance and control? Would they depend on increasingly more sophisticated and expensive support equipment? These questions are difficult to answer presently, mainly because similar systems have been used almost exclusively in other segments of the underground mining industry (i.e., coal, potash, etc.), not in deep hard-rock operations. As the depth of mining increases, and unless operational changes are made, the mucking component of mining cost will keep on expanding.

Not surprisingly, electric power is also a large cost item: $5.2 million (9.8% of total mining cost). A deeper operation of similar production levels would probably demand higher power consumption rates, due to increasing hoisting, compressed air, and ventilation and refrigeration requirements. It would have been interesting to be able to disaggregate the electric power cost into its various constituents so that they are added to every major item in the mining cost structure. The report shows a single entry for the electric power purchased by the entire operation.

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[80] Assuming that the skip weight is 5/8 of payload, the static safety factor (SF) at the head sheaves with the skip fully loaded at the 7080 loading pocket is calculated as follows (in English units, Edwards, 1992):

$$SF = \frac{\text{rope strength}}{\text{weight of rope} + \text{weight of skip} + \text{skip payload}}$$

$$SF = \frac{530{,}000 \text{ lb}}{[8.55 \text{ lb/ft} * (7080 \text{ ft} + 270 \text{ ft}] + [0.625 * 13 \text{ tons} * 2000 \text{ lb/ton}] + [13 \text{ tons} * 2000 \text{ lb/ton}]} = \frac{530{,}000 \text{ lb}}{105{,}093 \text{ lb}} = 5.04$$

Table 6. Lynx Brook Mine - Operating and repair cost for 1994

Item	Operating Costs ($)				Repair Costs ($)				Total ($)
	Op. Labour	Man. Labour	Mat./Other	Total	Labour	Mat./Sup.	O/H-Other	Total	
Mine development & delineation drilling	2,475,158	13,597	1,527,333	**4,016,088**	40,402	61,326	37,717	**139,445**	**4,155,533**
In-the-hole (production) drilling	839,455	---	593,430	**1,432,885**	359,598	403,395	247,963	**1,010,956**	**2,443,841**
Other mining methods	13,434	---	41,758	**55,192**	---	---	---	—	**55,192**
Blasting and explosives	492,156	356	1,322,942	**1,815,454**	---	---	---	—	**1,815,454**
Ore & waste mucking/removal	2,203,922	8,116	447,419	**2,659,457**	1,363,157	1,327,489	431,160	**3,121,806**	**5,781,263**
Backfill	1,072,910	40,292	2,366,540	**3,479,742**	201,228	69,654	74,441	**345,323**	**3,825,065**
Tramming ore and waste	1,913,960	8,720	168,101	**2,090,781**	434,768	289,265	172,974	**897,007**	**2,987,788**
Crushing, conveying, hoisting & surface handling	2,868,864	158,448	452,181	**3,479,493**	1,447,091	658,636	714,738	**2,820,465**	**6,299,958**
Underground upkeep and support	4,087,290	83,379	1,649,588	**5,820,257**	348,808	185,035	108,056	**641,899**	**6,462,156**
Drainage and pumping	269,109	47,570	814,501	**1,131,180**	217,094	205,811	111,507	**534,412**	**1,665,592**
Ventilation and heating	203,495	72,549	865,317	**1,141,361**	323,242	209,134	119,169	**651,545**	**1,792,906**
Electric power	5,460	8,560	5,018,442	**5,032,462**	76,455	45,294	34,345	**156,094**	**5,188,556**
Water supply and compressed air	58,469	5,713	300,381	**364,563**	263,099	61,063	75,994	**400,156**	**764,719**
Surface services, safety and plant protection	639,139	56,256	1,342,551	**2,037,946**	57,712	59,950	47,470	**165,132**	**2,203,078**
Other equipment repairs	---	---	(240,398)	**(240,398)**	557,362	681,785	275,103	**1,514,250**	**1,273,852**
Mine management	352	---	6,479,780	**6,480,132**	---	---	---	—	**6,480,132**
TOTAL MINING COST	**17,143,173**	**503,556**	**23,149,866**	**40,796,595**	**5,690,016**	**4,257,837**	**2,450,637**	**12,398,490**	**53,195,085**
Overheads, depreciation, R&D, surface transport	1,134,166	885,419	18,874,998	**20,894,583**	155,295	269,810	23,476	**448,581**	**21,343,164**
TOTAL PRODUCTION COST	**18,277,339**	**1,388,975**	**42,024,864**	**61,691,178**	**5,845,311**	**4,527,647**	**2,474,113**	**12,847,071**	**74,538,249**

The seventh largest cost component is mine development, which in 1994 was $ 4.2 million or 7.8 % of total mining cost. It includes principally labour and supplies costs incurred during the development of VCR sills. Only a drastic change of mining method or the introduction of an entirely new excavation system would be able to reduce the cost of mine development. Although mine development produces important amounts of ore, large and complicated mine development programs delay actual production, reduce the flexibility of the operation, and impose severe restrictions to mine sequencing and production plans (see Section 4.4).

5.2.3.3 The Cost of Labour

Significant attention has been given recently to the labour cost of underground mining operations. Subjects discussed include its impact on profitability and efficiency, as well as the need to reduce it through better process design, organizational structures, and the introduction of new technologies (Poole et al., 1996; Scoble, 1994; Holmes, 1993; Udd, 1993). It will be interesting, thus, to analyze and quantify the impact of labour on total production cost at Lynx Brook Mine.

Table 7 shows aggregated 1994 operating and repair cost figures for Lynx Brook Mine as percentages of their respective total values. It can be seen that direct operating labour cost accounts for 32.2% of total mining cost. The corresponding percentages for maintenance and repair labour cost, operating supplies and other, and repair supplies are 11.7%, 43.5%, and 8.0%, respectively. Although *direct labour* cost is still significant when compared with similar figures from other industries such as manufacturing, it is not nearly as high as it has been reported in the past (Baiden, 1993a; Gentry, 1976). When divisional overheads are added to calculate total production cost (ore delivered to the processing facility), operating labour and maintenance and repair labour costs become 24.5% and 9.7% of such a total, respectively.

Tramming ore and waste, underground upkeep and support, and production development and delineation drilling are the three items with the largest operating labour components (64.0%, 63.3%, and 59.6%, respectively). A reduction of the relative importance of operating labour cost in these cases (which together account for 25.6% of total mining cost) would involve the modification of current operating procedures in short-hole horizontal drilling, support systems installation, and horizontal ore/waste transport. The operating labor components of mucking and production drilling are 38.1% and 34.4%, respectively. Although such items represent only 15.5% of mining cost, they seem to receive a lot of attention from recent mine automation projects (Poole et al., 1996; Baiden, 1996).

Direct operating labour cost tends to increase as the depth of mining increases. This is a consequence of longer personnel transport times, difficult working environments (heat, ground conditions, etc.), and a reduction in the efficiency and productivity of manned equipment in deep operations. However, the relative significance of such an increase is difficult to assess, since it is sensitive to modifications to the mining method and the type of equipment employed. In the next section, the impact of relatively minor modifications to the mine design parameters, mining sequence, and equipment configuration will be investigated. It is believed that there is still some room for improvement within the existing technological framework. Notwithstanding, if no changes are made at all, it is very probable that labour cost not only will increase in absolute terms[81] but also will represent a larger percentage of the total production cost.

If radically different mining equipment is introduced or a new mining method is devised, the direct operating labour cost may in fact decrease. However, the reliance on more sophisticated, possibly unproved technologies and processes could result in escalating repair and maintenance labour costs, and may also lead to increased overheads and a higher overall production cost.

5.2.3.4 Cost Data for Decision-Making

There is an extraordinary amount of detail in the internal reports provided by the operator. However, they are mostly concerned with accounting procedures, and disregard the kind of information required by operations management and decision-making. For instance, mucking cost data are shown in Table 8 and Table 9. Table 8 summarizes overall mucking costs: the information is gathered according to mining method and is further broken down into its operating and repair cost components.

[81] Increasing the production cost in dollars per pound of metal produced.

Table 7. Lynx Brook Mine - Operating and repair cost for 1994 (percentages)

Item	Operating Costs (%)				Repair Costs (%)				Total
	Op. Labour	Man. Labour	Mat./Other	Total	Labour	Mat./Sup.	O/H-Other	Total	(%)
Mine development & delineation drilling	59.56	0.33	36.75	**96.64**	0.97	1.48	0.91	**3.36**	**100.00**
In-the-hole (production) drilling	34.35	0.00	24.28	**58.63**	14.71	16.51	10.15	**41.37**	**100.00**
Other mining methods	24.34	0.00	75.66	**100.00**	0.00	0.00	0.00	**0.00**	**100.00**
Blasting and explosives	27.11	0.02	72.87	**100.00**	0.00	0.00	0.00	**0.00**	**100.00**
Ore & waste mucking/removal	38.12	0.14	7.74	**46.00**	23.58	22.96	7.46	**54.00**	**100.00**
Backfill	28.05	1.05	61.87	**90.97**	5.26	1.82	1.95	**9.03**	**100.00**
Tramming ore and waste	64.06	0.29	5.63	**69.98**	14.55	9.68	5.79	**30.02**	**100.00**
Crushing, conveying, hoisting & surface handling	45.54	2.52	7.18	**55.23**	22.97	10.45	11.35	**44.77**	**100.00**
Underground upkeep and support	63.25	1.29	25.53	**90.07**	5.40	2.86	1.67	**9.93**	**100.00**
Drainage and pumping	16.16	2.86	48.90	**67.91**	13.03	12.36	6.69	**32.09**	**100.00**
Ventilation and heating	11.35	4.05	48.26	**63.66**	18.03	11.66	6.65	**36.34**	**100.00**
Electric power	0.11	0.16	96.72	**96.99**	1.47	0.87	0.66	**3.01**	**100.00**
Water supply and compressed air	7.65	0.75	39.28	**47.67**	34.40	7.99	9.94	**52.33**	**100.00**
Surface services, safety and plant protection	29.01	2.55	60.94	**92.50**	2.62	2.72	2.15	**7.50**	**100.00**
Other equipment repairs	0.00	0.00	(18.87)	**(18.87)**	43.75	53.52	21.60	**118.87**	**100.00**
Mine management	0.01	0.00	99.99	**100.00**	0.00	0.00	0.00	**0.00**	**100.00**
TOTAL MINING COST	**32.23**	**0.95**	**43.52**	**76.69**	**10.70**	**8.00**	**4.61**	**23.31**	**100.00**
Overheads, depreciation, R&D, surface transport	5.31	4.15	88.44	**97.90**	0.73	1.26	0.11	**2.10**	**100.00**
TOTAL PRODUCTION COST	**24.52**	**1.86**	**56.38**	**82.76**	**7.84**	**6.07**	**3.32**	**17.24**	**100.00**

Table 9 displays the details regarding mucking costs for two production levels. Several observations are pertinent. First, repair accounts are not employed: maintenance and minor repairs are charged to operating accounts (major repairs and scheduled maintenance jobs are charged to repair accounts). Second, data are aggregated on a level-by-level basis, making it impossible to identify the individual pieces of equipment (in this case, scooptrams) that incurred such costs. Third, it is not possible to define the specific working area (i.e., stope or heading) that originated the costs. Finally, even though total labour cost of mucking in 7000 Level was $326,684 (63% of the 7200 Level cost), the corresponding supplies cost was only $2,430, or about 5% of the 7200 Level supplies cost figure. Most probably, this highlights a cost-allocating oversight, difficult to evaluate unless one is fully familiar with the operation.

The lack of pertinent details on production cost data makes it impossible to produce the following operating parameters, deemed critical to efficient mining operations management:

a. production cost (expressed in dollars per ton of material produced/transported) for individual pieces of equipment; and,
b. production cost (expressed in dollars per ton of material extracted) for specific areas of each level or even particular stopes.

In order to be able to obtain these and other equally important cost figures and ratios, data would have to be collected at the source in a different manner. This would not represent an extreme proposition, since repair cost information is already being gathered with a good amount of detail. However, even in the case of repair accounts, expenses are charged against general accounts, without specifying the actual working areas in which they were incurred (panels, stopes, or development headings). There would be a definite increase in administrative cost associated with more detailed reports (i.e., the time taken by operators, foremen, and shift bosses to fill out the forms or enter the data into the computer). Nonetheless, it is firmly believed that the wealth of information they would provide in turn would improve decision-making, compensating such a cost. The reports would have to be designed so that only relevant data are collected and the rest is properly aggregated.

Table 8. Lynx Brook Mine - Summary of mucking costs by mining method

Mining Method	Operating Cost ($)	Repair Cost ($)	Total ($)
Shrinkage stoping	577	---	577
Blasthole mining	5,081	---	5,081
Vertical retreat mining	2,322,443	---	2,322,443
Total	**2,328,101**	**---**	**2,328,101**

Table 9. Lynx Brook Mine - Operating mucking costs for 7000 and 7200 levels

Level	Labour ($)					Supplies ($)	Total ($)
	Base	Bonus	Overtime	Fringe	Total		
7000	158,019	48,289	4,155	116,221	326,684	2,430	329,114
7200	238,590	100,231	3,296	180,144	522,261	47,758	570,019
Total	**396,609**	**148,520**	**7,451**	**296,365**	**848,945**	**50,188**	**899,133**

Initially, the current system (devised mainly for accounting purposes) and the operations management-oriented one would have to coexist. A full transition could take a couple of years to complete. On the other hand, a *system* (most certainly, a *computer-assisted* one) would have to be put in place in order to make sure that all data are processed in an efficient and error-free manner, and are employed for the intended purposes. The mine-wide communications system already installed at Lynx Brook would facilitate the reporting phase. Software would have to be acquired or developed to manage the resultant (and fairly large) database and produce relevant reports for several levels of operations management, including summaries, charts, and figures.

5.2.3.5 Summary and Discussion

Proper manipulation and aggregation of the data as shown in Table 6 and Table 7 above have provided a more accurate and clear picture of the production cost structure at Lynx Brook. Indeed, Figure 22 and Figure 23 are two different ways of looking at and presenting its cost data. The former illustrates the fact that the four largest cost items account for almost one half of total mining cost, whereas the latter highlights the significance of operating labour cost (32.2% of total mining cost). It is obvious that proper data acquisition and aggregation are critical to the analysis of mining cost and the evaluation of the impact of changes to existing operating practices. This implies an adequate understanding of the existing operation and a clear definition of the objectives of the proposed modification. Since the reduction of production cost is always an important issue in mining projects,[82] the additional effort involved in gathering and processing the data is more than compensated for by the improved quality of the information thus obtained. Unless major operational changes are introduced,[83] a deeper operation at Lynx Brook would certainly see an increase of:

1. electric power requirements: equipment in general will not be as productive and ore/waste transportation routes will be longer;
2. production development cost: narrower orebodies will require more feet of development per ton of proven/probable ore reserves;
3. backfill cost: in addition to longer transportation distances (which increase the cost as well), higher stresses may demand stronger backfill; and,
4. hoisting cost: the productivity of the entire system is drastically reduced as it approaches the critical hoisting depth.[84]

On the positive side, level upkeep could be kept at a minimum if an extraction sequence is carefully developed so that only the required levels are maintained at any point in time. This would also have the advantage of lowering overall support requirements. In order to quantify the previous qualitative statements, some kind of simulation/sensitivity analysis would have to be carried out. Purely theoretical analyses, such as those employed in open pit mining (see Koniaris, 1991), are not applicable due to the complexity of the underground mine production process.

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[82] It must be remembered that underground hard-rock mines compete almost exclusively on cost (see Section 2.4).

[83] These include (but are not restricted to): a change in mining method; the adoption of new technologies for mine development and production; and innovative ore and waste transportation systems (such as vertical conveyors, slurry pumps, etc.). In order for them to have a significant impact on the total production cost, they would have to result in a reduced labour cost component, increased productivity of the equipment, lower dilution, and higher ore recovery. It is not clear, however, what the impact of such changes would be on the overhead costs.

[84] Already there is a serious restriction to the vertical transport of ore and waste from the deeper areas (not serviced by the existing hoist). Indeed, the muck must be hauled to an eagle crusher, crushed, and transported to the deepest loading pocket. This expensive and time-consuming process is not expected to change in the near future.

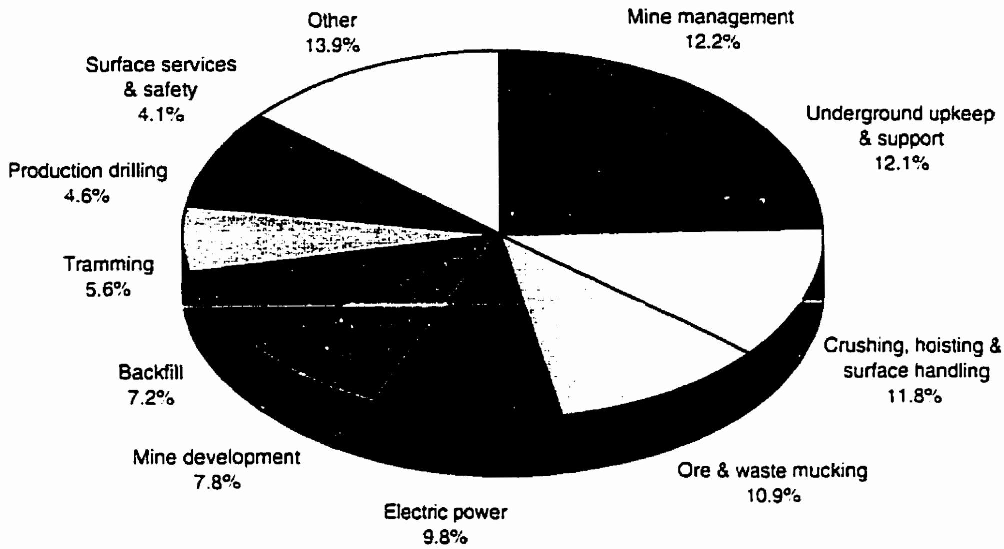

Figure 22: Lynx Brook Deep - Structure of 1994 mining cost.

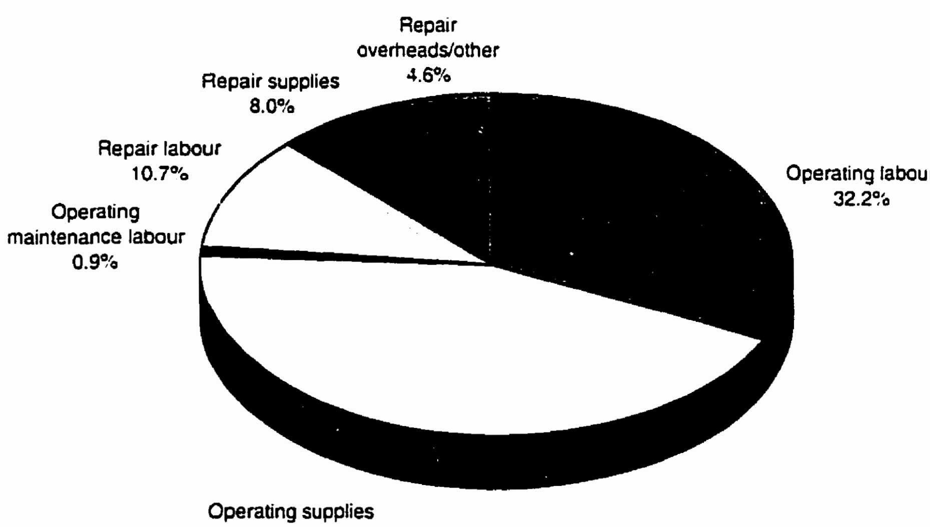

Figure 23: Lynx Brook Deep - Operating and repair components of 1994 mining cost

Labour cost has always been (and will continue to be) an important constituent of the cost structure of underground mining operations. As shown in Figure 23, the labour component at Lynx Brook was in 1994 43.9% of total mining cost. When overheads, depreciation, amortization, and surface ore transport are taken into account, total labour cost drops to 34.2% of production cost (see Table 7). It should be noted that mine management, the largest component of mining cost, **is not** considered a labour cost. Unless the adoption of innovative production technology is coupled with equally sophisticated (and effective) repair and maintenance systems and equipment, it will be difficult to reduce repair labour cost (in 1994, 10.7% of mining cost).

The overall cost structure of Lynx Brook is illustrated by Figure 24. Overheads plus mine management accounted for 37.3% of total production cost. These two large cost components should be carefully analyzed. If, as expected, new technologies introduced into underground hard-rock mining reduce the direct labour mining cost, will this result in increasingly higher overhead costs? Lynx Brook was developed as a traditional drill-blast-muck-hoist mining system. Thus, it might be better to keep and optimize the existing technologies and processes instead of adopting innovative systems that may demand extensive modifications to the ore-production process. The answers to these and other equally critical questions have to come from a well developed and implemented mining strategy, i.e., an integral approach must be adopted. However, it is possible to ascertain that the cost implications of the options considered, although difficult to evaluate, will have considerable impact on the final decision. Nonetheless, there is a good chance that, as the mine becomes deeper, the significance of overhead and mine management cost will increase (both in absolute and relative terms).

The cost structure shown in Figure 24 will definitely change as the mine deepens and modifications to the mining method as well as new equipment are introduced. The impact on the profitability of the operation will depend on the strategy adopted. Escalating maintenance, repair, and overhead costs would certainly negate any competitive advantages that could be gained through a reduced labour force. It would be better to target specific components of the cost structure with the highest labour cost component (such as underground upkeep, and tramming ore and waste) so that significant labour cost reductions are achieved more easily. In cases such as mucking, where repair costs are 54.0% of total cost (repair labour alone accounts for 23.6% of such a total) a special analysis should determine how the new system/equipment would reduce an existing troubling condition.

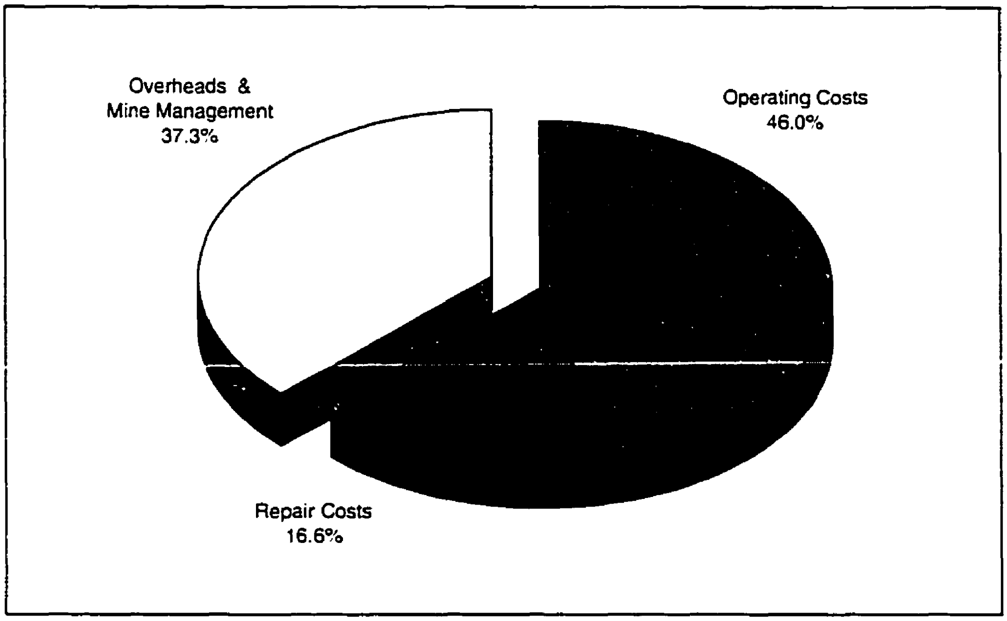

Figure 24: Lynx Brook Deep - Structure of 1994 total production cost

The main conclusions from this overview of the cost structure at Lynx Brook Mine are:

a. cost analysis must be comprehensive: it should focus on specific aspects or components without losing perspective and considering direct and indirect effects on other cost items;

b. as currently gathered, cost data cannot be used to produce adequate information for operations management and decision-making. A prime example is the case of trackless mucking equipment: it is not possible to calculate the cost of operating them in different parts of the mine (different levels, mucking development headings, stopes, etc.); and,

c. a working *model* of the processes and equipment/technologies involved must be developed with significant input from operations personnel. Such a model should facilitate the identification of areas that can be improved (i.e., with high cost-reduction potential) and the evaluation of proposed changes to the cost structure as indicated in the previous paragraph. In addition, it would allow a more accurate understanding of the complex interactions between the several cost components.

5.2.4 Future Development of Lynx Brook Deep - Main Issues

In the light of the distribution of existing mineral reserves and resources at the entire mining complex, the operator has appropriately determined that its future resides on Lynx Brook Deep.

In fact, at the time of the site visits, medium-term production commitments demanded that mining on 7400 Level commence within eighteen months. The area below 7000 Level would then be producing ore at a rate of 6,180 tonnes/week (883 tonnes/day, seven-day schedule), whereas in 1993 it only produced about 4,535 tonnes/week (907 tonnes/day, five-day schedule).

Several pressing issues must be taken into account in designing and planning the development of the deep reserves. They span the five factors identified in Section 1.4 and analyzed in Chapter 4. The following have been deemed to have the highest impact, from both operational and strategic viewpoints:

1. Vertical ore transport
   a. Shaft access
   b. Ramp access
2. Horizontal ore transport
   a. Equipment selection
   b. New technologies
3. Mine Development
   a. Production rate
   b. Dilution control
   c. Ore recovery
   d. Dimensions of mine development openings
   e. Horizontal mine development

It should be noted that ventilation and delivery of personnel and supplies also present challenging problems. The former cannot be adequately analyzed without explicit knowledge of the actual ventilation network, production schedule, and personnel and equipment requirements. Even if such data were available, the analysis would not only require extensive use of computer assisted tools, but also would depend on decisions regarding ore transport and mine development. The latter cannot be evaluated without detailed information about the ongoing operation. Its study also would demand familiarity with labour-relations and supplier policies. Thus, it was decided to focus on the factors listed above without neglecting to consider the effects on ventilation and delivery of personnel and supplies.

5.3 Vertical Ore Transport

In the light of the current depth of mining, existing ore reserves, and the need to meet short- and medium-term production commitments, the option of sinking a new shaft from surface is not a valid solution to this issue. Similarly, innovative technology such as vertical conveyors or hydraulic transport would not be economically feasible at this scale. Furthermore, they certainly would not complement well the existing hoisting and ore-handling facilities. Thus, Lynx Brook Deep is left with the following alternatives for the vertical transport of ore and waste from the deeper areas: deepen the existing shaft, sink an internal shaft, and drive a ramp. They will be reviewed in this section using the following common mining parameters:[85]

➢ Ore production rate:[86]	2,500 tonne/day (900,000 tonne/year)
➢ Waste production rate:	500 tonne/day (180,000 tonne/year)
➢ Elevation of lowest production level:	2,378 m

5.3.1 Shaft Deepening

The operational and economic feasibility of this option depend on:

- the capacity of the existing hoisting system;
- the ability of the operation to meet its production commitments while the shaft is being deepened; and,
- resulting capital and operating costs.

5.3.1.1 Capacity of the Existing Hoisting System

The first step is to evaluate the operating conditions of the existing hoisting system assuming that all production is hoisted from 7080 loading pocket. Input parameters, rope specifications, cycle calculations, safety factors, and power requirements (calculated using formulae provided by Edwards, 1992), are shown in Table 10. The second column of cycle times and distances uses current rope speed and acceleration rates (both of them higher than those usually allowed for steel guides). The static factor of safety is close to the minimum of 5.0 required by Ontario legislation.

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85 This section has benefited from ideas and information provided by Djan-Sampson (1996).

86 In 1994, Lynx Brook hoisted about 900,000 tonnes of ore and 90,000 tonnes of waste. The upper levels produced the majority of the ore (about 650,000 tonnes), which was loaded from the two upper loading pockets.

Table 10. Lynx Brook Mine – Operation of existing hoist – 7080 loading pocket

	Ore	Waste	Other	Total
Daily Tonnage (tonnes)	2,500	500	-	3,000

Hoisting depth	2,158	metres	7,080	feet
Headframe height (sheaves)	82.30	metres	270	feet
Dumping height (from surface)	40	metres	131	feet
Total hoisting distance	**2,198**	**metres**	**7,211**	**feet**
Shifts (skipping)	2	per day		
Shift duration	12	hours		
Availability for ore/waste skipping	80.00	%		
Time available for ore/waste skipping	19.20	hours/day		
Hoist utilization	74.00	%		
Actual skipping time	14.21	hours/day		
Hourly Tonnage	**211**	**tonnes/hour**	**233**	**tons/hour**
Rope Specifications (flattened strand, fibre cored 6 x 27 UHT)				
Diameter	5.72	cm	2.25	inches
Breaking strength	240,403	kg	530,000	lbs
Weight	12.72	kg/metre	8.55	lbs/ft
Drum diameter	**4.6**	**metres**	**15.00**	**feet**
Drum face width, single layer of rope	**9.6**	**metres**	**31.51**	**feet**
Drum face width, two layers of rope	**2.4**	**metres**	**7.88**	**feet**

Skipping Cycle - Maximum speed	**15.0 m/sec**		**16.8 m/sec**	
- Acceleration	**0.50 m/sec2**		**0.54 m/sec2**	
	time (sec)	distance (m)	time (sec)	distance (m)
Accelerate to 0.5 m/sec	1.0	0.3	1.0	0.3
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Accelerate to maximum speed	29.4	224.9	30.3	258.9
Run at full speed	116.2	1,742.7	99.9	1,674.7
Decelerate to creep speed	29.4	224.9	30.3	258.9
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Decelerate to stop	1.0	0.3	1.0	0.3
Load/Dump	15.0	-	15.0	-
Total	**202.0**	**2,198.0**	**187.6**	**2,198.0**

Skip load	**11.8**	**tonnes**	**11.0**	**tonnes**
Skip load	13.1	tons	12.1	tons
Safety Factor at loading pocket	**5.03**		**5.18**	
Motor Power Requirements				
Self-ventilated d-c motor rms Power	**4,160**	**kW**	**4,732**	**kW**
Induction (a-c) motor rms Power	**4,414**	**kW**	**5,061**	**kW**

Table 11 displays an estimate of the operating conditions of the existing hoist when all production is loaded from 7955 loading pocket. Apart from hoisting depth, the only other change made to the parameters shown in Table 10 is hoist utilization, which was increased to a reasonable 80%. It can be seen that, regardless of rope speed, the static factor of safety would drop below 5.0. If permission to operate the hoist under such conditions were obtained (there is already a case of such permission being granted in Ontario), the existing hoist, which is equipped with 11.8-tonne skips, bigger drums, and a 5,220-kW motor, would be capable of carrying out the job.

Table 11. Lynx Brook Mine – Hoisting system design: Shaft-deepening option

	Ore	Waste	Other	Total
Daily Tonnage (tonnes)	**2,500**	**500**	**-**	**3,000**
Hoisting depth	2,425	metres	7,955	feet
Headframe height (sheaves)	82.30	metres	270	feet
Dumping height (from surface)	40	metres	131	feet
Total hoisting distance	**2,465**	**metres**	**8,086**	**feet**
Shifts (skipping)	2	per day		
Shift duration	12	hours		
Availability for ore/waste skipping	80.00	%		
Time available for ore/waste skipping	19.20	hours/day		
Hoist utilization	80.00	%		
Actual skipping time	15.36	hours/day		
Hourly Tonnage	**195**	**tonnes/hour**	**215**	**tons/hour**
Rope Specifications (flattened strand, fibre cored 6 x 27 UHT)				
Diameter	5.72	cm	2.25	inches
Breaking strength	240,403	kg	530,000	lbs
Weight	12.72	kg/metre	8.55	lbs/ft
Drum diameter	**4.6**	**metres**	**15.00**	**feet**
Drum face width, single layer of rope	**10.7**	**metres**	**35.16**	**feet**
Drum face width, two layers of rope	**2.7**	**metres**	**8.83**	**feet**
Skipping Cycle - Maximum speed	**15.0**	**m/sec**	**16.8**	**m/sec**
- Acceleration	**0.50**	**m/sec2**	**0.54**	**m/sec2**
	time (sec)	distance (m)	time (sec)	distance (m)
Accelerate to 0.5 m/sec	1.0	0.3	1.0	0.3
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Accelerate to maximum speed	29.4	224.9	30.3	258.9
Run at full speed	134.0	2,009.4	115.8	1,941.4
Decelerate to creep speed	29.4	224.9	30.3	258.9
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Decelerate to stop	1.0	0.3	1.0	0.3
Load/Dump	15.0	-	15.0	-
Total	**219.8**	**2,464.7**	**203.5**	**2,464.7**
Skip load	**11.9**	**tonnes**	**11.0**	**tonnes**
Skip load	13.1	tons	12.2	tons
Safety Factor at loading pocket	**4.69**		**4.82**	
Motor Power Requirements				
Self-ventilated d-c motor rms Power	**4,404**	**kW**	**5,011**	**kW**
Induction (a-c) motor rms Power	**4,646**	**kW**	**5,323**	**kW**

5.3.1.2 Impact on Current Operation

As noted by Makuch (1994, p. 66) maintaining full mine production while deepening a shaft may be difficult. A major concern would be blasting in the shaft. Indeed, all activity in the shaft would have to stop while blasting gases are ventilated. All the waste produced by the deepening project (shaft sinking, shaft stations, main levels, etc.) would have to be hoisted to surface, since it could not be disposed of underground. Hoisting of personnel and materials to and from the bottom of the existing shaft would add also to the already busy schedule of the service hoist.

Assuming a shaft-sinking rate of 1.25-metre/day[57] and 100 tonnes/day of rock broken at other excavations, about 250 tonnes/day of additional waste would be produced. Such a tonnage would demand a higher hoisting rate from 7080 loading pocket (229 tonnes/hour) and higher power requirements (that could still be met by the existing motor). It also would lower the factor of safety to about 5.02, assuming that current skips, rope speed, and acceleration rate are used.

Once the shaft has been deepened, the loading pockets installed, and ore/waste passes completed, the excavation of ventilation raises and main levels can commence. If one 2.4-m round per day were achieved in each of two 4.4-m x 4.1-m headings, about 250 tonnes/day of additional waste would be generated. Such a tonnage would have to be added to the 3,000 tonnes/day already being hoisted from the upper loading pockets. As shown in Table 11, the existing system would be capable of hoisting from a single loading pocket all of the ore and waste produced by Lynx Brook Deep. Therefore, the additional 250 tonnes/day of waste could be hoisted from the deep loading pocket by slightly increasing the skip size (and further lowering the safety factor).

In summary, shaft deepening would affect seriously the ongoing operation. Preliminary estimates indicate that existing hoisting facilities would be able to meet current production commitments. However, the final static factor of safety at the deep loading pocket would be 4.69, about 6.9% lower than the minimum of 5.00 required by Ontario legislation.

5.3.1.3 Capital and Operating Costs

- ***Capital Costs***

  Table 12 presents estimates of the time and cost involved in excavating each of the major openings.[58] The table does not show the time to install the shaft-sinking arrangement (30 days), and the time needed to install the new wire rope and bring the production hoist into operation (30 days). Total time for the completion of the shaft-deepening project is, thus, 864 days (about 29 months).

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[57] Makuch (1994) reviewed past shaft deepening projects and found that the average advance rate was 1.0 metre/day.

[58] In this section, the purpose of estimating capital and operating costs is to compare the alternatives considered, not to produce accurate figures. Thus, the relationships developed by O'Hara (O'Hara and Suboleski, 1992, pp. 405-424), together with capital and operating cost estimates compiled by Western Mine Engineering (1996) and information supplied by the operator, were used to provide preliminary cost figures.

Table 12. Lynx Brook Mine – Excavation costs: Shaft-deepening option

Excavation	Dimensions (m)			Length	Tonnage	Excavation	Cost	
	Width	Height	Diameter	metres	tonnes	Time (days)	US $/m	US $
Shaft(*)	—	—	6.4	257	24,804	206	N.A.	5,196,877
Orepasses (two)	—	—	2.1	582	6,049	146	1,131	658,657
Ventilation raises (two)	—	—	2.4	582	7,901	146	1,277	743,645
Main levels (four)	4.4	4.1	—	1,000	54,181	226	1,753	1,752,810
Loading pocket/bins	—	—	—	—	3,500	30	N.A.	218,932
Shaft stations (four)	6.0	5.0	—	100	9,000	50	1,460	145,985
Total					105,435	804		8,716,907

(*) Shaft-sinking cost estimated using O'Hara's formulae (O'Hara and Suboleski, 1992)

The only additional item is 2¼ inch, 6x27 flattened strand, UHT wire rope, whose cost is calculated: 16,250 ft * U.S. $25/ft * 1.0748014 = U.S. $436,640. Total capital cost would be:

- Total capital cost = U.S. $8,716,910 + U.S. $436,640 = U.S. $9,153,550.

- *Operating Costs*

The hourly operating cost of a 5,220-kW (7,000-hp) double-drum hoist equipped with 4.6-m (15-ft.) diameter drums can be estimated as follows (Western Mine Engineering, 1996):

- Parts: U.S. $58.63/hour
- Maintenance labour: U.S. $42.23/hour
- Electric power: U.S. $292.95/hour
- Lubricants: U.S. $16.62/hour
- Operating labour U.S. $34.52/hour
- **Total U.S. $444.95/hour**

5.3.2 Internal Shaft

An internal shaft could be sunk so that ore and waste from the deeper levels would be hoisted and transferred to the deepest loading pocket for transportation to surface through the current production shaft. The exact location of the internal shaft is not relevant to this study. However, in order to minimize initial investment (i.e., shaft sinking) and reduce total transportation time, it would be sunk 75 metres away from the existing shaft and collared at the 6900 Level elevation. A 60-m long conveyor belt on 7000 Level would transport muck to an ore pass connecting to 7080 loading pocket. Given the production rate and mine life, a service hoist would be needed as well.

It has already been concluded that, provided permission is obtained to operate with a safety factor lower than 5.0, the current facilities can hoist the entire ore and waste production from the 7080 loading pocket (see Section 5.3.1). Thus, the operational and economic viability of this option would be determined by the ability to continue mining at current rates while the internal shaft is being sunk, and the capital and operating costs related to the operation of the new hoist.

5.3.2.1 Impact on Current Operation

Sinking the internal shaft could greatly affect the production capacity of the ongoing operation. While there are several areas that could be influenced, the most critical would be the ability to continue hoisting the required daily tonnage of ore and waste. In order to estimate the amount of waste generated by shaft-sinking, preliminary designs of the production and service hoists[89] were carried out and the results summarized in Table 13 and Table 14. Due to the shallow hoisting depth (322 m) and relatively low hoisting requirements (3,000 tonnes/day), the factor of safety for the production hoist would be about 5.8 or 5.3, depending on skip load and rope speed. Preliminary estimates of the diameter of the shaft, area of the corresponding hoist room, and headframe height (in feet) were obtained using O'Hara's formulae (O'Hara and Suboleski, 1992):

$$D = 5.5 * T^{0.15} = 5.5 * (3.307)^{0.15} = 18.5$$

$$A = 0.085 * (D_1^{2.2} + D_2^{2.2}) = 0.085 * (100^{2.2} + 100^{2.2}) = 4{,}270$$

$$H = 8.0 * T^{0.3} + 1.2 * S^{0.5} = 8.0 * 3.307^{0.3} + 1.2 * 700^{0.5} = 91.0 + 31.7 = 122.7$$

where:

D :	shaft diameter (in feet)
T :	hoisting rate (in tons per day)
A :	area of hoist room (in ft^2)
D_1 :	drum diameter of production hoist (in inches)
D_2 :	drum diameter of service hoist (in inches)
H :	headframe height (sheaves, in feet)
S :	rope speed (in feet per minute)

[89] The analysis of the alternatives for vertical ore and waste transport does not consider the capital and operating costs associated with the service hoist. Its specifications were determined so that the hoist room and headframe for the internal shaft option could be sized (and costed) properly.

Table 13. Lynx Brook Mine – Production hoisting system design: Internal shaft

	Ore	Waste	Other	Total
Daily Tonnage (tonnes)	2,500	500	-	3,000
Hoisting depth	322	metres	1,055	feet
Headframe height (sheaves)	37	metres	120	feet
Dumping height (from collar)	15	metres	49	feet
Total hoisting distance	**337**	**metres**	**1,104**	**feet**
Shifts (skipping)	2	per day		
Shift duration	12	hours		
Availability for ore/waste skipping	80.00	%		
Time available for ore/waste skipping	19.20	hours/day		
Hoist utilization	80.00	%		
Actual skipping time	15.36	hours/day		
Hourly Tonnage	**195**	**tonnes/hour**	**215**	**tons/hour**
Rope Specifications (flattened strand, fibre cored 6 x 27)				
Diameter	3.18	cm	1.25	inches
Breaking strength	73,935	kg	163,000	lbs
Weight	4.08	kg/metre	2.74	lbs/ft
Drum diameter	**2.5**	**metres**	**8.33**	**feet**
Drum face width, single layer of rope	**1.6**	**metres**	**5.38**	**feet**
Drum face width, two layers of rope	**0.7**	**metres**	**2.17**	**feet**
Skipping Cycle - Maximum speed	**3.5**	**m/sec**	**3.0**	**m/sec**
- Acceleration	**0.50**	**m/sec2**	**0.50**	**m/sec2**
	time (sec)	distance (m)	time (sec)	distance (m)
Accelerate to 0.5 m/sec	1.0	0.3	1.0	0.3
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Accelerate to maximum speed	6.4	12.2	5.4	8.9
Run at full speed	87.6	306.7	104.4	313.2
Decelerate to creep speed	6.4	12.2	5.4	8.9
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Decelerate to stop	1.0	0.3	1.0	0.3
Load/Dump	15.0	-	15.0	-
Total	127.4	336.6	142.2	336.6
Skip load	**6.9**	**tonnes**	**7.7**	**tonnes**
Skip load	7.6	tons	8.5	tons
Safety Factor at loading pocket	**5.82**		**5.28**	
Motor Power Requirements				
Self-ventilated d-c motor rms Power	**294**	**kW**	**275**	**kW**
Induction (a-c) motor rms Power	**305**	**kW**	**284**	**kW**

Assuming a 5.5-m internal shaft, an average shaft-sinking rate of 2.0 m/day,[90] and a maximum of 150 tonnes of additional waste, the production hoist would have to transport about 3.300 tonnes of muck per day during the excavation phase. If existing hoist utilization, skips, and rope speed were kept, the safety factor would drop to 4.99 but the current motor would be able to meet the new power requirements, even if the entire production is hoisted from the 7080 loading pocket.

[90] This is a somewhat optimistic rate. Halls (1982) reported that the average construction rate for 305-metre deep. 6.1-m diameter shafts was 43.7 weeks (306 days). This is equivalent to a 1.0-m/day shaft-sinking rate.

Table 14. Lynx Brook Mine – Service hoisting system design: Internal shaft

Maximum hoisting depth	322	metres	1,055	feet
Headframe height (sheaves)	37	metres	120	feet
Total hoisting distance	**322**	**metres**	**1,055**	**feet**
Rope Specifications (flattened strand, fibre cored 6 x 27)				
Diameter	3.18	cm	1.25	inches
Breaking strength	73,935	kg	163,000	lbs
Weight	4.08	kg/metre	2.74	lbs/ft
Drum diameter	**2.5**	**metres**	**8.33**	**feet**
Drum face width, single layer of rope	**1.6**	**metres**	**5.17**	**feet**
Drum face width, two layers of rope	**0.6**	**metres**	**2.07**	**feet**
Cycle - Maximum speed	**2.5**	**m/sec**	**2.5**	**m/sec**
- Acceleration	**0.50**	**m/sec2**	**0.50**	**m/sec2**
	time (sec)	distance (m)	time (sec)	distance (m)
Accelerate to 0.5 m/sec	1.0	0.3	1.0	0.3
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Accelerate to maximum speed	4.0	6.2	4.0	6.2
Run at full speed	121.5	303.7	121.5	303.7
Decelerate to creep speed	4.0	6.2	4.0	6.2
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Decelerate to stop	1.0	0.3	1.0	0.3
Load/Unload	15.0	-	15.0	-
Total	**156.5**	**321.6**	**156.5**	**321.6**
Cage load - personnel (50 miners)	**5.7**	**tonnes**	**5.7**	**tonnes**
Cage load - personnel (50 miners)	6.3	tons	6.3	tons
Maximum cage load (materials)	**8.0**	**tonnes**	**9.0**	**tonnes**
Maximum cage load (materials)	8.8	tons	9.9	tons
Safety Factor at lowest level - materials	**5.55**		**4.99**	
Motor Power Requirements				
Self-ventilated d-c motor rms Power	**235**	**kW**	**263**	**kW**
Induction (a-c) motor rms Power	**241**	**kW**	**269**	**kW**

5.3.2.2 Capital and Operating Costs

- *Capital Costs*

Table 15 shows estimates of the time and cost involved in excavating each of the major openings. Hoist-related costs (in 1988 U.S. dollars) were estimated using O'Hara's formulae (O'Hara and Suboleski, 1992):

$$\text{Production hoist} = \$700 * (0.9 * D)^{1.4} * Hp^{0.2} = \$700 * 90^{1.4} * 402^{0.2} = \$1{,}264{,}400$$

$$\text{Hoist installation} = \$64 * D^{1.8} = \$64 * 100^{1.8} = \$254{,}800$$

$$\text{Headframe} = \$19 * (1.2 * 0.12 * H^{3} * (D/100)^{2})^{0.9} = \$19 * (0.144 * 120^{3} * (100/100)^{2})^{0.9} = \$1{,}364{,}800$$

$$\text{Ore bins, skips, etc.} = \$700 * T^{0.7} = \$700 * 3{,}307^{0.7} = \$203{,}600$$

$$\text{Total hoist - related cost} = \$1{,}264{,}400 + \$254{,}800 + \$1{,}364{,}800 + \$203{,}600 = \$3{,}087{,}600$$

Table 15. Lynx Brook Mine – Excavation costs: Internal shaft

Excavation	Dimensions (m)			Length	Tonnage	Excavation	Cost	
	Width	Height	Diameter	metres	tonnes	Time (days)	US $/m	US $
Shaft(*)	---	---	5.5	329	23,347	165	N.A.	5,112,632
Headframe (37 m)	6.0	6.0	---	37	3,950	24	1,460	53,396
Hoist room (400 m2)	20.0	20.0	---	7	8,400	28	16,000	112,000
Orepasses (two)	---	---	2.1	582	6,049	146	1,131	658,657
Ventilation raise	---	---	2.4	291	3,950	73	1,277	371,823
Main levels (four)	4.4	4.1	---	1,000	54,181	226	1,753	1,752,810
Loading pocket/bins	---	---	---	---	3,500	30	N.A.	218,932
Shaft stations (four)	6.0	5.0	---	100	9,000	50	1,460	145,985
Total					112,378	742		8,426,235

(*) Shaft-sinking cost estimated using O'Hara's formulae (O'Hara and Suboleski, 1992)

Total hoist-related costs in 1996 U.S. dollars (the units used throughout this section) are $3,087,600 * 1.2383907 = $3,823,700. The installed cost of a 60-m long conveyor with a 0.60-m wide belt and capable of handling up to 450 tonne/hour is US $126,000 The cost of 1¼ inch, 6x27 flattened strand, FC wire rope is: 2,450 feet * $9.50/foot * 1.0748 = $25,000. Therefore, total capital cost for this option is:

- Total capital cost = $8,426,200 + $3,823,700 + $25,000 + $126,000 = $12,400,900

Table 15 does not include the time required to install and commission the new production and service hoists (150 days). Total time to complete this option is 892 days (about 30 months).

- ***Operating Costs***

Table 16 shows the hourly operating cost of the production hoist installed in the internal shaft, the existing production hoist, and the conveyor belt installed in 7000 Level.

Table 16. Lynx Brook Mine – Operating costs: Internal shaft

	Operating Cost (US $/hour)					
	Parts	Maintenance Labour	Electric Power	Lubricants	Operating Labour	Total
300-kW Production hoist	17.16	12.36	20.93	4.86	34.52	89.83
5,220-kW Production hoist	58.63	42.23	292.95	16.62	34.52	444.95
Conveyor belt (60-m long)	3.00	2.92	1.67	0.72	8.63	16.94
Total	78.79	57.51	315.55	22.20	77.67	551.72

Source: Western Mine Engineering (1996)

5.3.3 Truck Haulage

Access to the bottom of the mineralized area can be obtained by driving a 15% ramp in the footwall. Starting at the 7200-Level elevation (the lowest level currently developed), the ramp would connect all main levels down to the 7800 Level. Total length of the ramp would be 1,220 metres. Apart from the main levels (which must be excavated regardless of the type of access), the only other openings required by this option are re-muck stations (to facilitate driving the ramp) and a ventilation raise. Muck would be transported to the 7000-Level crusher via an existing ramp and then loaded into skips through the 7080 loading pocket. As in the previous case, the feasibility of this option depends on the degree of interference with the ongoing operation, the related excavation costs, and the cost of operating the vertical transport system.

5.3.3.1 Impact on Current Operation

The flexibility of truck haulage systems makes it more difficult to estimate accurately the impact of driving the 1,220-metre long ramp on current mining operations. In fact, development waste could be dumped in old workings (stopes, levels, ore-passes, etc.) already found below 7000 Level. This practice would simultaneously reduce any interference with the production hoist (which then could be devoted to hoisting from the upper levels) and the actual cost of driving the ramp (since part of the muck would not have to be hoisted to surface for disposal). Nonetheless, in order to make valid comparisons with other vertical transport alternatives, it will be assumed that all of the waste produced at the ramp and related excavations will be hoisted to surface.

As discussed in Section 4.4.2, ventilation requirements and mobile equipment dimensions determine the cross-section of development openings. In this case, equipment size is not the limiting factor. Indeed, 30-, 40-, and 50-tonne mine trucks have very similar dimensions: about 3.0 m wide, 2.7 m high, and 10.0 m long. On the other hand, Lynx Brook is a hot mine and already has serious ventilation problems. Thus, a 5.0 m x 4.9 m ramp, wider and higher than required by safety regulations, will be assumed. An average daily advance rate of 3.3 m will be used, reflecting the fact that the ramp would be driven in a deep environment, prone to ground control, ventilation, and logistical problems. This would result in about 250 tonnes of muck per day. If a maximum of 150 tonnes per day are generated by other excavations, the production hoist would have to transport 3,000 + (250 + 150) = 3,400 tonnes/day. In order to meet the new hoisting rate of 239 tonnes/hour (assuming 80% availability and 74% utilization, and maintaining rope speed and acceleration rate), the skip payload would have to be increased to 12.5-tonnes. The corresponding safety factor would fall to 4.93.

5.3.3.2 Capital and Operating Costs

- ***Capital Costs***

  Table 17 shows excavation times and costs for the major openings involved in providing ramp access to Lynx Brook Deep. It should be noted that orepasses are not required in this option, since LHDs would directly load trucks to minimize both trucking distance and development time.[91]

  Figure 25 was used to determine tramming capacity and fleet size for the trucks that would transport muck to 7000 Level during the production phase. It can be seen that three 40-tonne trucks would be able to haul the entire production (3,000 tonnes/day) from the first three levels. In fact, as long as production is still coming from the upper levels, two such trucks would be enough to initially haul muck from 7350 Level. A third truck would be needed once production shifts completely to 7350 Level, and an additional truck would have to become available to haul exclusively from 7800 Level. The cost of each truck is U.S. $500,000.

  The only other significant capital investment would be in ventilation and air conditioning. It is inevitable to employ diesel equipment for driving the ramp, but the use of diesel trucks for transporting 3,000 tonnes/day of muck to 7000 Level must be avoided. Unfortunately, it was impossible to obtain capital and operating cost data for trolley-assisted mine trucks. Therefore, diesel trucks will be assumed and an estimate of the *additional* ventilation cost associated with the operation of the trucks will be included.

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[91] An important advantage of the use of a ramp for accessing the deep reserves is that there is no need to wait until the bottom of the orebody is reached to commence production. Indeed, provided adequate ventilation is supplied, hoisting capacity exists, and an adequate mining sequence is devised, stopes could be mined as soon as their corresponding access openings are developed. This would have the additional benefit of facilitating a top-down mining sequence that, as discussed in Section 3.3, can help in reducing ground control problems.

Table 17. Lynx Brook Mine – Excavation costs: Truck haulage option

Excavation	Dimensions (m)			Length	Tonnage	Excavation	Cost	
	Width	Height	Diameter	metres	tonnes	Time (days)	US $/m	US $
Ramp	5.0	4.9	---	1,220	89,767	367	1,871	2,282,451
Ventilation raise	---	---	3.0	244	5,171	61	1,277	311,474
Main levels (four)	4.4	4.1	---	1,000	54,181	181	1,647	1,646,774
Re-muck stations (twenty	5.0	4.9	---	300	22,074	75	1,871	561,258
Total					**171,192**	**684**		**4,801,957**

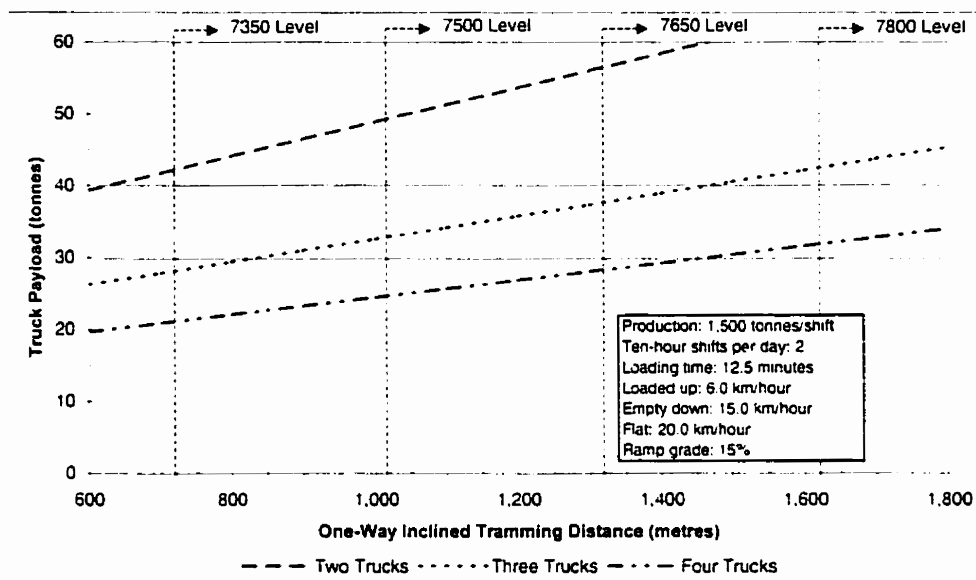

Figure 25: Lynx Brook Deep – Truck tramming capacity and fleet size

The air requirements of four 335-kW (450-bhp) 40-tonne trucks is estimated using the rule of thumb indicated by Hartman et al. (1997, p. 524): 0.0949 $m^3/s \cdot kW$ (150 cfm/bhp)

- Total air = 150 cfm/bhp * 450 bhp * (1.0 + 0.75 + 0.5 + 0.5) = 185,625 cfm

The cost of a 200,000-cfm axial fan complete with accessories is U.S. $143,000 and that of the corresponding 300-kW electric motor is U.S. $37,000. To estimate the additional air-conditioning requirements it is necessary first to calculate the heat liberated by the diesel engines of the four trucks (Hartman et al., 1997, p. 601):

- Heat load = 12.3 gal/h * 140,000 Btu/gal *0.9 *2.75 = 4,261,950 Btu/h = 1,250.0 kW

Table 18. Lynx Brook Mine – Total capital cost: Truck haulage option

Item	Cost U.S. $
Excavation	4,801,957
40-tonne mine trucks (4)	2,000,000
200,000-cfm axial fan	143,000
300-kW motor (fan)	37,000
Air conditioning plant	1,325,000
Total	8,306,957

Whillier (1982) noted that the total cost of air conditioning in a mine is about US $200/kW of cooling per year, of which 33% can be attributed to the capital investment. If it were assumed that the equipment is depreciated over a 10-year period, and converting the amount to 1996 U.S. dollars, it would result in capital investment requirements of $1,060/kW. In this case, total investment in air-conditioning would be 1,250 kW * $1,060/kW = $1,325,000. Total capital cost for this option is summarized in Table 18 (see above).

- ***Operating Costs***

  Table 19 shows the hourly operating cost of the mine trucks, fan, and air conditioning plant required by the truck haulage option.

5.3.4 Evaluation of Alternatives

The three options for vertical ore and waste transport presented in this section can be evaluated in terms of their economic impact on the operation and strategic significance.

5.3.4.1 Economic Analysis

A summary of the capital and operating costs involved in each of the three vertical transport alternatives can be found in Table 20. Based on the known reserves (see Table 1) and a yearly production rate of 900,000 tonnes, a mine life of about 6 years was determined (dilution was not accounted for). Total construction and commissioning times were 29 months for shaft deepening, 30 months for internal shaft, and 23 months for truck haulage. Operating costs were determined using the net operating hours estimated for each piece of equipment (shaft, conveyor, truck, etc.).

Table 19. Lynx Brook Mine – Operating costs: Truck haulage

	Operating Cost (US $/hour)							
	Parts	Maint. Labour	Diesel Fuel	Electric Power	Lubric.	Tires	Operating Labour	Total
40-tonne truck	10.06	7.24	9.83	N.A.	3.35	5.44	34.52	70.44
5,220-kW hoist	58.63	42.23	N.A.	292.95	16.62	N.A.	34.52	444.95
200,000-cfm axial fan	3.38	4.52	N.A.	N.A.	0.64	N.A.	N.A.	8.54
300-kW motor (fan)	1.03	1.36	N.A.	20.93	N.A.	N.A.	4.32	27.64
Air conditioning plant	15.33	4.60	N.A.	82.50	N.A.	N.A.	34.52	136.95
Total	**88.43**	**59.95**	**9.83**	**396.38**	**20.61**	**5.44**	**107.88**	**688.52**

Sources: Western Mine Engineering (1996); Whillier (1982)

Table 20. Lynx Brook Mine – Vertical ore transport: Economic evaluation

Option	Type of Cost	Year									Total	NPV @ 10%	
		C1	C2	C3	O1	O2	O3	O4	O5	O6		Cap/Oper	Total
Shaft Deepening	Capital	3,784.1	2,815.0	2,554.4	-	-	-	-	-	-	9,153.6	7,685.7	15,736.6
	Operating				2,460.4	2,460.4	2,460.4	2,460.4	2,460.4	2,460.4	14,762.4	8,050.8	
Internal Shaft	Capital	3,989.1	5,849.4	2,562.5	-	-	-	-	-	-	12,400.9	10,385.8	19,742.8
	Operating				2,859.6	2,859.6	2,859.6	2,859.6	2,859.6	2,859.6	17,157.4	9,357.0	
Truck Haulage	Capital	-	2,092.2	5,214.7	500.0	-	-	500.0	-	-	8,307.0	6,245.1	22,654.9
	Operating				4,583.5	4,989.2	4,989.2	4,989.2	5,395.0	5,395.0	30,341.1	16,409.8	

Note: All figures in thousands of 1996 U.S. dollars

Shaft deepening resulted in the lowest total operating cost ($14.8 million, the cost of operating the existing production hoist) and the second lowest capital investment ($9.2 million). Although it requires 29 months to become operational, the NPV of the total cost of this option is $15.7 million, the lowest of the three vertical transport alternatives considered in this section.

The construction and commissioning of an internal shaft would require the highest investment ($12.1 million) over 30 months (the longest pre-production period). Its operating cost is higher than that of the shaft-deepening option, since it requires the additional operation of an internal winze and a conveyor belt. The NPV of the total cost of using an internal shaft for vertical ore transport is $19.5 million.

Truck haulage requires the lowest capital investment ($8.3 million) and the shortest pre-production period (two years). However, due to the 24-hour/day operation of the air conditioner and 200,000-cfm fan, as well as the need to use the current production hoist to bring the muck to surface, it has the highest operating cost ($5.4 million/year, with four 40-tonne trucks in operation). The NPV of the total cost of the truck haulage option is $22.7 million.

5.3.4.2 Discussion

In spite of its lowest total cost, deepening the current shaft may not be the optimum solution to the vertical ore transport problem. The following issues must be further considered:

- Without knowing current operating procedures, it was difficult to quantify adequately the impact of shaft deepening on the production capacity of the entire mine. If, for instance, hoist availability for skipping were reduced to 70%, actual skipping time would fall to 12.4 hours. In order to sustain current production rates during shaft deepening, a 13.6-tonne skip and a larger hoist motor would have to be installed, and the factor of safety would drop to 4.75. If production targets were not met, the loss of income could significantly increase the total cost of this option.
- The resulting static factor of safety at the loading pocket is lower than 5.0, even when assuming a hoist utilization of 80% (that results in 15.4 hours/day of net hoisting time). It is not clear if permission to operate under such conditions would be obtained.
- The production capacity is virtually fixed, with very little chance of increasing it in the future. Furthermore, if deeper ore reserves were found, an alternative vertical transport

system would have to be found, since further extending the shaft would not be a feasible option at that time.

The construction of an internal shaft takes longer to complete, but does not directly interfere with the ongoing production process or require changes in the operating parameters of the existing production hoist. Although the amount of waste generated would be slightly higher than in the case of shaft deepening, total tonnage would be still within the range of the current hoisting facilities (provided permission to operate a hoist with a safety factor of less than 5.0 is obtained).

A major disadvantage of both the shaft deepening and internal shaft alternatives is that the only route for access to the deeper levels is through the shafts, which become effectively the bottlenecks of their respective transport systems. As discussed in Section 4.5, the overall efficiency of a deep operation can suffer significantly if the delivery of personnel and materials is not carried out as scheduled. Therefore, both options would require a service ramp to complement maintenance and materials transport functions, as well as to increase the flexibility of the operation. However, the cost of such a ramp (around $2.3 million) would reduce the economic attractiveness of the shaft-related alternatives and increase the amount of pre-production waste, further affecting the ability to meet production requirements during construction.

As presented here, truck haulage is not a viable alternative for vertical transport of muck. However, by assuming the use of diesel units, this alternative was unfairly *penalized* in relation to the other two transport options. In fact, if electric (i.e., trolley-powered) units had been considered, ventilation and air conditioning requirements as well as energy costs would have been much lower. In spite of the need to install the trolley lines, this would have reduced drastically both initial investment and direct operating costs.

Truck haulage does have important advantages. As pointed out in Footnote 91, the operation does not have to wait until the ramp is complete to initiate trucking from the deeper levels. In theory, mining could start at the beginning of the second (and last) construction year. By allowing the early generation of positive cash flows, this could significantly improve the NPV calculations of the truck haulage option. However, the ventilation and air conditioning systems would have to be also in operation at the end of the first construction year (*without* the ventilation raise, which can be excavated only after reaching 7800 Level), eliminating some of the financial benefits of early production. On the other hand, and provided the performance of electric trucks is similar to that of their diesel counterparts, four 40-tonne trucks could support production from two more 45-m levels (down to 8100 level). Likewise, production and/or development rates can be increased from the upper levels if deemed necessary. In theory additional trucks can be added to achieve such targets, but larger fleets create traffic problems in the ramp, reduce the overall efficiency of the transport system and, thus, can be counterproductive.

Last but not least, a ramp greatly enhances the ability to deliver personnel and materials to actual working places, facilitates supervision and coordination of the work, and can be located so that allows additional exploration and mine development. Unfortunately, these operational aspects are not easy to quantify and demand thorough knowledge of the operation to be evaluated properly.

5.3.5 Conclusion

The final decision on the vertical transport issue has to consider both hard (i.e., capital and operating costs) and soft (i.e., flexibility and efficiency) issues. Although the economic evaluation clearly shows that shaft deepening would result in lower overall costs, serious issues regarding this solution cannot be addressed adequately at this stage. Furthermore, since the government would have to authorize the operation of the production hoist with a safety factor of less than 5.0, this alternative is not feasible at this time. The internal shaft is expensive and somewhat inflexible, but is not constrained by the limitations imposed on the existing system. Although expensive and time-consuming, proper design and planning could allow both increased production rates and further expansion into deeper ore reserves. The use of electric trucks would have made ramp haulage a realistic vertical transport option. The flexibility, direct level access, and expandability associated with this alternative certainly enhance its attractiveness.

This preliminary assessment of the vertical transport options for Lynx Brook Deep indicates that shaft deepening is not feasible. Similarly, truck haulage must be re-evaluated assuming the use of electric units. The final selection would have to consider only an internal shaft and trucking.

5.4 Horizontal Ore Transport

The presence of an ongoing mining operation at Lynx Brook limits the number of options available for horizontal transport in the deeper areas. The adoption of significantly different technologies would certainly conflict with the current LHD-based systems, not to mention the increased training, maintenance, and warehousing costs. Furthermore, the tabular nature of the deep reserves coupled with more strict ground control practices would prevent the use of continuous loading and conveying systems, which typically require high production rates to become economically sound. Thus, this section will address equipment selection issues for LHD-based systems as well as discuss briefly the possibility of employing innovative methods for horizontal transport at Lynx Brook Deep.

5.4.1 Equipment Selection - LHDs

In underground mines, equipment selection heavily depends on the choice of mining method, planned production rate, and productivity of single mining units (stopes or panels). At Lynx Brook, proper sequencing has managed to provide effective ground control, although dilution has increased and the ability to recover all of the ore resource has been affected. It is reasonable to assume that VCR stopes will be the main source of ore in the deeper areas. As noted in Section 5.2.3, Lynx Brook Deep is expected to produce about 6,200 tonnes of ore per week. However, limitations imposed by the existing production capacity of the shaft and hoist, and the mining sequence itself, demand a more conservative estimate.

At least down to the 7600 Level, the current productivity of stopes and panels will be maintained.[92] If 10.5-m x 10.5-m panels were used and each blast was 4.8 m high, about 2,223 tonnes of ore would be produced. This would be enough to satisfy more than one half of the required daily production rate of about 900 tonnes for four days. To account for delays and unexpected operational problems, this panel configuration would result in a minimum of three panels in production (i.e., mucking stage of the production cycle) at any given time.

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[92] The orebody becomes very thin at the 7800 Level elevation and, as previously indicated, could be considered as a vein-type deposit of irregular strike for all practical purposes, including mining sequencing and ground control.

To illustrate the range of choices available for equipment selection, an attempt will be made to determine the size and number of units needed to muck and haul ore from the panels to the dump located at 6970 Level. It will be assumed that standard LHDs and low-profile trucks will be employed. Additionally, the following assumptions have been made:

- number of shifts per day: 2
- shift length: 8 hours
- shift delays (transporting personnel, materials, etc.): 1.0 hour/shift
- mid-shift break: 0.5 hour/shift
- effective minutes per hour: 50
- production rate of Lynx Brook Deep: 908 tonnes/day

Based on these assumptions and using the calculation procedures suggested by Wagner Mining Equipment Co. (1987), it is possible to plot the one-way horizontal tramming distance versus the minimum rated capacity of the scooptram(s) employed. The relationship between LHD capacity and tramming distance is expressed by the following formula:

$$L = R * \left[f_{time} + \frac{2D}{16.67S} \right]$$

where:

L: rated capacity of the LHD unit, in cubic metres

R: production rate, in cubic metres per minute

f_{time}: fixed time to load/dump/maneuver each cycle, in minutes

D: one-way horizontal tramming distance, in metres

S: average speed over distance D, in kilometres per hour

Results obtained are shown in Figure 26 and Figure 27. The former presents the case of a single LHD unit mucking all the Lynx Brook Deep ore production. The latter assumes that two identical LHDs share the production of 454 tonnes/shift (908 tonnes/day) of ore. Two lines representing different average speeds (6.0 km/hour and 8.0 km/hour) were plotted in each figure to investigate the impact of varying working conditions on productivity. It can be seen that as tramming distance increases such as impact is more severe. Let us apply these concepts to the specific case of Lynx Brook Deep. Figure 28 and Figure 29 show plan layouts for levels 7350 and 7500, respectively.[93] Based on such layouts, it was found that the average horizontal one-way tramming distance for LHD units from muckpiles to a dumping area close to the main access ramp is about 427 metres. Minimum and maximum tramming distances vary between 300 and 360 m, and 480 and 580 m, respectively. As depicted by Figure 26, the minimum rated capacity increases from 3.75 m^3 to 4.8 m^3, (a 28.0% increase) when one unit is in operation and the average speed drops from 8.0 to 6.0 km/hour. Figure 27, shows a similar effect (i.e., a 28.0% increase in rated capacity) when two LHDs are employed.

In addition to the assumptions listed above, the following parameters were used to calculate truck requirements for the haulage of ore up the main ramp to the dump located at 6970 Level:

- truck loading time: 5.5 minutes
- truck speed, loaded up: 5.1 kilometres/hour
- truck speed, empty down: 10.1 kilometres/hour
- average main ramp grade: 15.0%

The calculation procedure in this case is also a modified version of the one proposed by Wagner Equipment Co. (1987). The following formula can be used to determine the minimum truck payload that will meet production requirements for a certain one-way inclined tramming distance:

$$L = \frac{\frac{R}{n}}{\frac{t_{shift}}{t_l + t_m + \left[\frac{D}{16.67 * v_{up}}\right] + \left[\frac{D}{16.67 * v_{down}}\right]}}$$

where:

L: single truck payload, in tonnes

R: production rate, in tonnes per shift (note: two shifts per day)

n: number of trucks in fleet

t_{shift}: effective working minutes per shift (without delays or breaks)

93 These layouts correspond to the 45-metre lift height case that is analyzed in the following section.

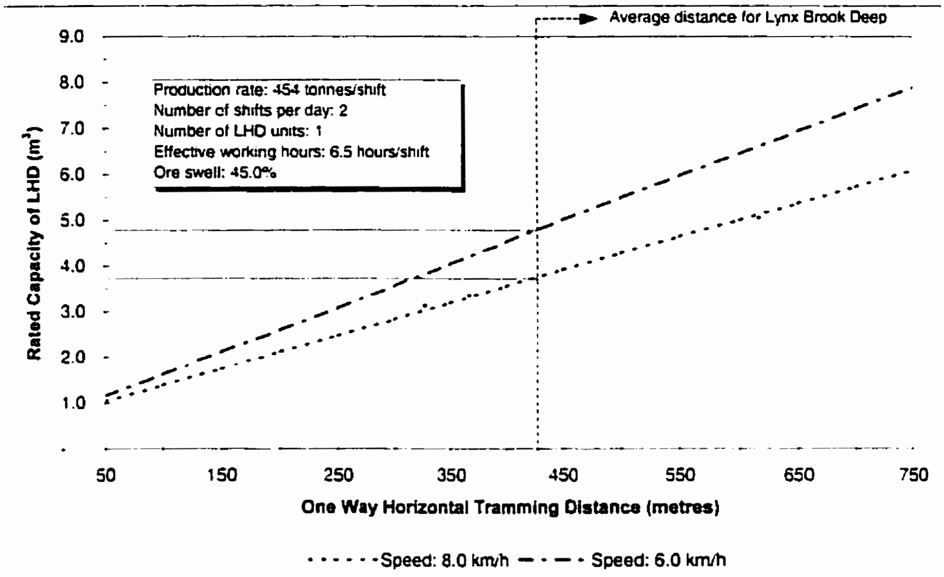

Figure 26: Lynx Brook Deep - LHD capacity versus tramming distance, one unit

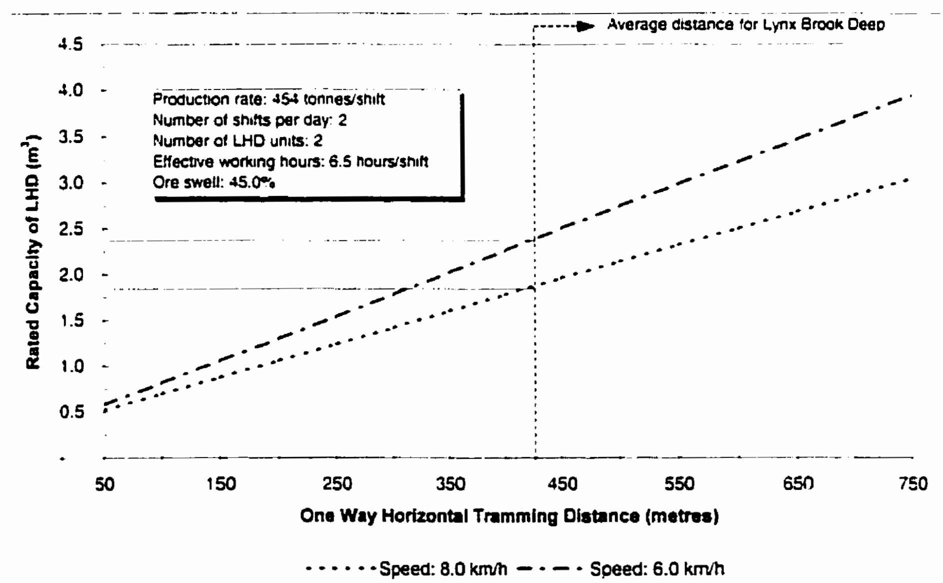

Figure 27: Lynx Brook Deep - LHD capacity versus tramming distance, two units

t_l: truck loading time, in minutes

t_m: fixed time to load/dump/maneuver each cycle, in minutes

D: one-way inclined tramming distance, in metres

v_{up}: average truck speed over distance D, loaded going up the ramp, in kilometres/hour

v_{down}: average truck speed over distance D, empty going down the ramp, in kilometres/hour

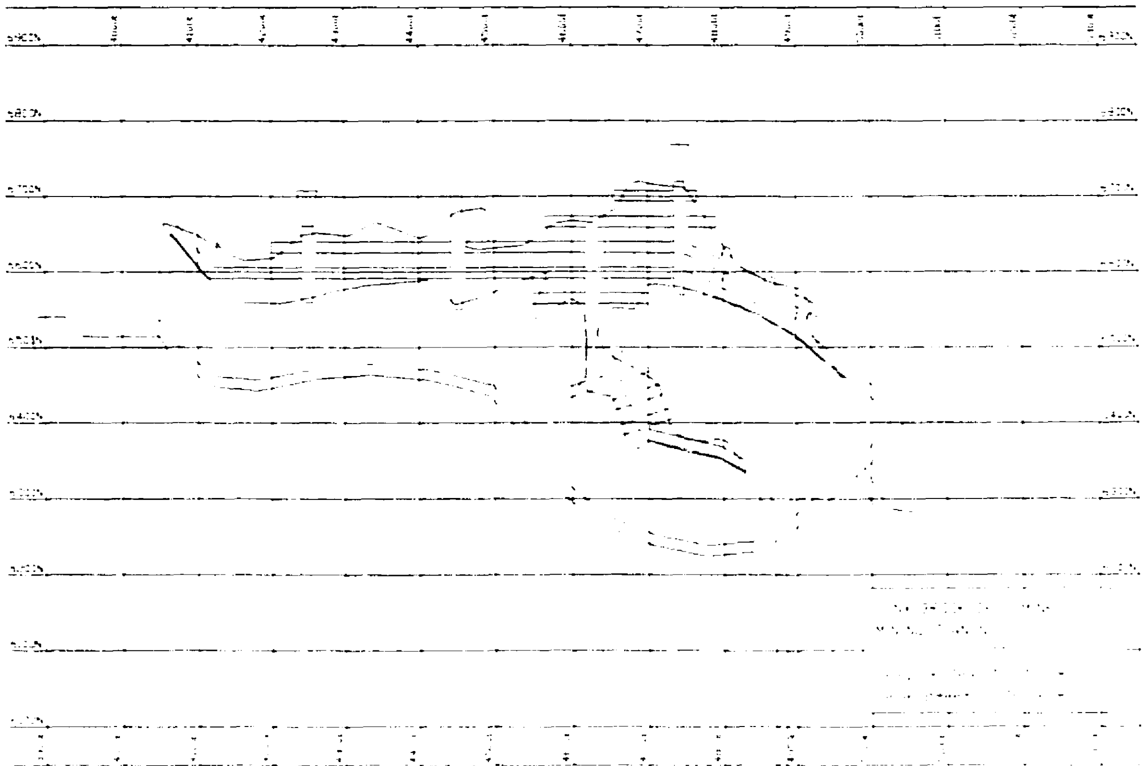

Figure 28: Lynx Brook Deep - Proposed development for 7350 Level

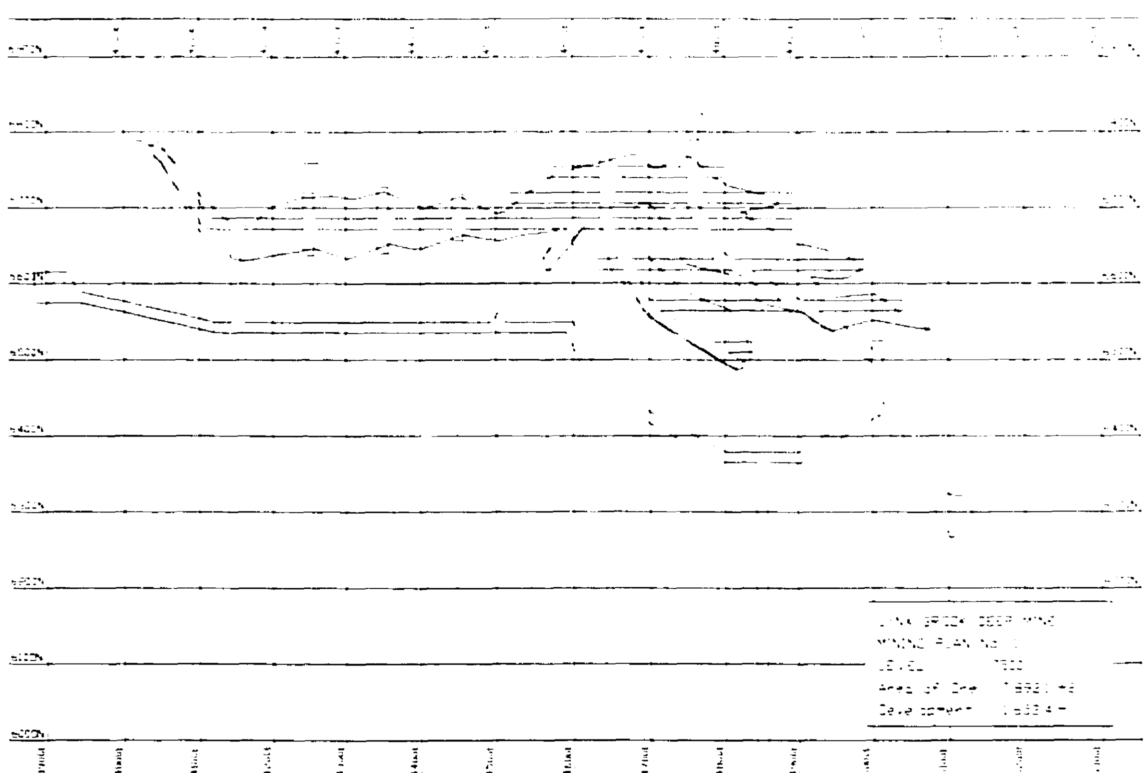

Figure 29: Lynx Brook Deep - Proposed development for 7500 Level

Truck fleets with one, two and three units were investigated with this formula (see Figure 30.). Since truck payload is inversely proportional to fleet size, significant reductions in required truck payload can be achieved by adding units to the smaller fleets only. Indeed, a 50.0% reduction in truck payload is accomplished by going from a one-truck to a two-truck fleet, but only a 33.3% payload reduction is obtained by adding one more unit to the two-truck fleet. This straightforward (inversely proportional) relationship is somewhat artificial, since other factors that *always* vary according to the size of the truck have been considered as fixed in the previous formula. These include truck loading time and time to load/dump/maneuver (smaller fixed times in smaller units), and average truck speeds: higher maximum speeds can be achieved normally with larger trucks (Wagner Equipment Co., 1987). If their true variability were also taken into account, real curves (not straight lines) would be plotted in Figure 30.

Three vertical dotted lines that depict one-way inclined tramming distances from 7190, 7350, and 7500 levels, respectively, to the dumping spot on 6970 Level are shown in Figure 30. On a first impression, it seems as if there would be a wide selection of truck sizes that could satisfy the production requirements down to 7500 Level, ranging from 7.5 to 40.0-tonne units. Most mine operators, however, would prefer larger trucks to reduce maintenance and repair costs, improve discounted cash flow indicators, and minimize labour needs.

Equipment size influences excavation dimensions, which should be minimized to facilitate ground control, and reduce mine development time and cost (see Section 4.4.2). In this case, the truck fleet at least would have to be able to meet 7190 Level production requirements, and then provide a reasonable fleet expansion path to cope with the increasing tramming distances from deeper levels. If only one truck were to be purchased initially (a likely option), it would have to be in the 25- to 35-tonne class. Although it would be over-designed for production from 7190 Level, the need for a second truck would not arise until 7350 Level has been mined out. The two-truck fleet would be capable of handling ore production requirements down to 7800 Level.[94]

[94] In fact, using the above formula and parameters (which can be considered as conservative), it was found that two 30.0-tonne trucks should be able to transport 454 tonnes/shift (eight-hour shifts) of ore from the 7800 Level. The one-way inclined tramming distance (15.0% ramp) would be about 1,950 metres.

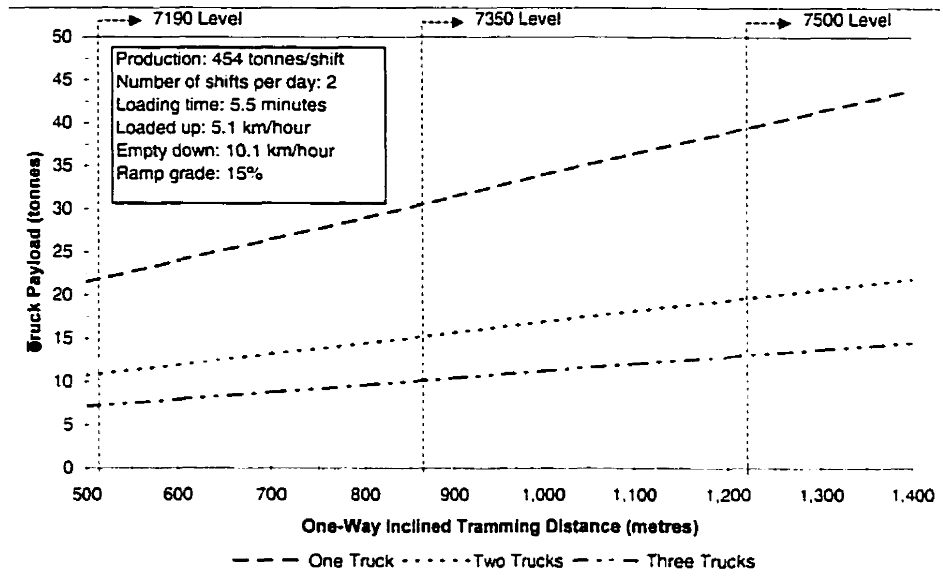

Figure 30: Lynx Brook Deep - Truck payload versus inclined tramming distance

LHDs in the 3.5 to 4.5-m^3 range match trucks in the 25.0 to 35.0-tonne class (a minimum of four passes should be needed to load a truck). This means that, even under the pessimistic scenario (average speed of 6.0 km/hour) a single 4.5-m^3 scooptram would be able to produce the required tonnage over the average horizontal tramming distance of 427 m (see Figure 26). However, it would not be able to sustain the required production rate if the tramming distance increases, as is expected to occur for a large percentage of stopes located in the edges of the orebody. The best choice in this case would be to acquire two 3.5-m^3 LHDs, which would be capable of producing 454 tonnes/shift of ore for horizontal tramming distances in excess of 750 m (see Figure 27)

It is quite evident that the selection and sizing of mucking and tramming equipment presents a number of options that, depending on the focus of the analysis, may or may not constitute so-called *optimum* alternatives. This is not an isolated case in mining equipment selection. It is also the case of hoisting systems (see Section 4.1.1) and almost every other underground mining subsystem. Minimization of capital expenditures and direct operating costs is only one of several factors to be considered in the final decision. Technical and economic studies are required to demonstrate the applicability of the proposed solution. An integral evaluation should ultimately determine its impact on the operation as a whole over the medium- and long-terms.

5.4.2 Applicability of Innovative Technologies

This section briefly considers the pros and cons of new technology adoption for horizontal muck transport at Lynx Brook. The discussions of the factors affecting deep underground mining (see Chapter 4) addressed the introduction of new technologies (i.e., hydraulic transport, automation, etc.). It was seen that they had the potential of minimizing the negative impact of the deep environment, and generating significant operational and/or economic benefits. At Lynx Brook, evidence of a challenging new mining environment at depth is provided by current trends in:

- ground stress;
- ventilation and air conditioning; and,
- orebody geometry and shape.

Indeed, ever-increasing ground control problems have prompted the development of strict mining sequences capable of re-distributing stresses efficiently while allowing high production rates from single mining units and enhancing overall productivity. Similarly, as discussed in Section 5.3, high rock temperatures have resulted in stringent air-conditioning and ventilation requirements, virtually eliminating the possibility of using diesel equipment for long hauls. On the other hand, it is likely that the narrow and tabular nature of the deep orebody (compare Figure 21 and Figure 31 below) will render current ground control and mine development practices ineffective (they were developed for wider ore). Therefore, there is a need to modify ***both*** current technologies and production methods. As discussed in Section 3.2.2.2, the simultaneous change of two variables of the product-market-technology significantly decrease the chances of success. It is reasonable, therefore, to assume that the mine operator would try to test new methods with existing technology in order to reduce the initial risks involved.[95]

Ventilation and air conditioning restrictions will make it very difficult to continue justifying the use of diesel LHDs for drawpoint mucking. Electric units could be used, but they would require the adoption of a rigid planning philosophy (to ensure that they are deployed in suitable stopes) and extensive use of ore passes, which are difficult and expensive to maintain in stressed ground.

[95] This assumption is validated by research carried out by Abegglen and Stalk (1985), who noted that the adoption of automation in Japanese factories followed several years of process improvement.

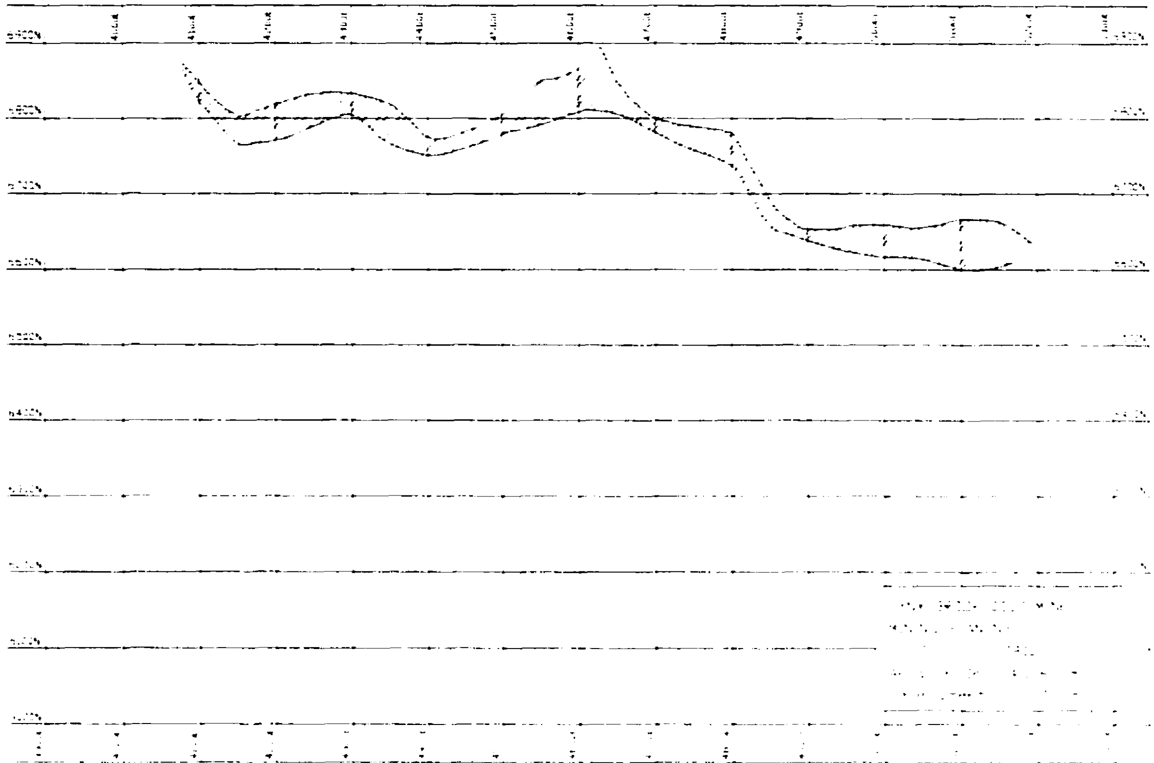

Figure 31: Lynx Brook Deep - Ore outline at 7800 level elevation

The main disadvantages of the use of continuous loading and conveying systems are related to the geometry of the deep orebody and a mining sequence dictated by ground control principles. In the light of previous experiences in deep Ontario mines (Pelley, 1994), and based on the current practices in the shallower sections of Lynx Brook, sound ground control most probably will call for thin and fairly high stopes at depth. However, such stopes may not be able to provide the high extraction rates required to justify the high capital costs of such systems. Moreover, they may not be large-tonnage stopes either. Short extraction periods increase both the complexity of the mine plan and overall operating cost, since transporting, installing, and setting up the loading and conveying equipment are time-consuming and labour-intensive (i.e., expensive) tasks.

Hydrotransport cannot be considered exclusively for horizontal transportation. In fact, as noted in Section 4.1.3, one of the main advantages of hydrohoisting is that it facilitates the integration of several components of the mining process (i.e., hoisting, cooling, drainage, etc.). Therefore, its use for horizontal transport would be justified only as part of a comprehensive plan that radically changes the entire muck transport system and targets ventilation and pumping issues as well.

In summary, it is very likely that the development and exploitation of Lynx Brook Deep will be carried out with conventional LHD-based systems for horizontal muck transport. The application of non-traditional transport systems such as continuous loaders and slurry pumping would conflict with current mining practices and may not be economically feasible, given existing ground stresses and orebody geometry. Moreover, the full benefit of new technology introduction can be achieved only when a comprehensive approach to the problem is adopted. Thus, it would require the concurrent and extensive modification of both vertical and horizontal transport systems. It is believed that this is neither practical nor economically sound in the case of Lynx Brook Deep.

5.5 Mine Development

Based on the structure of Section 4.4 and relevant aspects of mine development at Lynx Brook, this section reviews and discusses critically issues related to production rate, dilution, ore recovery, the dimensions of development openings, and horizontal (i.e., lateral) development.

5.5.1 Production Rate

Table 1 indicates that a downward trend in ore reserves (on a per-level basis) is expected as the mine deepens and the orebody thins. Indeed, at the 7800 Level the deposit could be considered as a vein for all practical purposes, including mine design and sequencing (compare the outlines shown in Figure 21 and Figure 31). This will be certainly a deciding factor in mine development and production plans, and indirectly will affect the determination of the production rate.

5.5.1.1 Application of Rules of Thumb

Two of the rules of thumb discussed in Section 4.4.3.1 (Taylor's law and the "seven-year minimum mine life criterion") can be used to determine quick estimates of the preliminary production rate for Lynx Brook Deep. Their validity in this case, however, should be cautiously investigated, since they are not directly applicable to the expansion of deep underground mines. The corresponding production rates would be:

- Taylor's Law:

$$T_{5-day} = \frac{4.88 * (5{,}283{,}258)^{0.75}}{250} = 2{,}151 \text{ tonnes/day}$$

$$T_{7-day} = \frac{4.88 * (5{,}283{,}258)^{0.75}}{350} = 1{,}536 \text{ tonnes/day}$$

- Seven-year minimum mine life criterion:

$$P_y = 5{,}283{,}258/7 = 754{,}751 \text{ tonnes}$$

The respective daily production rates would be 3,019 tonnes (five-day week) and 2,156 tonnes (seven-day week).

5.5.1.2 Discussion

The mining sequence and current extraction facilities impose serious restrictions to the optimum sustainable production rate. As seen in Section 5.3, the most important limitation to an increased (or, with depth, sustained) production rate comes from the existing hoisting system. In 1994, it hoisted 881,415 tonnes of ore, which is equivalent to 3,526 tonnes/day assuming a five-day week. As currently configured, it can hoist 213 tonnes/hour from the deepest loading pocket. If we assume 17 *effective* hoisting hours per day (Edwards, 1992), this would result in a maximum hoisting rate of 3,621 tonnes/day (ore and waste). Existing ore reserves in upper levels allow a higher global production rate for the entire complex. It must be remembered that it will not always be possible to dispose of all of the waste produced by mine development underground and, thus, some of it will inevitably end up being transported to surface. Given the size of development openings, the tonnage of waste generated is significant and could further limit the ore-hoisting capacity. It is interesting to see that the rates obtained with Taylor's law and the seven-year mine life criterion are within the limits of the existing production system.

Speed of mine development restricts the production rate of deep mines (Taylor, 1986). At Lynx Brook, this is true for two reasons. First, the mining method requires extensive and costly in-ore and wall development, which must be completed ahead of production. The operational restrictions and large financial requirements involved in properly addressing those issues probably would prevent a higher production rate than that is presently achievable.[96] Second, the existing ore reserves do not justify sinking a new shaft from surface to hoist larger tonnages of ore and waste.[97] An additional ore handling system would have to deliver muck from the deeper levels to the existing loading pocket. This is expensive and results in a complex transport system.

5.5.2 Dilution

Two types of dilution must be differentiated. Some dilution is an intrinsic part of the minable reserves. It is generated by internal waste that cannot be segregated from the surrounding *economic* ore. It is also created by waste coming from irregularities in the hangingwall/footwall contacts that cannot be avoided at the scale of the equipment or mining method employed. The mining plan takes into account this *built-in* dilution and the entire mine/mill system is designed accordingly. The ongoing switch to large-tonnage underground mining methods, for instance, has resulted in design dilution figures that would have been considered unacceptable only 10-15 years ago, when more selective mining methods were popular. Indeed, Pelley (1994) concluded: "*... grade control and dilution ... have been adversely affected by ... bulk methods*".

On the other hand, dilution also results from inadequate mining practices, poor choice of mining method and/or equipment, and sudden, unexpected changes in ground conditions or geological characteristics of the orebody. This *operational* dilution comes from barren rock and/or backfill and (at least according to the mine design and production plan) should have been avoided by properly following the established mining/operational procedures. It is not related to parameters such as the minimum mining width or ground support system, and is added to the *built-in* dilution previously discussed. In general, operational dilution can be managed by applying sound ground control measures, adopting appropriate mining sequences, and efficient drilling and blasting.

[96] Careful application of the maximization of the net present value criterion can help in determining the economic feasibility of expanding the existing underground rock handling system (see, for example, Section 5.3).

[97] As discussed in Section 5.3.1, there are also design difficulties with this option, since increased depths of hoisting would lower the safety factor below 5.0, the minimum required by Ontario regulations.

Dilution can only be controlled at a cost, either by adopting additional ground control systems, switching to more selective mining methods or reducing the extraction rate[98] to lessen contamination from the walls. Dilution control is an economically feasible option as long as the increased production cost is less than the economic benefits directly derived from reduced dilution. The problem is, the determination of such benefits is often not easy or straightforward.

The difficulties in dealing with and properly identifying and quantifying dilution are illustrated by the Lynx Brook case. The operator reported that, regardless of inter-level spacing (lift height), the average dilution for 21 stopes from 6700, 6900, and 7200 levels was 14.0%. However, if we assume that the figures made available to the public are correct (Scales, 1996), a 14.0% dilution does not explain a 28.0% decrease in grade[99]. Furthermore, the reported 1994 production grade represents a 15.0% decrease from the reported *in situ* grade (Scales, 1996). It is not clear if the 14.0% figure mentioned by the operator corresponds to the *built-in* or the *operational* dilution, since it seems as if the *in situ* reserves already account for some kind of dilution.

5.5.2.1 The Cost of Dilution

The most obvious consequence of dilution is the reduction in the profitability of an operation by increasing the mining (production) cost as expressed in dollars per unit of commodity produced.[100] This, unfortunately, is not the only problem with dilution. It has a dual economic effect on an underground operation: lost production due to material with no value taking the place of ore in the production process, and resources being wasted on barren material. In an open pit, barren or low-grade material could be sent to a waste dump or to a stockpile for later processing, if required (Koniaris, 1991). However, in an underground mine dilution results in the extraction, transportation, and processing of material with no economic value. Thus, it is not only the revenues lost due to the smaller metal production rate that must be considered in the analysis, but

[98] Typically, smaller sections of the hangingwall are exposed simultaneously (and for shorter periods) when lower extraction rates are used, significantly facilitating dilution control.

[99] Since mineralization at Lynx Brook is fairly regular, an average *in situ* ore grade was assumed (the actual value was not made available). If the ore at the upper levels (which currently contributes about 75.0% of total production) is similar to the one found at depth, it could be argued that this is an overly conservative assumption.

[100] Obviously, the overall production cost in terms of dollars per ton remains the same, but the resultant lower grade due to dilution increases the cost of mining each unit of metal contained in the ore produced.

also the resources wasted in mining and processing material with no revenue-generating capabilities. The total cost of dilution can be calculated as follows:

$$\text{Total Dilution Cost} = \text{Lost Revenue} + \text{Resources Wasted}$$

Table 21 shows the value of the saleable metal contained in a tonne of diluted ore produced at Lynx Brook during 1994 (1994 metal prices were used). Because the actual mill recovery could not be obtained, a figure lower than the one reported for a similar operation was used instead (Scales, 1996). In that year Lynx Brook ore had a value of US $ 139.0/tonne, i.e., equal to gold ore having an average diluted grade of 0.366 oz Au/tonne[101] (11.4 g/tonne) which can certainly be considered as high-grade ore. For every tonne of waste and/or backfill that became mixed with real ore, the operation lost US $ 139 (CAD $ 187.65) of revenue.[102] Table 21 also shows that the value of diluted ore was 82.5% of in situ ore. Then, if waste material contained no saleable metal, each tonne of in situ ore was mixed with 0.213 tonnes of waste (i.e., 21.3% dilution).

The assessment of the effects of dilution is further complicated by the lack of adequate cost data. For instance, Table 22 shows pertinent[103] Lynx Brook costs items taken directly from the summary report provided by the operator. Based on such data, it was determined that the cost of bringing one tonne of material (either ore or waste) to surface was $27.95. On the other hand, basic cost data were analyzed and re-aggregated in Section 5.2.3 and the results shown in Table 4. Using the new data, the cost of dilution was determined to be $36.90/tonne[104] (see Table 23), or 32.0% higher than the previously calculated figure.

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[101] Value calculated using a gold price of US $ 380.00/oz and a mill recovery of 90.0%.

[102] The implicit assumption is that the mine exceeds production capacity. In other words, there exists ore that could be mined and hoisted instead of the waste originating dilution. See Koniaris (1991) for a discussion on stockpiling and production strategies.

[103] It has been assumed that only direct costs incurred ***after*** blasting are relevant to the analysis of dilution. Other components of the mining cost such as ventilation, management, and overheads (that may or may not be applicable, depending on the focus of the analysis) have not been considered in this case. Mine development, drilling, and blasting definitely are not relevant components in this case, since these activities are not carried out with the purpose of extracting waste.

[104] A more complete estimate of the cost of dilution would include the ore processing cost. This is because such a cost (in $/tonne of material processed) does not significantly change with the grade of the ore. In fact, regardless of grade, the material has to be crushed, ground, floated, etc.

Table 21. Lynx Brook Mine - Value of diluted ore, 1994 Data

Value of Ore		Mill Recovery	Value of Saleable Metal Contained
in situ Reserves	Diluted		
US $/tonne	US $/tonne	%	US $/tonne
187.2	**154.4**	**90**	**139.0**

Table 22. Lynx Brook Mine – Cost of dilution: original data

Cost Item	Unit Cost ($/tonne)
Mucking	2.64
Haulage	2.63
Underground mobile equipment fuel oil	0.33
Underground transport equipment repairs	4.98
Backfill	4.34
Crushing and conveying	2.43
Hoisting	3.76
Surface handling	0.95
Electric power	5.89
Total	**27.95**

Table 23. Lynx Brook Mine - Cost of dilution: re-aggregated data

Cost Item	Unit Cost ($/tonne of ore)
Mucking	6.56
Backfill	4.34
Tramming	3.39
Crushing, conveying, hoisting & surface handling	7.15
Underground upkeep and support	7.33
Electric power	5.89
Ore transport to processing facility	2.24
Total	**36.90**

Waste and backfill dilution must be calculated to estimate the total cost of dilution for Lynx Brook in 1994. The operator's formula for calculating dilution (as a percentage) is as follows:[105]

$$\text{dilution} = \frac{\text{tonnes of waste rock} + \text{tonnes of backfill}}{\text{tonnes of ore}} * 100$$

We know that total material produced in 1994 (ore plus dilution) was 881,415 tonnes and that average dilution was about 21.3%. Thus, replacing the values in the above formulae we obtain:

$$\text{tonnes of waste} + \text{tonnes of backfill} = \frac{21.3 * 881{,}415}{121.3} = 154{,}774 \text{ tonnes}$$

Total lost revenues due to dilution are:

$$\text{Lost Revenue} = 154{,}774 \text{ tonnes x } \$187.65/\text{tonne} = \$29{,}043{,}341$$

Depending on the data used to determine the per-tonne extraction cost, the cost incurred in extracting waste and backfill as dilution would be:

$$\text{Resources Wasted}_{\text{Original Data}} = 154{,}774 \text{ tonnes x } \$27.95/\text{tonne} = \$4{,}325{,}933$$

$$\text{Resources Wasted}_{\text{Re-aggregated Data}} = 154{,}774 \text{ tonnes x } \$36.90/\text{tonne} = \$5{,}711{,}161$$

Assuming the re-aggregated data provides the best estimate of the actual costs involved in ore extraction, the total cost of dilution at Lynx Brook in 1994 was:

$$\text{Total Dilution Cost} = \$29{,}043{,}341 + \$5{,}711{,}161 = \$34{,}754{,}502$$

It could be argued about the validity of using lost revenues to determine the cost of dilution.[106] However, the second component (i.e., resources wasted) represents actual resources employed in manipulating barren material and, thus, the figure obtained has a very real (and measurable) meaning. Indeed, it represents 7.7% of the total production cost at Lynx Brook.

105 A variety of methods for calculating and reporting dilution is used in the mining industry (Tapp, 1982; Murray, 1982; Cokayne, 1982).

106 As calculated here, such a figure implies that the mine is capable of replacing the tonnage of dilution with ore at the average grade. This may not be a feasible proposition in some bulk underground mines.

5.5.2.2 Discussion

Dilution usually increases as the thickness of the orebody decreases (Florez, 1982). This is because the ratio of surface area of hangingwall and footwall contacts to ore volume increases as the thickness of the orebody decreases. In other words, there is a higher percentage of *border* panels (i.e., panels which are limited on at least one side by either the hangingwall or footwall) in thinner orebodies. Therefore, since the mining method will not change drastically in the near future, it is reasonable to expect increased dilution figures in Lynx Brook Deep. Moreover, the shallower dip of the orebody will result in inclined panels (between 50° to 65°) which, in turn, will produce increased dilution coming from both the hangingwall and adjacent backfilled panels.

Depending on the configuration of the orebody, another method of controlling dilution is by sacrificing ore recovery. Some of the drawbacks of this option are increased development cost (expressed as dollars per unit of commodity recovered) and shorter mine lives (since the ore reserves are reduced). However, the profitability of the operation may benefit significantly, mainly if level upkeep and ground support costs can be reduced.

5.5.3 Ore Recovery

Ore recovery is the percentage of minable reserves that is extracted during the mining process. Maximization of ore recovery and dilution control are in most cases opposing objectives. Indeed, increased ore recovery usually affects dilution negatively and vice versa. These two issues must be balanced so that profitability is improved (as measured by the return on the investment) and, at the same time, the efficiency of the operation is not affected. The switch from labour-intensive, selective mining methods to high-tonnage bulk mining has undermined the ability to control ore recovery (Pelley, 1994). At Lynx Brook, ore recovery is affected by three main factors:

a. Ore *wedges* at both the topsill and bottomsill of the panel cannot be extracted because they provide access to mined-out areas for backfill purposes. At least in theory, they could be extracted **after** the panels that surround them (in the levels above and below them) have been mined and backfilled. However, the cost involved would certainly exceed the potential additional revenue, not to mention the resources that could be best employed in other areas.

b. Ore left against backfill. There is usually a certain amount of ore immediately adjacent to the backfill that cannot be retrieved due to irregularities of the ore-backfill contact surface and limitations of the drilling pattern.

c. Oxidization of ore stuck to the footwall. This is a direct consequence of the extraction sequence: ore blasted at the beginning of the extraction phase must remain in the stope until the last slice of ore is blasted and extracted. In large panels, and due to increased lift height and/or horizontal cross-sectional extent, this can be a considerable amount of time. For example, if a panel has a horizontal area of 120 m^2 and each slice is 3.0 metres high, every blast would produce about 1,500 tonnes of ore to be mucked in three days. If the lift height were 60 metres, ore from the first few blasts would have to remain in the stope for more than two months and one half before being mucked (assuming a five-day week schedule).

Table 24 shows dilution and ore recovery estimates for one cross-section of Lynx Brook Deep. It can be concluded that:

1. Ore losses increase as the inter-level spacing (i.e., panel height) increases. In this particular section, an increase from 30-m to 60-m lift height would increase total ore losses by 61.0% (from 12.26% to 19.71%).
2. Whereas losses in ore wedges tend to become less significant as the panel height increases, losses due to oxidization and ore left against backfill rapidly scale up.
3. In spite of the fact that the same mining method was assumed in every case (vertical retreat mining),[107] there is a good direct correlation between ore losses and dilution: an increase in lift height increases both dilution and ore losses.

However, Table 24 does not reveal one effect of increased lift height: the absolute value of backfill dilution has remained constant, although (in theory) it should increase both in value and as a percentage as the height of panels increases. This is due to the larger area of backfill walls exposed to blast damage over the longer periods required to extract higher, larger-tonnage panels. As the orebody becomes thinner, shallower, and more tabular in nature, ore recovery at Lynx Brook Deep will certainly deteriorate. This will be particularly critical to the operation if efforts

[107] As opposed to modifying the mining method to take advantage of, for example, the increased inter-level spacing and thus facilitate the recovery of a larger percentage of the ore reserves. Such a modification, however, could result in extremely high dilution figures.

are made to reduce dilution in order to improve its economics. In any case, several dilution/ore recovery scenarios would have to be studied to determine the optimum from both operational and economic viewpoints.

5.5.4 Sizing Extraction and Access Openings

The issue of determining the cross-sectional dimensions of extraction and access openings was discussed in Section 4.4.2. As is the case of the vast majority of underground mines, the largest possible (i.e., available) equipment is being used at Lynx Brook. Nonetheless, and in the light of the ventilation, ore transport, dilution, and ground control problems recently encountered, it is very likely that it would benefit from a review of its excavation and equipment sizing practices. In fact, they have the potential of improving significantly the overall efficiency and profitability of the deepening project. However, the existing mining operation constitutes the major constraint to reducing the size of development openings and mobile equipment, and modifying operational practices. Two main areas of concern can be identified:

- ***Compatibility issues***

  Smaller equipment would not be compatible with the larger units that would still be in operation in the upper levels. Indeed, older equipment would not be able to access the deeper sections of the mine. There would be an increase in the maintenance and inventory costs, as more spare parts would have to be kept in stock. Furthermore, depending on the new equipment's type and manufacturer, additional maintenance crews may be needed.

- ***Mining/operational philosophy***

  As noted in Section 4.4.2, the advantages of smaller development sizes can be fully realized only when they are coupled with simultaneous changes in operational practices. At Lynx Brook this would mean improving both grade and ground control. This is not simple, since it would imply extensive training of supervisory and hourly personnel so that the benefits of the proposed changes are clearly understood. The bulk mining nature of the operation would not be altered by the use of smaller openings and equipment, but the new emphasis on more effective and efficient mining would require a long transition from the current methods.

Table 24. Lynx Brook Mine - Ore losses and dilution for selected inter-level spacings

Inter-Level Spacing	Ore Losses								Dilution					
	Wedges		Against Backfill		Oxidized in Stope		Total		Walls		Backfill		Total	
metres	tonnes	%(*)	tonnes	%(*)	tonnes	%(*)	tonnes	%(*)	tonnes	%(*)	tonnes	%(*)	tonnes	%(*)
60	2,423	7.47	2,418	7.46	1,551	4.78	6,392	19.71	7,172	22.12	286	0.88	7,458	23.00
45	1,792	8.19	1,058	4.83	708	3.23	3,558	16.25	3,318	15.17	286	1.31	3,604	16.48
40	1,462	7.86	605	3.25	490	2.63	2,557	13.74	2,233	12.00	286	1.54	2,519	13.54
30	1,462	10.17	101	0.70	200	1.39	1,763	12.26	1,425	9.91	286	1.99	1,711	11.90

(*) As a percentage of geological resource in the panel

It is believed that Lynx Brook should evaluate the option of using smaller openings in order to optimize its deep operation. Even if the results did not justify such a drastic change, the exercise would serve to identify potential areas for operational improvement.

5.5.5 Horizontal Mine Development

From a strategic perspective, there are two important horizontal mine development issues:

- cross-sectional dimensions of the openings; and,
- relationship between horizontal development and ore reserves.

The topic of development opening sizing in general has been dealt with in Sections 4.4 and 5.4, as well as in Section 5.5.4 above. In this case, having determined that it is very likely LHD-based systems will be used for horizontal muck transport, the size of development openings will be dictated most probably by ventilation requirements, regardless of the dimensions of mobile extraction equipment. Development cost *per se* would not become a deciding factor, given the importance of ventilation and air conditioning issues in a deep environment and the virtual lack of alternatives for horizontal muck haulage.

It has been noted already that the orebody adopts a tabular nature and significantly thins with depth. Thus, if current development practices are maintained (i.e., simultaneous footwall and hangingwall access), the horizontal development cost in dollars per tonne of ore will certainly increase. Hangingwall access in addition to standard footwall development is the result of the mining sequence developed to control stress re-distribution at depth. As such, it can be eliminated only if the new sequence demonstrates that it can provide better ground control management capabilities. This aspect of mine development is discussed further in the following section.

Two additional incentives to rationalize mine development are current limitations in hoisting capacity and the high cost of maintaining openings at depth. The mining sequence determines the speed of ore extraction, which in turn defines the need to develop and maintain infrastructure at one or more levels simultaneously. The current sequence advances one level at a time, but geotechnical considerations and the reduced unit extraction rate typical of a thin orebody may require switching to a multi-level sequence such as Kidd Creek's (see Figure 11). A multi-level operation results in higher development rates (metres/month), ventilation and air conditionings requirements, and level upkeep costs. This would affect Lynx Brook severely, since it already has serious problems in such areas.

5.6 Evaluation of Mine Development Alternatives

The objective of this section is to analyze and discuss concisely mine development alternatives available for the exploitation of the deeper ore at Lynx Brook. This is an issue of the greatest importance for the operator, from both investment[108] and operational perspectives. The study focuses on the orebody between the 7190 and 7800 Levels. Such levels define the upper and lower limits of the orebody information made available by the mine operator.

Two development aspects have been identified that could potentially become the starting point of a new *mining philosophy* at Lynx Brook: inter-level spacing and hangingwall access.[109] It should be noted that a detailed evaluation and quantification of the alternatives is beyond the scope of the thesis. In fact, the purpose of the study is to discuss and analyze options available, but not to produce actual mining plans or investment programs. Nonetheless, an effort was made to establish the impact of alternatives on the long-term profitability of the operation.

5.6.1 Main Assumptions and Methodology

The analysis is based on the information provided by the operator and summarized in Section 5.2. Other basic assumptions include:

[108] The operator estimates that the development of the 7400-7600 level will require investing between \$ 14.2 million and \$ 19.6 million in new equipment, and between \$ 9.0 million and \$ 14.1 million in underground construction. The variation is due to different mining methods and inter-level spacings. As discussed in Section 5.3, just the vertical ore transport system needed to exploit the ore down to 7800 Level would demand an investment of close to U.S. 10.0 million.

[109] A third development topic (extraction and access opening sizing) was also deemed significant from a strategic viewpoint. However, after completing the discussion on LHD and truck requirements (see Sections 5.3.3 and 5.4.1) it was evident that, for the study to produce relevant results, it would have to consider additional site-specific information. Moreover, such a study would have to cover mine development issues other than direct capital and operating costs (e.g., ventilation and the delivery of personnel and materials). Since pertinent information was not supplied by the operator, and cost data was too aggregated to allow detailed studies without having to rely on *first principles* estimations, it was concluded that it could not de considered for the current analysis.

a. Mining method: vertical crater retreat mining as applied currently at Lynx Brook. Panels with 10.2 m x 10.2 m horizontal section would be mined using the sequence described previously (see Section 5.2.2).

b. Extraction (ore and waste transportation) method: standard diesel-powered LHDs for mucking and horizontal tramming and low-profile trucks for ramp tramming.

c. Excavation costs were taken from a spreadsheet supplied by the operator. Costs per metre for different drift/ramp sizes are as follows:[110]
   - 3.6-m x 4.2-m drift: $2,050.5[111]
   - 4.2-m x 4.2-m drift: $2,017.7
   - 4.2-m x 4.2-m ramp (gradient = 15%): $2,427.8[112]

d. Dilution and ore recovery are not considered in the analysis, i.e., dilution is zero and mining recovery is 100%. As will be discussed later, these aspects deserve special attention and warrant additional research to investigate their effects on the findings of the present project.

e. Ore and waste must be delivered to the dumping spot on 6970 Level.

f. Topsill development is carried out immediately underneath the backfilled bottomsills.

g. No mine layouts of the existing operation were provided: the AutoCAD files contained only orebody outlines for several elevations, cross-sections, and longitudinal sections. Some liberty was taken to define the location of the main access ramp and other existing underground installations.

The approach adopted is straightforward but time-consuming: first, three-dimensional AutoCAD mine layouts are used to determine the generalized development schedule for the option being investigated. Then, the mining sequence is established taking into account the guidelines discussed above. Finally, the cost of development is determined (both capital and production/mine development) and the results analyzed. Due to the time-consuming nature of the process described, only

[110] The original spreadsheets were modified to better reflect the conditions of this case study and protect confidential information.

[111] The apparent contradiction of a smaller size drift being more expensive to excavate is the result of different equipment being used in each case. In the opinion of the author, the 3.6-m x 4.2-m drift is being unfairly *penalized* (the truck employed is too small for the volume of muck produced).

[112] The original spreadsheets did not consider such a small ramp (the smallest ramp included was a 5.0-m x 4.8-m). The figure shown here was calculated multiplying the unit cost of a horizontal drift of similar dimensions by a factor of 1.2. The factor was determined after analyzing the development cost of drifts and ramps of similar sizes.

one complete set of mine layouts, development program, and mining sequences was developed. The figures thus obtained were then used to estimate that of the remaining alternatives.

5.6.2 Inter-Level Spacing

Underground operators have given special attention to the issue of inter-level spacing. This highlights the importance of what it is believed constitutes a major factor in the future of deep mining. Three different inter-level distances have been considered for mining the deep orebody between 7190 and 7800 levels: 30.5 metres, 45.7 metres, and 61.0 metres. They imply the development of seven, five and four main levels, respectively. The 45.7-metre option, which requires five main levels, was the only alternative developed in detail. Development and cost figures for the 30.5 and 61.0-metre cases were obtained by averaging values from the fully developed 45.7-metre option. The objective of the exercise is to quantify the impact of adopting different level configurations and determine its significance to Lynx Brook Deep.

5.6.3 Hangingwall Access

A major drawback of the current mining method at Lynx Brook Mine is the requirement of both footwall and hangingwall access. Indeed, the mining sequence dictates that it is necessary to mine from the centre of the orebody toward the orebody extremities in order to establish the initial diamond-shaped slot. Such a slot is required for ground control. The development of footwall and hangingwall access is extremely expensive, in terms of both excavation and maintenance costs. It also significantly delays the development program. On the other hand, it provides some flexibility to (and simplifies) the mining process.

The most obvious way around having simultaneous hangingwall and footwall development is to excavate through backfilled panels (Pelley, 1990). Major drawbacks of this solution include the large amounts of waste backfill produced, the danger of backfill collapse, and ground control problems. However, this limitation may soon become academic: as illustrated by Figure 28 and Figure 31, the thickness of the orebody drastically reduces with depth and becomes tabular-like. In fact, of all the levels considered in this case study, only 7500 could be considered as *massive* (i.e., three-dimensional, see Figure 29). The need for hangingwall access disappears as the horizontal dimensions of the mining panels approach the thickness of the orebody.[113] In this case, occasional hangingwall development would be needed for ventilation purposes only.

Horizontal development plans produced for this case study demonstrate (at least in theory) that simultaneous footwall and hangingwall access is not required for the deeper areas of Lynx Brook. Only footwall development was considered. In fact, due to the reduced thickness of the orebody, it was very difficult to develop the diamond-shaped slot in most levels. The final sequence virtually resembles that of a vein-type orebody: opening up the orebody in the middle (i.e., in longitudinal section) and retreating toward the extremities. In thin ore, such a sequence would provide the same kind of stress re-distribution as the one based on the diamond-shaped slot.

Some backfill development would be required, but only on those levels in which the orebody is still thick enough to warrant it. The return raises are all located in the hangingwall, thus additional excavations in backfill also will have to be carried out to maintain the airflow.

5.6.4 Analysis of the Results

The process of constructing each mine development option starts with the study of orebody configuration. Figure 32 shows a plan view of level 7190 and the grid system employed for sectioning. Figure 33 depicts a cross-section looking northwest that more adequately displays the true thickness and dip of the nose that intrudes into the footwall rock (see Figure 32 to locate the section on plan view). Figure 34 and Figure 35 show cross-sections looking east at the 4200 and 4700 east coordinates, respectively. All sections display mine development excavations (access drifts, crosscuts, ramps, and ventilation raises). Orebody sections were obtained from the three-dimensional solid model provided by the operator. It can be seen that the deep orebody is irregular and tends to thin with depth, becoming only a thick vein at 7800 Level. The footwall nose, which accounts for a sizable percentage of the reserves of the upper levels, poses a major

[113] In fact, as far as the mining sequence is concerned, once the orebody thickness is reduced to about twice the horizontal dimensions of the panels, there is virtually no reason to have hangingwall access. The mining sequence in that case can start by mining the hangingwall panel first and then proceeding with the footwall panel. Alternate panels would be mined so that the number of exposed backfill walls is reduced.

challenge to mining due to its proximity to the main orebody and shallow dip (< 40°). As illustrated clearly by Figure 33 and Figure 35, the nose disappears midway between 7500 and 7650 levels.

Figure 36 provides a three-dimensional view of all the development work for the exploitation of the deep orebody between 7190 and 7800 levels. In addition to footwall and in-ore level development (topsill development for 7190 Level and bottomsill development for all other levels), the main access ramp[114] and four ventilation raises are shown. Some of the minor excavations (refuge and re-muck stations, lunchrooms, etc.) have been omitted since they do not affect the analysis.

Figure 37 depicts the proposed mining sequence for the 7500 Level. Even in this level, the thickest of those analyzed, the creation of a *diamond-shaped* slot is not obvious. It could be argued that it only works as originally designed in the eastern section of the orebody, where the presence of the footwall intrusion to the South of the main orebody requires the extraction of several waste panels. However, such a support mechanism is not required in thin areas, where three or fewer panels are mined across the orebody.

Table 25 shows a summary of horizontal development for the 45.7-m option.[115] It is evident that, as the orebody becomes thinner and tabular with depth, the amount of horizontal development decreases (in absolute figures). In general, the thinner the orebody, the more development per tonne of ore it requires. However, the geometry (in plan view) of the deposit also plays an important: role: 7350 Level has 93% of the reserves of 7500 Level, but requires 7% more horizontal development due to its irregular outline. Similarly, because of its elongated shape, 7800 Level demands 97% of the horizontal development of 7650 Level, although it only has 74% of its reserves.

[114] Since the location of the ore was known with a fair degree of accuracy, it was possible to design a ramp with long straight segments and very few curves. As noted by Walker (1988, pp. 242-244) this improves safety, facilitates road maintenance, and increases productivity by allowing higher maximum speeds.

[115] Vertical and ramp development, although considered in the plans, are not included in the final analysis since they are virtually constant, regardless of the inter-level spacing.

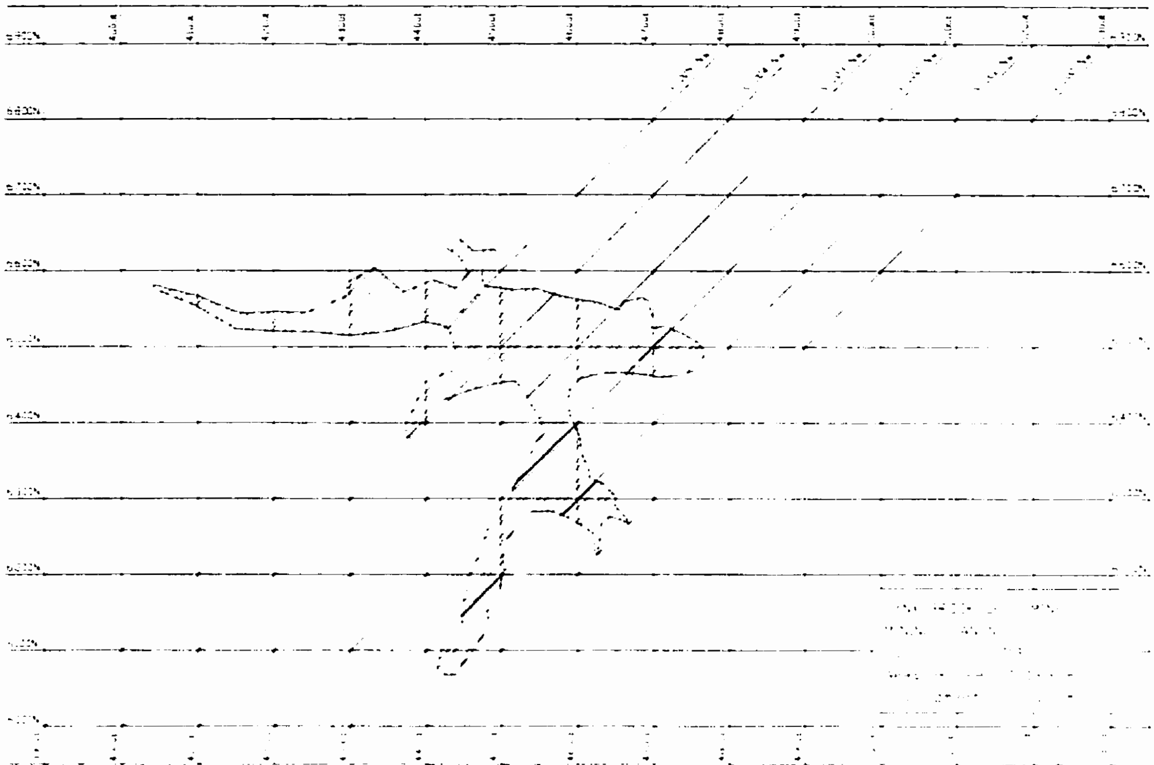

Figure 32: Lynx Brook Deep - Ore outline at 7190 level elevation and NW sections

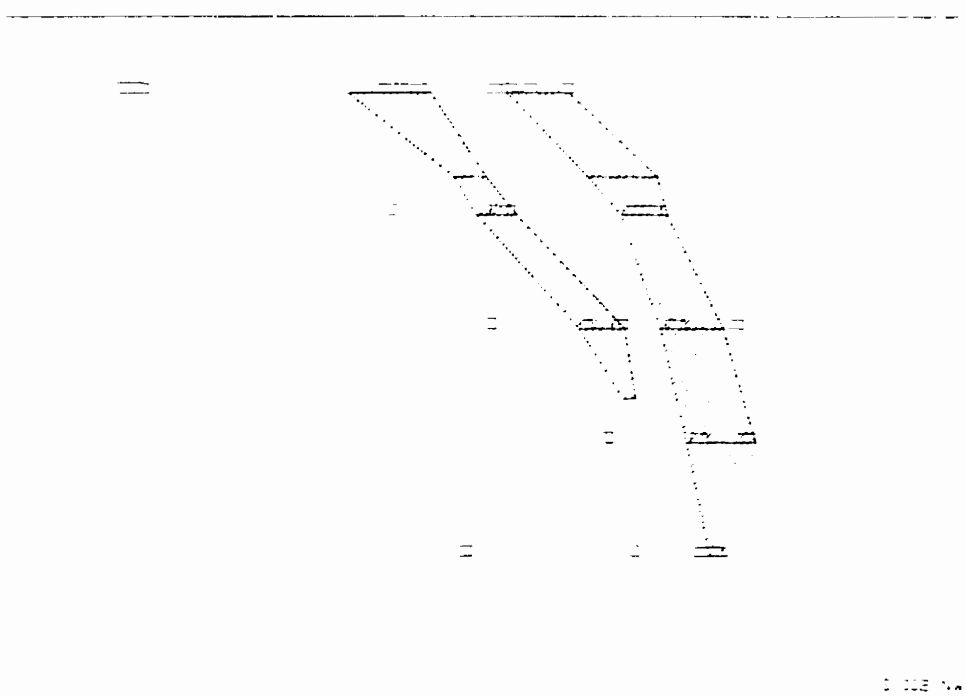

Figure 33: Lynx Brook Deep - Section 002 northwest

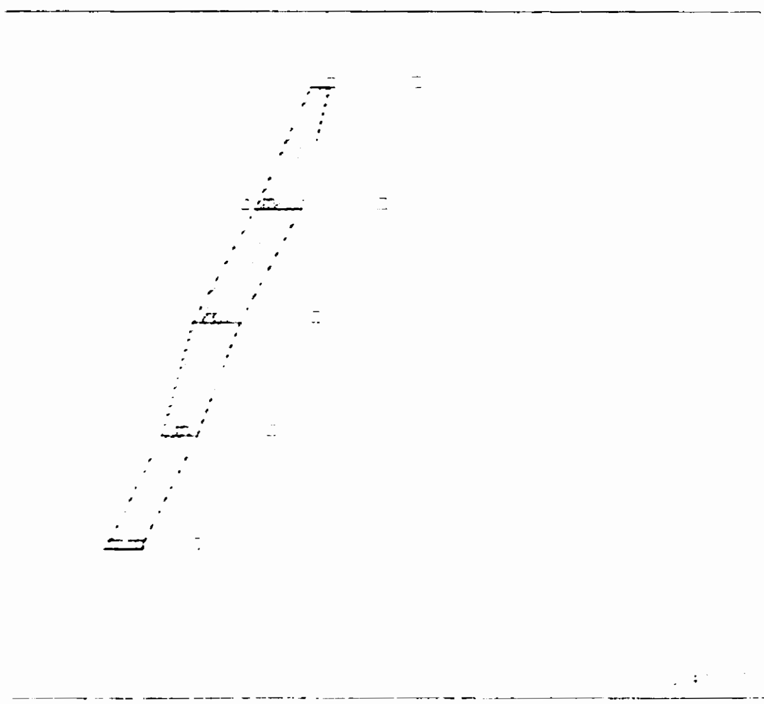

Figure 34: Lynx Brook Deep - Section 4200 east

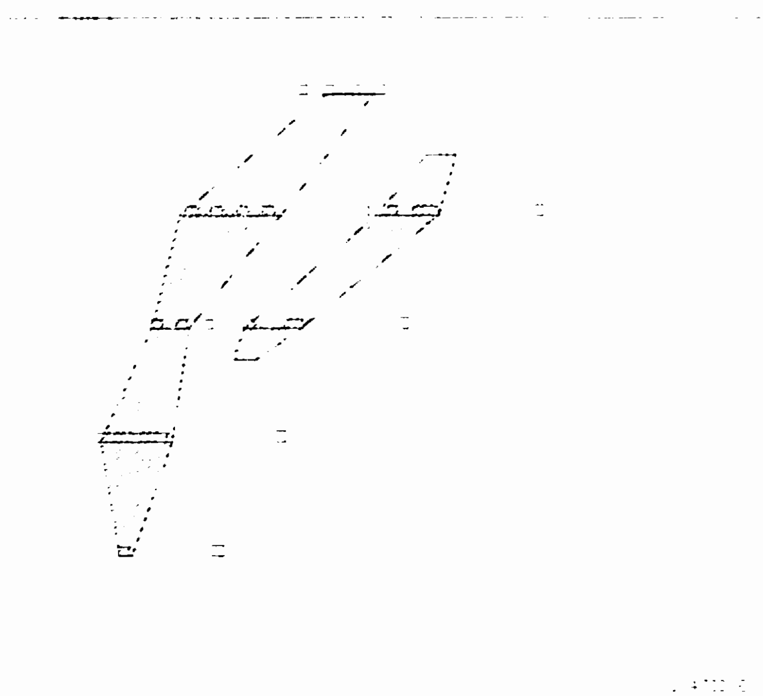

Figure 35: Lynx Brook Deep - Section 4700 east

Table 26 presents a summary of horizontal development for the inter-level spacings considered. It is clear that a reduction in the number of levels required to access and mine the ore significantly decreases mine development. Indeed, if the lift height is increased from 30.5 m to 45.7 m (a 50% increment) the number of main levels is reduced from seven to five, and mine development decreases by 30.3%. However, if the inter-level spacing is increased by an additional 15.3 m to 61.0 m, only one more level is eliminated. The corresponding reduction in mine development (1,684.3 m) is not as dramatic and represents about 50% of the previous decrease.

Table 27 shows capital expenditures in horizontal development by size of excavation for each of the alternatives investigated. The reductions in capital expenditures achieved by decreasing the number of levels are directly proportional to the respective reductions in total length of horizontal development. Indeed, total capital expenditures in horizontal development for the 61.0-metre option are 54.4% of that of the 30.5-metre alternative, whereas the corresponding figure for the 45.7-metre case is 69.7%.

For each level, mine development must be completed well before the majority of its reserves are extracted. Thus, from a cash flow viewpoint, a reduction (or increase) in the number of levels required to mine a section of the orebody has serious implications and warrants careful consideration. Operational issues, however, must be taken into account in the analysis. At Lynx Brook, 61.0-metre lift heights have been employed before. Although the resulting distance between levels would be pushing the limits of blasthole drilling equipment, the potential for savings is so great that the option must be seriously considered.

The main concern in this case would be the effect of increased inter-level spacing on dilution, ore recovery, and ground control practices. As discussed in Section 5.5.2 dilution is already a problem at Lynx Brook. Similarly, the current mining method does not facilitate full recovery of the ore reserve, even in cases in which the geometry of the orebody is favourable. Finally, increased stress levels will certainly result in higher support costs, but an adequate extraction sequence could lessen the impact on overall profitability. The impact of these three factors must be assessed before committing the deep mine to a definitive development plan.

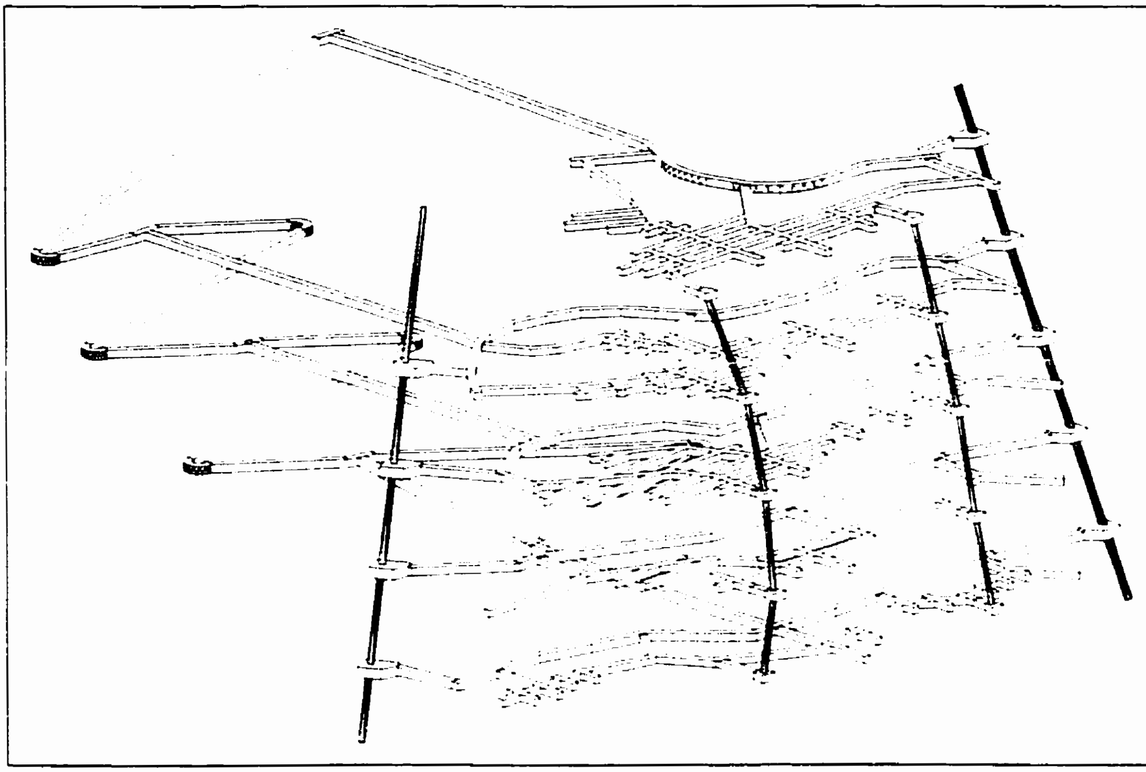

Figure 36: Lynx Brook Deep - Development between 7190 - 7800 levels, looking SW

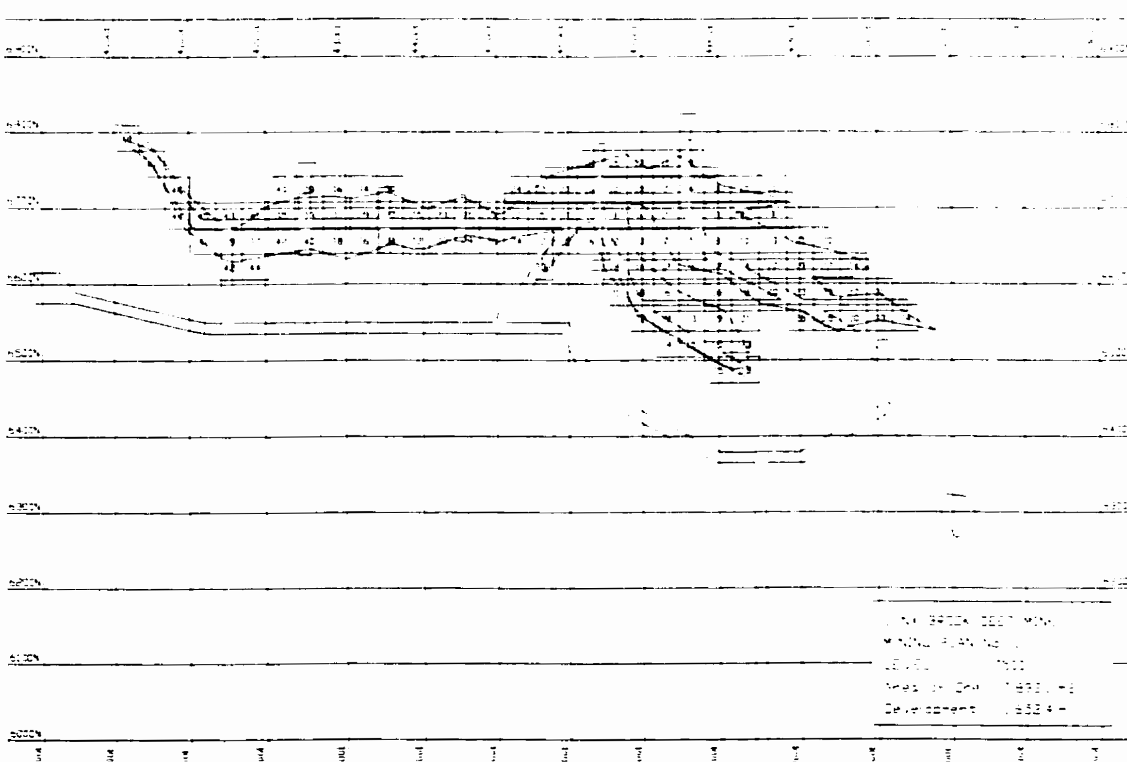

Figure 37: Lynx Brook Deep - Mining sequence for 7500 Level

Table 25. Lynx Brook Mine - Horizontal development: 45.7-metre option

Level	Horizontal Development (metres)				
	Main Level 4.2m x 4.2m	Ventilation Drift 4.2m x 4.2m	Ventilation Drift 4.2m x 3.6m	Drift/Crosscut 4.2m x 3.6m	Total
7190	415.2	30.4	48.3	1,111.4	1,605.3
7350	599.0	50.6	14.4	1,101.3	1,765.3
7500	524.3	93.4	7.9	1,026.8	1,652.4
7650	437.4	117.6	17.3	774.0	1,346.3
7800	517.4	90.1	33.4	666.4	1,307.3
Total	**2,493.2**	**382.1**	**121.4**	**4,679.9**	**7,676.6**

Table 26. Lynx Brook Deep – Summary of horizontal development

Option	Horizontal Development (metres)				
	Main Level 4.2m x 4.2m	Ventilation Drift 4.2m x 4.2m	Ventilation Drift 4.2m x 3.6m	Drift/Crosscut 4.2m x 3.6m	Total
30.5-m level spacing	3,570.7	586.6	208.8	6,654.8	11,020.9
45.7-m level spacing	2,493.2	382.1	121.4	4,679.9	7,676.6
61.0-m level spacing	1,920.3	337.1	107.6	3,627.3	5,992.3

Table 27. Lynx Brook Deep – Capital expenditures in horizontal development

Option	Horizontal Development				
	4.2m x 4.2m Drift		4.2m x 3.6m Drift		Total
	length (m)	$	length (m)	$	$
30.5-m level spacing	4,157.3	8,388,236	6,863.6	14,073,975	22,462,211
45.7-m level spacing	2,875.3	5,801,603	4,801.3	9,845,125	15,646,728
61.0-m level spacing	2,257.4	4,554,790	3,735.0	7,658,611	12,213,401

5.7 General Discussion and Conclusion

The approach to the deepening of Lynx Brook and its eventual success will depend on decisions made at corporate and operations levels. Input from senior management is critical since the deepening project is a major step toward securing the long-term survival of the operation and cannot be based on technical grounds exclusively. Options presented to top management are evaluated in the light of their financial and economic merits, metal market conditions (current prices and trends), and corporate strategic value. This case study was concerned with the strategic implications, from an operational viewpoint, of decisions made regarding issues such as ore and waste transport, mine development, and (indirectly) ventilation and air conditioning.

The analysis of the production cost structure was carried out so that more "solid" operating cost estimates could be used in the evaluation of alternatives. However, after processing the data supplied, it was found that it had not been collected with the intention of using it for decision-making or planning purposes. This prompted a more in-depth analysis that eventually determined its unsuitability for the objectives of this case study.

The lack of adequate operating cost data made it impossible to achieve all of the objectives initially established for this case study, namely the evaluation of changes of cost structure with depth. Nonetheless, the availability of a computerized orebody model and detailed cost data, coupled with *first principles* estimates developed from cost surveys, allowed a valid comparison of various approaches to developing a strategy for the extraction of the deep ore reserves. Certain assumptions had to be made, but they were controlled by available information.

5.7.1 Ore and Waste Transport

The only realistic options available for vertical muck transport at Lynx Brook are deepening the existing shaft, sinking an internal shaft, and driving a ramp for truck haulage. Based on a preliminary assessment of the current hoisting facilities it was determined that deepening the existing shaft, an apparently cost-effective proposition, would not be the best solution to the problem. The two remaining alternatives need further investigation to allow a sound final decision. It is believed that, by considering diesel trucks for ramp haulage, such an option was penalized in this study. Trolley-powered trucks not only would result in lower direct operating costs, but also would demand significantly lower ventilation and air conditioning support. This alternative becomes more attractive when it is considered that it provides the ability to phase the initial capital investment (i.e., initial number of trucks) and shorten the construction stage to less than two years. Both the internal shaft and ramp haulage options can accommodate production rate increases and further deepening.

In general, deep mines should minimize the amount of waste that is hoisted to surface. This can be achieved only by three means: by maximizing the grade of the ore extracted (i.e., reducing dilution), minimizing the amount of waste development, and disposing of the inevitable waste rock produced underground (e.g., as rock fill). The deeper areas of Lynx Brook will not require hangingwall development, thus significantly reducing the amount of development waste. On the other hand, dilution control must become a priority of the operation, not only to increase the efficiency of the system, but also to allow more time to be devoted to ore hoisting. Underground disposal of development waste requires tight scheduling, which can be achieved only with proper operations management.

Although it is not the ideal mucking system (particularly for deep mines), LHDs most probably will continue to be used at Lynx Brook Deep. Electric units must be carefully assessed. Their deployment will require mine designs and production plans that specifically address their operational limitations (limited flexibility, need to set up the power infrastructure, etc.).

5.7.2 Mine Development

Deep mining can be accomplished only by strict adherence to proven ground control procedures. In the upper levels of Lynx Brook, this meant using a single-pass extraction sequence that caused stresses to be systematically shed to the abutment. This, coupled with limited hoisting capability from the deepest loading pocket, will result in a lower extraction rate for ore below the 7200 Level than would normally be expected for a reserve of that size. A major impact on the future cost structure should be expected.

Dilution is particularly critical in highly stressed environments. Dilution control, although expensive, can be achieved through decreased inter-level spacing, smaller stopes, and smaller blasthole diameters and more efficient blasting practices. Such solutions, however, could increase development cost and time, reduce the productivity of mining units, and further restrict the production rate. There are two components in the cost of dilution. The first addresses revenues lost due to barren material displacing ore from a production system with a definite upper production limit such as Lynx Brook's, whose hoisting system is operating near to its design capacity. In 1994, lost revenues due to dilution were $ 29.0 million. The second component accounts for resources wasted handling barren material. Using re-aggregated cost data, it was determined that in 1994 this cost was about $ 5.7 million.

At any underground operation, a compromise must be reached between the number of pieces of haulage equipment and their respective rated capacity. This is a critical issue in deep mines. In fact, there are problems of waste removal; heat generation from the equipment and the rock walls; resistance created by moving ventilating air through the openings; and increased cost of providing personnel and supplies to operate the equipment. In general, larger equipment requires larger excavations, which accentuates the above-mentioned problems.[116] The tonnages and distances involved in hauling from Lynx Brook Deep to the existing dump on the 6970 Level suggest that larger units are the preferred solution.[117] This study was not able to properly evaluate the other factors involved.

5.7.3 Cost Structure

Cost data provided by the operator were aggregated according to accounting requirements and showed some inconsistencies caused probably by deficient cost-collecting methods. Thus, such data were not adequate for the evaluation of deep mining issues and potential cost changes with depth. It must be noted that the manner in which cost data were aggregated and recombined in the

[116] The only exception would be personnel requirements: for a set production rate, fewer large pieces of equipment reduce the labour force. It should be noted that even if the number of equipment operators could be drastically reduced (or even if operators could be eliminated altogether) the other problems would still subsist.

[117] Interestingly enough, such units would not be as large as the ones that are presently used at Lynx Brook.

summary reports provided could lead to a different and most probably erroneous interpretation of the actual case. However, this case study processed and recombined raw cost data from the detailed reports. The following conclusions can be drawn from the analysis of the 1994 cost structure:

1. Mine management, the largest direct mining cost item, accounted for 12.2% of total mining cost. Typically, the significance of this item is a function of the management style of the corporation. If it is considered that overheads are not included in the mine management cost, it could be argued that it is particularly high at Lynx Brook. The second largest item is upkeep and support (12.1% of mining cost). This can be partially attributed to the fact that the upper levels of the mine are undergoing pillar recovery in areas previously mined by cut and fill, demanding high services and rehabilitation costs. As mining activities become more concentrated in the deeper levels, where single pass mining methods will be employed, upkeep and support costs should decrease despite the highly stressed ground conditions. Crushing, hoisting and surface handling, and mucking and the third and fourth largest items, accounting for 11.8% and 10.9% of mining cost, respectively.

2. Labour costs accounted for 43.9% of the direct mining cost. As depicted by Figure 38, upkeep and support; crushing, hoisting and surface handling; and ore and waste mucking accounted for 53.9% of total labour cost (operating, maintenance and repair labour).[118] . Mostly through automation, existing technology makes it possible to reduce significantly the operating labour component of crushing, hoisting, surface handling, and mucking. However, a reduction/elimination of the repair labour component would be more difficult to achieve. Underground upkeep and support is very labour intensive (labour accounted for 70.0% of this cost item) and could be better addressed by optimizing operational practices such as mining sequencing and ground control.

3. Overheads, depreciation and surface ore transport accounted for 28.6% of the total production cost (ore delivered to the processing facility). If mine management were added to this figure, it would become 37.3% of total production cost.

4. Mine development costs represented 7.8% of the mining cost. The unit costs of development (in dollars per tonne of ore) will likely increase with depth as the orebody narrows and level spacing is decreased. The downward trend in ore reserves and horizontal

[118] It is interesting to note that, after mine management, these are the three largest mining cost items (see Figure 22).

development with depth can be better appreciated in Figure 39, which plots such figures against level elevation. Indeed, reserves dropped from 1.34 million tonnes at 7650 Level to 1.00 million tonnes at 7800 Level, but horizontal development was virtually constant at 1,300 metres in each case. This could be overcome by altering the mining sequence, stope geometry, and reducing hangingwall development.

5. The cost of ore and waste handling to surface, (crushing, hoisting and surface handling, exclusive of the ore and waste mucking by LHD units) was 11.8% of direct mining cost. A portion of the electrical energy cost would have to be added to obtain a total figure. This item will increase with depth unless an alternative technology is adopted. Furthermore, the direct unit operating cost of skip hoisting will increase as the depth of mining increases, and as the throughput of the existing system decreases and fixed shaft maintenance costs come into play.

5.7.4 Conclusion

The future of Lynx Brook depends on its deep ore reserves. Due to their high grade, they will remain economic, even under the current depressed base metal markets. In fact, the 1994 total unit production cost ($84.6/tonne) represents 43.6% of the unit value of recoverable metal in the deep ore reserves ($193.8/tonne, using average grades and June 1998 metal prices).

The economic benefits of developing an integrated strategic plan that includes ground control practices, mining sequences and layouts, and innovative technology, particularly in the areas of dilution control, mine development, and vertical and horizontal muck transport, are very high. The assessment of such a plan can benefit greatly from the experience and information gained during the exploitation of the upper levels. This has the potential of effectively reducing the uncertainty and risk typically associated with this type of projects. Cost-collecting methods will have to be improved significantly so that changes in cost structure and operational practices can be properly evaluated using site-specific data.

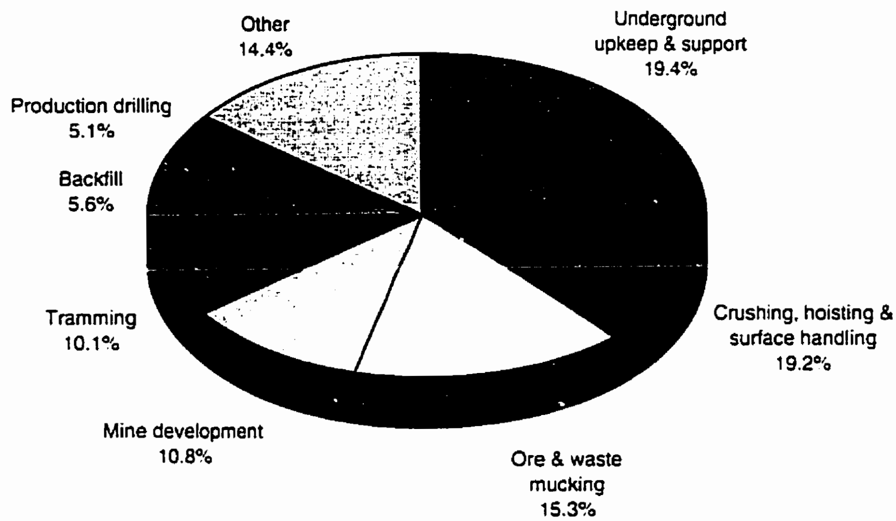

Figure 38: Lynx Brook Deep – Major components of labour cost

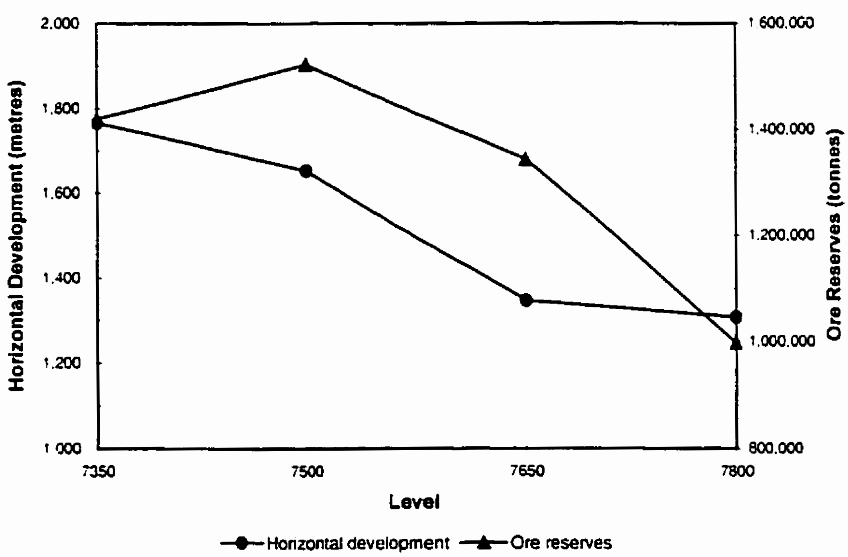

Figure 39: Lynx Brook Deep - Horizontal development and ore reserves vs. elevation

6. Case Study 2: Fox River Project

Due to insufficient data, the case study presented in Chapter 5 did not allow the analysis of several important deep underground hard-rock mining issues. Most importantly, the information provided by the operations[119] did not lend itself to the study of:

1. the impact of the use of smaller equipment and smaller excavations for mine development and mine extraction: and,
2. changes of cost structure with increased depth of mining.

Since such analyses are crucial to the subject of this research project, it was decided to create an artificial case study with the specific purpose of carrying them out. The *Fox River Project* was assembled using an artificial (i.e., computer-generated) orebody model, and capital, operating, and labour cost data obtained from cost-estimating handbooks and mine cost surveys. Data from the previous *real* case study were extensively used to check the validity of the assumptions made. This was particularly true for labour and supplies costs, as well as for mine equipment selection.

6.1 Objectives and Scope

Using concepts discussed in previous chapters, this case study will aim at analyzing the mine development issues of a new, deep, base metal operation. The importance of the other four factors affecting deep mining will be acknowledged by addressing their relationships and interactions with mine development. Moreover, the operational and strategic benefits (or lack of thereof) of a solution to a particular development issue will not be measured and assessed in isolation, but based on the effects it has on the remaining factors. The specific objective of this case study is twofold:

1. To investigate the impact of the use of various sizes of equipment ***and*** various sizes of development openings on:
   a. mine development;

[119] Cost information supplied was already aggregated when it was first entered in the respective systems. This made it impossible to break it down according to specific working areas (i.e., levels and stopes), pieces of equipment or, more importantly, activities (stoping, mine development, backfill, etc.).

b. labour requirements;

c. ventilation requirements;

d. waste rock production;

e. flexibility of the operation; and,

f. economics of the operation in terms of dilution and dilution control; ore recovery; and overall efficiency (principally use of labour and equipment).

2. To study changes to the project's cost structure as depth of mining increases.

On the other hand, it should be noted that this case study is ***not*** concerned with issues such as:

- mining method selection;
- mine sequencing;
- short-term planning;
- detailed mine design; and,
- specific ground control problems.

6.2 Methodology

Scenario analysis was used to investigate the relative significance, from operational and economic viewpoints, of various equipment configurations and development opening sizes.

6.2.1 Scenario Analysis

Scenario analysis is a sound approach to considering and assessing future courses of action. This technique has been used as a forecasting tool for long-term planning (Pearce and Robinson, 1991, pp. 154–159); as part of comprehensive sensitivity analyses that involve interrelated variables (Ross et al., 1993, p. 231; Fraser et al., 1997, pp. 371–378); and for the evaluation of nuclear fuel waste disposal alternatives (Goodwin et al., 1994), and new technologies for natural gas conversion (Szladow and Fernet, 1990). Kahn and Wiener (1967, p. 6), noted that scenarios are

> "*...especially valuable in the study and evaluation of the interaction of complex and/or uncertain factors. Scenarios are hypothetical sequences of events constructed for the purpose of focusing attention on causal processes and decision points. They answer two*

kinds of questions: (1) Precisely how might some hypothetical situation come about, step by step? and (2) What alternatives exist, for each actor (sic) *at each step, for preventing, diverting, or facilitating the process?"*

The main advantages of scenario analysis include:

- The factors and variables that determine the outcome for a given condition, as well as the mechanisms that have the greatest effect in deciding the results, constitute the focus of the analysis.
- Information that cannot be quantified easily is incorporated into the evaluation process.
- The outcome of different scenarios can be assessed in a relative way.
- If required, the process can be carried out iteratively, with *learning scenarios* leading to more refined/focused scenarios that investigate in depth the salient aspects identified during the first phase.

In this case study, a scenario defines a possible future exploitation of the Fox River orebody, and includes a set of critical mine development factors that determine its operational and economic feasibility. Such factors are the cross-sectional dimensions of major mobile mining equipment, the thickness of the orebody, and the vertical distance between contiguous levels. Scenario analysis provides a logical framework that facilitates the comprehensive evaluation of the relative significance of each factor, as well as the impact on the issues identified in Section 6.1.

It must be noted that this case study is not concerned with scenario analysis as used for risk assessment in project evaluation (Torries, 1998, pp. 55–59).[120] Indeed, the scenarios were built with the specific purpose of studying mine development alternatives. They were not based on any predetermined assumption regarding their profitability and/or operational advantage. Rather than using meaningless probabilistic estimations, the relationships between the various design, development, and planning parameters were determined by careful analysis of the independent conditions of each scenario.

[120] Scenario analysis in project evaluation consists of developing a *base case* using so-called *"best"* estimates of the main project's parameters. Then, some assumptions are made regarding the possible performance of such parameters, and *"optimistic"* and *"pessimistic"* scenarios are built. The former assumes that all the parameters improve, whereas the latter is based on their deterioration. As correctly pointed out by Torries (1998, p. 58), the use of this technique without adequate understanding of its shortcomings is dangerous.

6.2.2 Structure of the Case Study

The Fox River case study was developed through seven distinct phases:

1. Definition of mining scenarios.
2. General assumptions.
3. Construction of mining scenarios.
4. Cost data compilation and organization.
5. Investment program.
6. Determination of cash flows.
7. Analysis of the results.

The iterative nature of the scenario development process is graphically illustrated by Figure 40.

6.3 Definition of Mining Scenarios

The main scenarios were defined according to the choice of mobile mine development and ore extraction equipment. Based on the information provided by the world's two largest manufacturers of underground mining equipment,[121] small, mid-size, and large equipment scenarios were established. As shown in Table 28, each scenario was distinguished by the rated capacity[122] of LHDs and low profile mine trucks.[123] Table 29, Table 30, and Table 31 show the main specifications of the LHDs and mine trucks, long-hole drills, and jumbo drills, respectively, that were taken into account in each scenario.

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[121] The North American representatives of both Atlas Copco/Wagner and Tamrock were contacted to obtain updated technical specifications and list prices for their trackless equipment. Both companies responded promptly, but Tamrock supplied the most complete set of technical and operational data covering its entire product line of underground mining equipment. Thus, in order to ensure consistency and simplify the equipment selection process, it was decided to use exclusively Tamrock data for this case study.

[122] It should be noted that the determining characteristic of a LHD is its payload capacity (in tonnes), although most people are used to identifying them according to bucket capacity. The volume of the standard bucket is provided for reference purposes.

[123] Other pieces of equipment are also different in each main mining scenario. However, the dimensions and ventilation requirements of mobile mine development and ore extraction equipment establish the minimum dimensions of access and ore transport excavations (see Sections 4.3 and 4.4). Therefore, they are the determining parameters for each scenario.

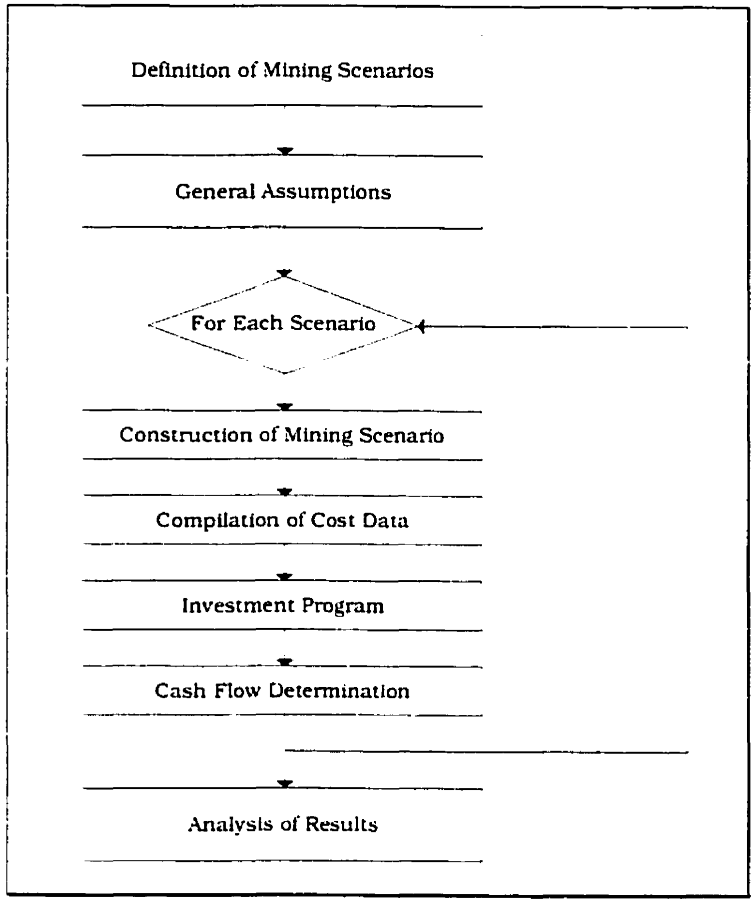

Figure 40: Fox River Project - Stages in the development of the case study

Within each main scenario, it was deemed important to investigate the operational and economic performance of the three sets of equipment under different orebody geometries.[124] Thus, each of them considers three orebody thicknesses: 10.0 m, 15.0 m, and 20.0 m. While a 10.0-m thick vein cannot be considered as *thin*, 20.0-m thick ore is not unusual in Canadian underground mines. In fact, of the 39 precious and base metal underground operations that provided orebody data to a 1996 Canadian mines survey, 17 exploited orebodies that were 20.0 m wide or wider (Scales, 1996). The average thickness of the 19 orebodies that were 10.0 m thick or thicker was 48.0 m, whereas the average thickness of orebodies that were less than 10.0 m wide was 4.7 m.

[124] The importance of the orebody geometry on the equipment selection process cannot be overemphasized. For instance, before bulk mining methods became popular, the thickness of the orebody dictated the maximum width of captive mucking and drilling equipment. As noted by Pelley (1994), larger equipment is used in bulk mines that exploit thick deposits in order to take advantage of increased productive capacity of single stopes and mining blocks. This case study will try to determine, under the particular circumstances of the Fox River project, the validity and applicability of such common practices.

Table 28: Fox River Project – Main mining scenarios

Scenario	Rated Capacity		
	LHD		Truck
	tonnes	m^3	tonnes
Small Equipment	6.2	3.0	20.0
Mid-Size Equipment	9.6	4.5	30.0
Large Equipment	15.0	7.0	50.0

Table 29: Fox River Project – Specifications of development and extraction equipment

Scenario	Type	Manufacturer	Model	Tramming Capacity	Volume		Main Dimensions			Turning Radius		Engine		
					Small	Large	Length	Width	Height	Inner	Outer	Manufacturer	Type	Output
				tonnes	m^3	m^3	mm	mm	mm	mm	mm			kW
Small Equipment	LHD	Tamrock	Toro 301DL	6.200	2.70	3.30	8,510	2,100	2,150	3,030	5,800	Deutz	F6L 413 FW	102.0
	Truck	Tamrock	EJC 20	20.000	10.00	11.50	8,382	2,438	2,438	4,318	7,696	Detroit Diesel	Series 50 DDEC	205.0
Mid-Size Equipment	LHD	Tamrock	Toro 400D	9.600	3.80	4.60	9,363	2,480	2,370	3,550	6,635	Deutz	F10L 413 FW	158.0
	Truck	Tamrock	EJC 430D	27.273	10.70	15.30	9,576	2,896	2,743	4,343	8,382	Deutz	F12L 413 FW	207.0
Large Equipment	LHD	Tamrock	Toro 650D	15.000	5.40	8.20	10,938	3,000	2,640	3,574	7,205	Detroit Diesel	Series 60-11.11	243.0
	Truck	Tamrock	Toro 50D	50.000	16.50	25.50	10,036	3,220	2,640	4,859	9,041	Detroit Diesel	Series 60 DDEC	354.0

Table 30: Fox River Project – Specifications of long-hole drills

Scenario	Manufacturer / Model	Drilling Capability		Driving Dimensions			Turning Radius		Power Requirements		Hole Size		Minimum Drift Size	
		Up	Down	Length	Width	Height	Inner	Outer	Main Motor	Total	Min.	Max.	Width	Height
		m	m	mm	mm	mm	mm	mm	kW	kW	mm	mm	mm	mm
Small Equipment	Tamrock / Solo 606 RA	30.0	40.0	5,050	2,300	2,750	1,000	4,250	55.0	78.0	64	89	3,600	3,600
Mid-Size Equipment	Tamrock / Solo 620 RA/C	30.0	40.0	8,700	2,240	2,640	3,100	5,900	55.0	78.0	64	89	3,650	3,650
Large Equipment	Tamrock / Solo 1020 RA/F	30.0	50.0	9,950	2,240	3,100	3,100	6,300	75.0	102.0	89	127	4,000	4,000

Table 31: Fox River Project – Specifications of jumbo drills

Scenario	Manufacturer / Model	Booms	Drilling Capability		Main Dimensions[125]			Turning Radius		Drifter	Total Power	Hole Diameter		Feed Length	Hole Length
			Width	Height	Length	Width	Height	Inner	Outer			Min.	Max.		
			m	m	mm	mm	mm	mm	mm		kW	mm	mm	mm	mm
Small Equipment	Tamrock / Monomatic H105D	1	7.20	5.69	12,050	1,700	2,850	3,350	6,150	HL 500S	60	45	57	5,995	3,905
Mid-Size Equipment	Tamrock / Minimatic H205D	2	8.70	6.00	12,840	1,980	3,200	3,250	6,300	HL 500S	108	45	57	6,605	4,515
Large Equipment	Tamrock / Mini 206D	2	9.26	6.30	13,630	1,980	3,200	3,250	6,400	HL 550S	133	45	57	7,100	5,270

[125] Total length with selected feed.

Having defined the equipment type and orebody geometry, the next step was to establish a factor that could allow the analysis of mine development, mine planning, operational flexibility, and dilution issues. This was achieved by taking into account various inter-level spacings for each combination of equipment size and orebody thickness. Three inter-level spacings were considered: 25.0-m, 50.0-m, and 75.0-m intervals. Although 25.0-m and 50.0-m lift heights are within the normal range of bulk underground operations in Canada, a 75.0-m lift height would be pushing the limits of standard production drilling equipment. Indeed, in 1995 17 Canadian precious and base metal underground mines utilized stopes that were 50.0 m high or higher, with an average stope height of 68.0 metres. In that year, there were 17 underground operations that mined shorter stopes with an average height of 29.4 m (Scales, 1996). The 27 mining scenarios that this case study comprises are schematically shown in Figure 41.

6.4 General Assumptions

The analyses carried out in this case study involved a degree of detail typical of mine feasibility studies.[126] Thus, a high level of confidence on the data available was assumed. Given the overall objective of the study, areas such as equipment selection, excavation design, and production planning were dealt with in more detail during the construction of the scenarios. Similarly, general assumptions were made to facilitate meaningful comparisons and analyses, and focus on relevant issues. They were applied to every scenario, regardless of equipment size, orebody thickness, and inter-level spacing, and covered the following areas:

- Location of the project.
- *In situ* ore reserves.
- Geological characteristics of the orebody.
- Geotechnical considerations.
- Main access and ore and waste extraction method.
- Mining method.

[126] At the feasibility stage, the mining method, mine layout, and the selection of major mining equipment are optimized, and a preliminary geotechnical assessment has been completed. Similarly, capital costs are estimated to within ± 10%, operating costs are based on mine plans, and detailed discounted cash flow analysis is required. The estimates obtained in such a way can be used for budgeting purposes (Smith, 1994).

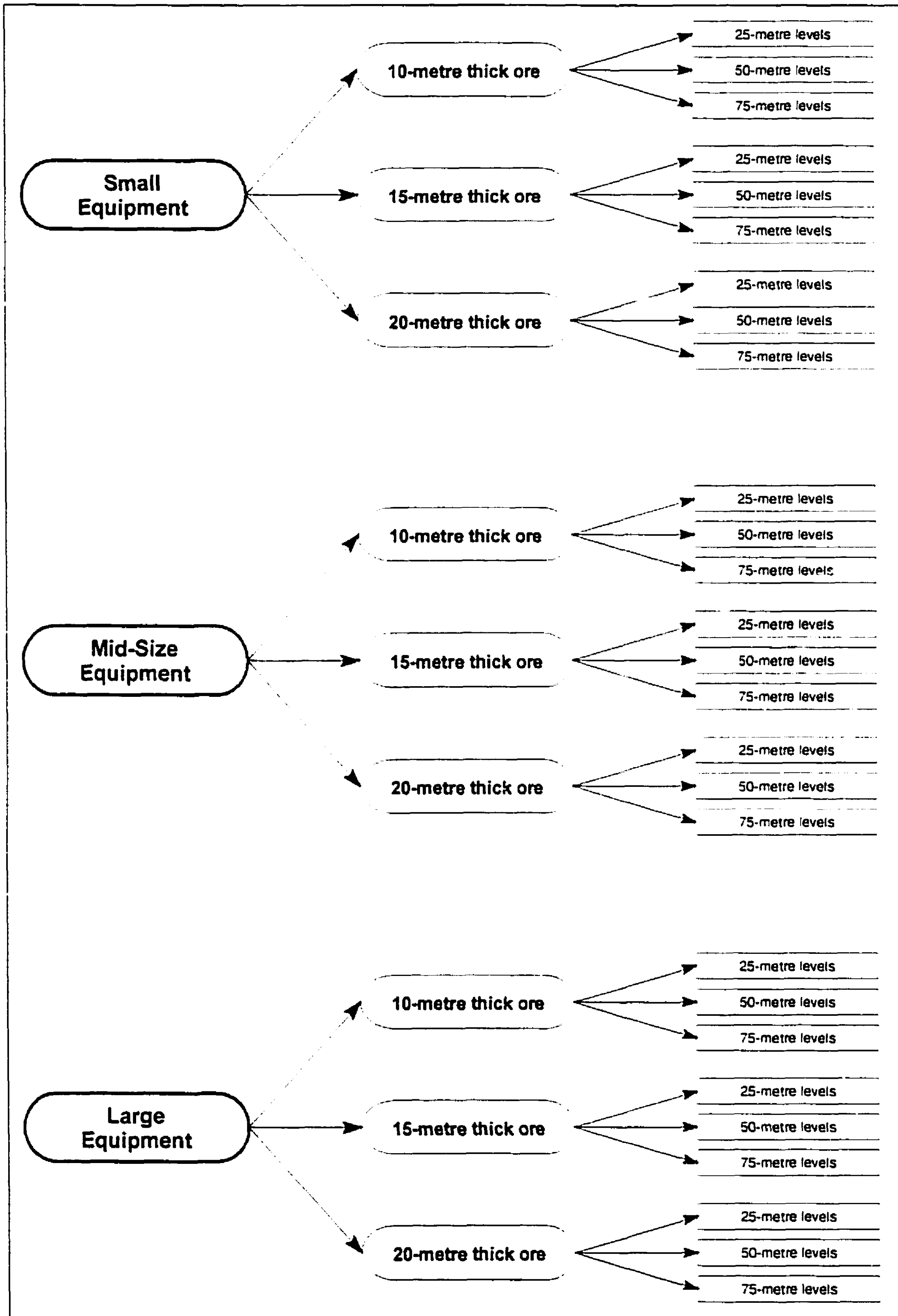

Figure 41: Fox River Project - Mining scenarios

6.4.1 Location of the Project

Fox River is a deep base metal ore deposit located in Northern Ontario. Based on geological and geotechnical data obtained through a diamond drilling exploration campaign, a decision has been made to develop an underground mine. Sulphide ore will be supplied to an existing concentrator plant located within 2.0 km of the project site.

6.4.2 Ore Reserves

Total geological *in situ* reserves are 12.0 million tonnes of base metal ore with a unit value of US $ 100.00/tonne.[127] The specific gravity of undiluted ore and barren wall rock are assumed to be 4.0 and 3.0, respectively. The *minable reserves* (tonnage and grade) are different for each mining scenario. This is because dilution and ore recovery, the two determining factors of tonnage and grade, vary according to equipment size, orebody thickness, and inter-level spacing.

6.4.3 Geological Features of the Orebody

Assumptions are related to orebody geometry, geological continuity, and grade distribution.

- ***Orebody geometry***

  The orebody is tabular (i.e., two-dimensional or vein-type) and vertically dipping. There are no changes in either orebody dip or strike. The strike length is fixed at 1,000 metres, which allows the study of the impact of increased haulage distances on cost and productivity. The bottom of the deposit is located 2,000 metres below surface. The volume of the orebody (and tonnage of geological reserves) is kept constant for all mining scenarios. Therefore, the overall height of the orebody changes according to orebody thickness: 300 m for 10-m wide ore; 200 m for 15-m wide ore; and 150 m for 20-m wide ore.

[127] Using June 1998 metal prices (copper: US $ 0.75/lb; zinc: US $ 0.50/lb; lead: US $ 0.25/lb), this figure is, for instance, equivalent to that of ore grading 2.0% Cu, 4.0% Zn, and 4.0% Pb. As a comparison, and employing the same metal prices, seventeen of the thirty-six underground operations that responded to a 1996 mine survey reported ore with an *in situ* value of US $ 100.00/tonne or greater (Scales, 1996).

- ***Geological continuity***

  It is assumed that the continuity of the mineralization is not affected by faults or any other structural feature.

- ***Grade distribution***

  The ore is uniformly distributed over the entire orebody. Ore grade does not change along the strike of the deposit or with depth. The hangingwall and footwall contacts are well defined. Both walls are barren and, thus, dilution from the rock walls does not contribute to the total metal content of extracted ore.

6.4.4 Geotechnical Considerations

This case study is not concerned with the details of the geotechnical design of the underground operation. However, it is pertinent to indicate that, regardless of orebody thickness and inter-level distance, average support requirements for permanent and temporary openings were assumed.

- ***Main access and extraction routes***

  The excavations will be supported with rockbolts, wire mesh, and shotcrete 100% of the time.

- ***Secondary access and extraction routes***

  The excavations will be supported with rockbolts and wire mesh. Shotcrete will be used 50% of the time.

- ***Stopes***

  All stopes will be backfilled immediately after extracting the muck and recovering the upper wedges. Stope brow plugs will be poured using a tailings-to-cement ratio of ten to one (10:1). The remainder of the stope will be filled using a 30:1 ratio. It has been assumed that cablebolts will not be needed to support the walls of the stopes. Higher stopes will be penalised with increased dilution.

- ***Stope Brows***

  Stope brows are supported with mesh, rockbolts, and shotcrete. Crosscuts leading to drawpoints are similarly supported for a distance of 10 m from the stope brow.

6.4.5 Access and Extraction Method

With the bottom of the mineralized zone located 2,000 metres below surface, the only valid option for access and ore/waste extraction is through a shaft. The main shaft will be located in the footwall, at a prudent distance from the orebody. Given the size of the ore reserves, Fox River will result in a medium- to high-tonnage operation. Therefore, two hoisting systems will be installed in the shaft: one to handle exclusively the muck (ore and waste) and one for mine services (i.e., to hoist personnel, supplies, and materials).

A main ramp will be driven from the uppermost level to the main haulage level at the bottom of the mineralized area. It will be used to transport personnel, materials, and supplies between levels and into the stopes and/or working areas.[128] An auxiliary ramp will be driven from the haulage level to the loading pocket level in order to provide access to the crushing and muck-handling facilities at the bottom of the shaft. The main haulage level, located 25 metres below the deepest main level, will be used to tram ore and waste from the ore/waste passes to the shaft.

Three main ore/waste passes will be raise-bored at regular intervals along the strike of the orebody.[129] Access to every working level will be provided by crosscuts. The bottom of the vertical orepasses will be located at the elevation of the main haulage level.

The ventilation requirements of every scenario will be met with the aid of two dedicated ventilation shafts: one for fresh-air and one for return air. They are to be sunk to the top and bottom of the active mining area, respectively.

6.4.6 Mining Method

The orebody will be mined using a modified version of the *sublevel stoping* mining method (also known as *longhole stoping*, see Haycocks and Aelick, 1992). The details of the mining method

[128] In spite of the resulting increased development cost and time, this arrangement can be considered as standard in modern underground mines, regardless of their depth, number of working levels, and production rate. For instance, the final design of Barrick's Meikle Mine in Nevada included a ramp to be driven from the 925-foot level to the bottom of the production shaft, located at a depth of 1900 feet (White and Kral, 1994).

[129] It could be argued that fewer orepasses (maybe two) would have been adequate for the project's haulage requirements. This issue could be further investigated in a second round of analyses.

are not relevant to the purposes of this case study. Nonetheless, it is important to outline the main steps of the ore-production cycle, since they affect both productivity and mining cost, and determine level development and layout as well as the extraction sequence.

- ***Mine development***

  Main levels are driven in the footwall at intervais that vary according to lift height. Crosscuts connect every level to the main ramp and each of the ore/waste passes. Depending on lift height and orebody width, some of the levels are connected to the main shaft. To provide direct access to the orebody and create the drawpoints, a number of crosscuts are driven at regular intervals from the main levels to the ore/hangingwall contact. The distance between the longitudinal axes of two consecutive crosscuts/drawpoints is 25.0 m for 10.0-m thick ore, and 20.0 m for 15.0-m and 20.0-m thick ore.

  At each level elevation, an in-ore sublevel or undercut, centred on the mid-plane of the orebody, is driven along the orebody strike. Once the top and bottom sublevels of an ore block or stope have been developed, the next step is to create the initial free face for production blasting. This is achieved by excavating a drop-raise in the middle of the stope and then using it to open a four-metre deep slot to the full height and width of the block.

  Before starting production activities in a particular level, it must have been previously connected to both ventilation shafts.

- ***Production drilling***

  Electric-hydraulic, top-hammer, longhole-drilling machines are used for production drilling. The diameter of the blastholes varies according to equipment size (see Table 30). Up-holes and down-holes are drilled in ring patterns. This eliminates the need to develop the drilling sill to the full width of the orebody, which not only extends the development stage, but also may create ground control problems.

- ***Blasting***

  Blasting is carried out with standard Anfo, since it is assumed that no significant water problems exist at the mine. The powder factor[130] (in kg of Anfo per tonne of rock) varies with blasthole diameter, which, in turn, is a function of equipment size. The powder factors for the small, mid-size, and large equipment scenarios are, respectively, 0.40 kg/tonne (0.8 lb/ton), 0.50 kg/tonne (1.0 lb/ton), and 0.60 kg/tonne (1.2 lb/ton). In other words, when compared to the mid-size equipment case, the explosive consumption rates of the small and large equipment scenarios are 20% lower and 20% higher, respectively.

- ***Mucking***

  LHD machines are used to muck broken ore from the drawpoints and transport it to the nearest orepass.

- ***Backfill***

  As discussed in Section 6.4.4, cemented backfill is used to support the stopes.

- ***Mining sequence***

  A top down, single lift, single-pass mining sequence, similar to those discussed in Section 3.3.3, is followed. The use of such a sequence is expected to benefit the operation by:

  - eliminating the creation of an ever-decreasing sill pillar, typical of bottom-up sequences, prone to rockbursts and other ground control problems;
  - reducing the need to develop and maintain several levels simultaneously; and,
  - increasing ore recovery, and minimizing ground control problems and maintenance costs associated with multi-pass sequences.

Each stope or panel will extend the full width of the orebody, thus eliminating the need to develop a transverse component in the mining sequence. Typically, the sequence in any level will start at the centre of the deposit and expand towards its extremities, effectively shedding the stresses to the abutments.

[130] The powder factor is a standard method of expressing the explosive consumption required to achieve the desired rock fragmentation. Fragmentation and particle size distribution are functions of blasthole diameter and burden (Aimone, 1992). Therefore, different powder factors will be needed in order to obtain the same degree of fragmentation and particle size distribution with varying hole diameters and drilling patterns. While the specific changes in powder factor will depend on rock strength and other geological and geotechnical characteristics of the rock mass, low powder factors will typically result in coarser fragmentation and vice versa. In general, a lower powder factor will be required to achieve the desired results if smaller holes drilled in a tighter pattern are used. This is due to better distribution of the explosive charge and increased blasting efficiency.

6.5 Construction of Mining Scenarios

An iterative approach was required for the construction of the 27 mining scenarios defined in Section 6.3. In general, scenario development progressed through the following stages:

a. Sizing of permanent and temporary excavations.

b. Mine layout design.

c. Estimation of minable reserves.

d. Preparation of mine development and production programs.

6.5.1 Permanent and Temporary Excavation Sizing

The main dimensions of permanent (main ramp, levels, ventilation shafts, etc.) and temporary (sublevels, crosscuts, draw-points, etc.) excavations were determined based on a preliminary estimate of ventilation requirements and mobile extraction equipment dimensions. The quantity of air needed to ventilate trackless mines is established essentially by the power of the engines of the diesel units. Thus, the preliminary assessment of ventilation requirements requires making initial assumptions regarding the ore production rate and, indirectly, speed of development.

Based on 12.0 million tonnes of *in situ* reserves, it could be assumed that the minimum and maximum yearly production rates would be 1.0 million tonnes (2,800 tonnes/day) and 1.5 million tonnes (4,200 tonnes/day), respectively. Then, if a single LHD could muck 700 tonnes/day from a stope, 2,800/700 = 4 LHDs would be required in the former case, whereas the latter would need 4,200/700 = 6 units. Similarly, a development crew would use one LHD and two trucks. Two such crews would be necessary when 2,800 tonnes/day are produced and three when mining at the higher rate. As noted by Hartman et al. (1997, p. 524), about 0.0949 $m^3/s \cdot kW$ (150 cfm/bhp) are needed to dilute the fumes produced by diesel equipment. Table 32 shows the ventilation requirements of major diesel equipment for the two production rates considered. It is reasonable to assume that diesel equipment accounts for 70% of total air demand and that about 30% of the air never reach its intended destinations (Hartman et al., 1997, p. 529; Bise, 1986, p. 23). Thus, total air requirements would be 761,000 cfm (@ 2,800 tonnes/day) and 1,141,000 cfm (@ 4,200 tonnes/day).

Table 32: Fox River Project – Air requirements for mobile diesel equipment

Production Rate		Production Crews			Development Crews						Total	
tonnes/year	tonnes/day	# of LHDs	Engine kW	Air m^3/s	# of trucks	Engine kW	Air m^3/s	# of LHDs	Engine kW	Air m^3/s	m^3/s	cfm
1,000,000	2,800	2	209	34.7	2	261	43.4	1	209	9.9	175.9	372,750
		2	209	34.7	2	261	43.4	1	209	9.9		
		4		69.4	4		86.7	2		19.8		
1,500,000	4,200	2	209	34.7	2	261	43.4	1	209	9.9	263.9	559,125
		2	209	34.7	2	261	43.4	1	209	9.9		
		2	209	34.7	2	261	43.4	1	209	9.9		
		6		104.1	6		130.1	3		29.7		

The procedure proposed by Hartman et al. (1997, pp. 437-444) was followed to produce a preliminary estimate of the diameter of the main ventilation shafts. The average of the previously determined minimum and maximum air requirements was employed:

- Air quantity = (761,000 cfm + 1,141,000 cfm)/2 = 951,000 cfm ≈ 1,000,000 cfm

Table 33 presents the main parameters used for the determination of the optimum diameter of the ventilation shafts.[131] The results are graphically shown in Figure 42. It can be seen that the total cost curve is flat for diameters ranging from about 5.5 to 6.0 metres. This indicates that there would not be a critical difference in total ventilation cost if shaft diameters within such a range were used. The higher end of the range (i.e., 6.0 metres) was selected, so that air requirement increases could be easily accommodated without having to incur in higher operating expenses.

The specifications of development openings were determined after considering ventilation needs, the dimensions of major mobile extraction equipment (see Table 29, Table 30, and Table 31), and Ontario regulations (Ontario, 1996, pp. R62-R63). Table 34 shows three sets of specifications (for small, mid-size, and large equipment scenarios), together with equipment dimensions, and minimum opening dimensions. Due to current trends in equipment design, excavations of the same type (i.e., ramps, levels, etc.) resulted in having similar heights (differences of 20 to 30 cm) but very different widths (differences of up to 80-90 cm).

[131] As noted in Section 6.4.5, two ventilation shafts would be needed: a fresh-air shaft that connects to the top of the active mining area, and a return-air shaft that reaches the main haulage at the bottom of the mineralized area. Since similar air quantities are expected to flow through both ventilation shafts (the production shaft would be neutral), they would have to have similar diameters. The calculations were carried out using return-air shaft data.

Table 33: Fox River Project – Parameters for determination of ventilation shaft diameter

Shaft depth	2100 metres
Operating life	15 years
Friction factor	2.00E-09 $lb \cdot min^2/ft^4$
Fan efficiency	65.0 %
Energy cost	0.066 U.S. $/kWh
Interest rate (for annual capital cost calculation)	10.0 %
Taxes, insurance, and maintenance	3.0 %

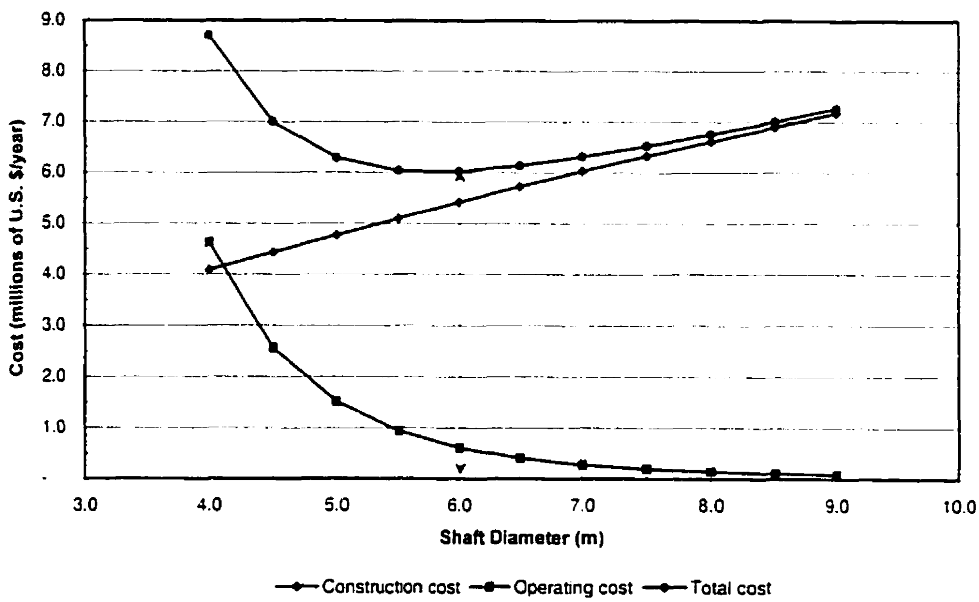

Figure 42: Fox River Project – Determination of optimum ventilation shaft diameter

As in the case of the ventilation shafts, the production shaft and ore/waste passes will have the same specifications, regardless of mining scenario. For the production shaft, the depth of the orebody, coupled with expected production rates and mobile equipment dimensions (which have to be brought into the working levels through the shaft), made it necessary to specify a 6.0-metre diameter opening. Similarly, 3.0-m diameter ore and waste passes will be used.

Table 34: Fox River Project – Basic specifications of development openings

Equipment Size	Excavation	Equipment Dimensions*		Minimum Dimensions**		Design Dimensions		Area	Gradient
		Width (m)	Height (m)	Width (m)	Height (m)	Width (m)	Height (m)	(m^2)	(%)
Small	Main Ramp	2.44	2.44	4.00	3.00	4.50	4.00	18.00	15.00
	Auxiliary Ramp	2.44	2.44	4.00	3.00	4.00	3.60	14.40	15.00
	Main Level	2.44	2.44	4.00	3.00	4.50	4.00	18.00	0.50
	Sublevel	2.10	2.15	3.60	2.70	3.60	3.60	12.96	0.50
	Crosscut	2.10	2.15	3.60	2.70	3.60	3.20	11.52	0.50
Mid-Size	Main Ramp	2.90	2.74	4.40	3.30	4.90	4.30	21.07	15.00
	Auxiliary Ramp	2.90	2.74	4.40	3.30	4.40	3.80	16.72	15.00
	Main Level	2.90	2.74	4.40	3.30	4.90	4.30	21.07	0.50
	Sublevel	2.48	2.37	4.00	2.90	4.00	3.60	14.40	0.50
	Crosscut	2.48	2.37	4.00	2.90	4.00	3.40	13.60	0.50
Large	Main Ramp	3.22	2.64	4.80	3.20	5.30	4.20	22.26	15.00
	Auxiliary Ramp	3.22	2.64	4.80	3.20	4.80	4.00	19.20	15.00
	Main Level	3.22	2.64	4.80	3.20	5.30	4.20	22.26	0.50
	Sublevel	3.00	2.64	4.50	3.20	4.50	4.00	18.00	0.50
	Crosscut	3.00	2.64	4.50	3.20	4.50	3.70	16.65	0.50

* Maximum dimensions of trackless equipment to be used in the excavations.

** Minimum dimensions according to *Ontario Regulations for Mines and Mining Plants*.

6.5.2 Mine Layout Design

The basic mine layout of every scenario was constructed according to the access, extraction, and mining methods described in Sections 6.4.5 and 6.4.6, as well as using the previously determined opening dimensions (see Section 6.5.1).

Actual three-dimensional mine layouts were established with the aid of the AutoCAD program. In addition to compatibility, portability, and ease of use, the main advantage of the use of AutoCAD for mine layout design is the ability to use true three-dimensional objects. Such objects (i.e., stopes, levels, ramps, etc.) can then be joined, intersected, and sectioned using planes of arbitrary orientation, greatly simplifying the task of building the mine models. Similarly, the objects' corresponding volumes and other data are automatically computed by the program.

Figure 43, Figure 44, and Figure 45 show perspective views of three large-equipment models built for 15-metre ore and 25-metre, 50-metre, and 75-metre inter-level spacings, respectively.

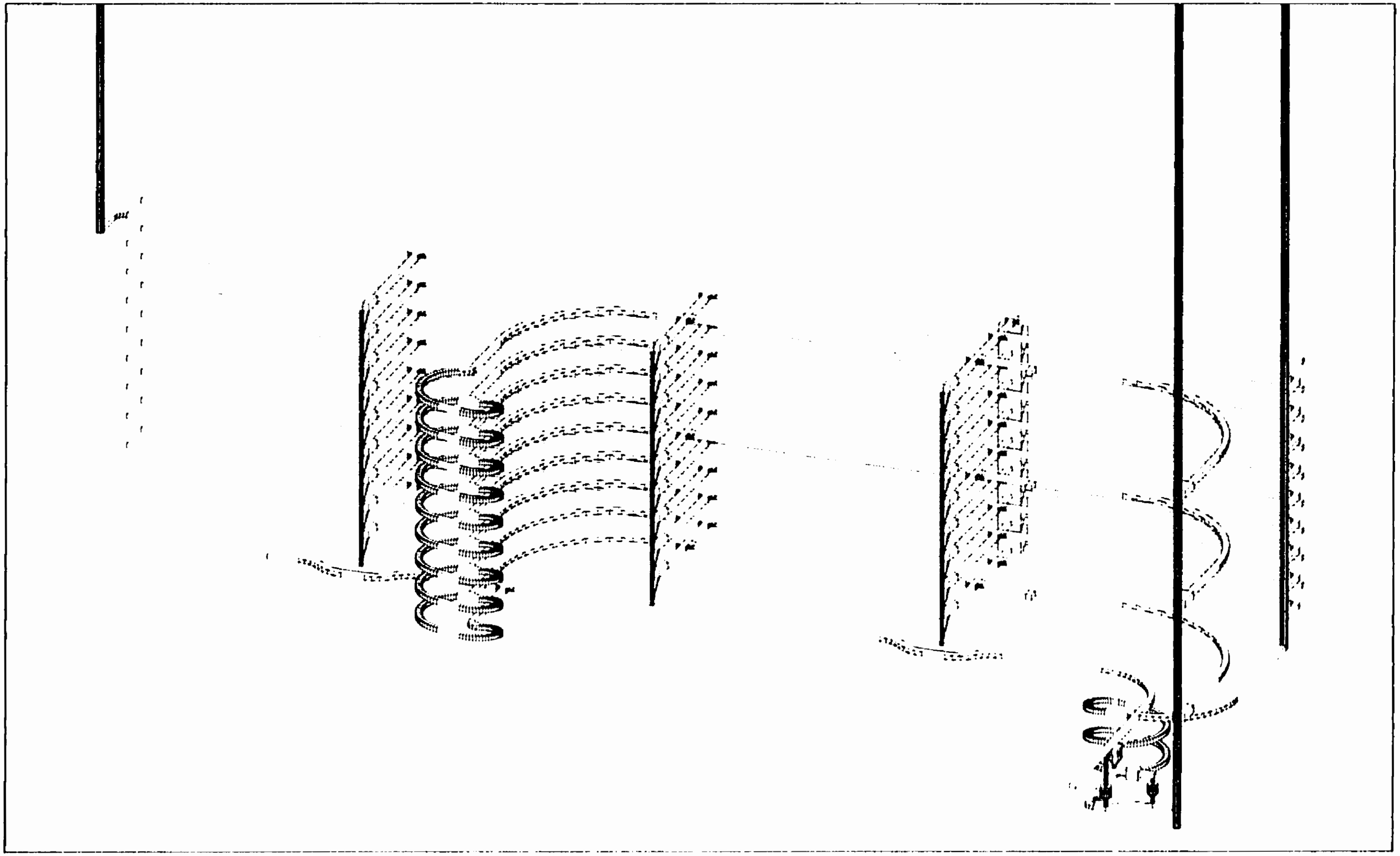

Figure 43: Fox River Project – Mine layout: large equipment, 15-m ore, 25-m levels

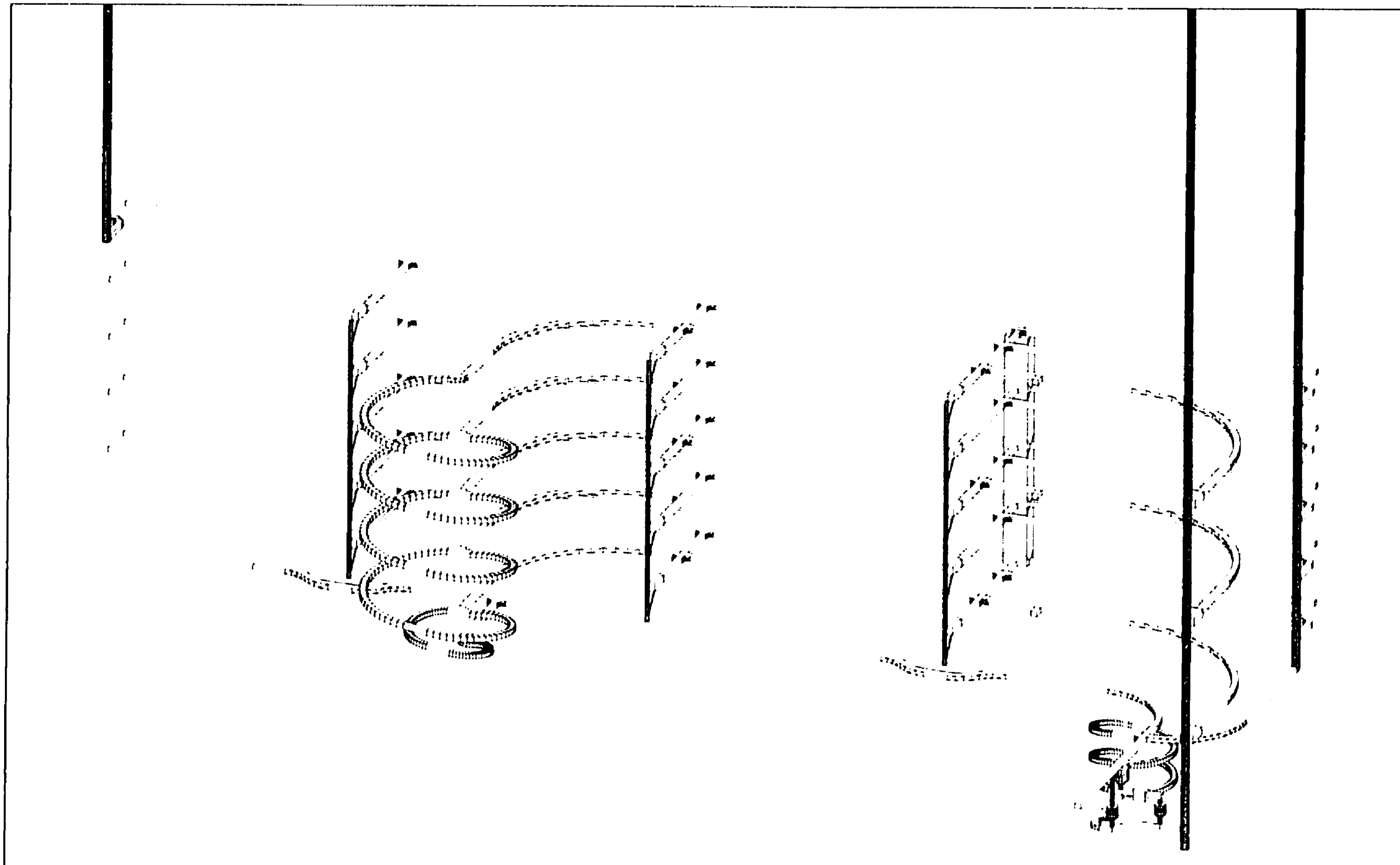

Figure 44: Fox River Project – Mine layout: large equipment, 15-m ore, 50-m levels

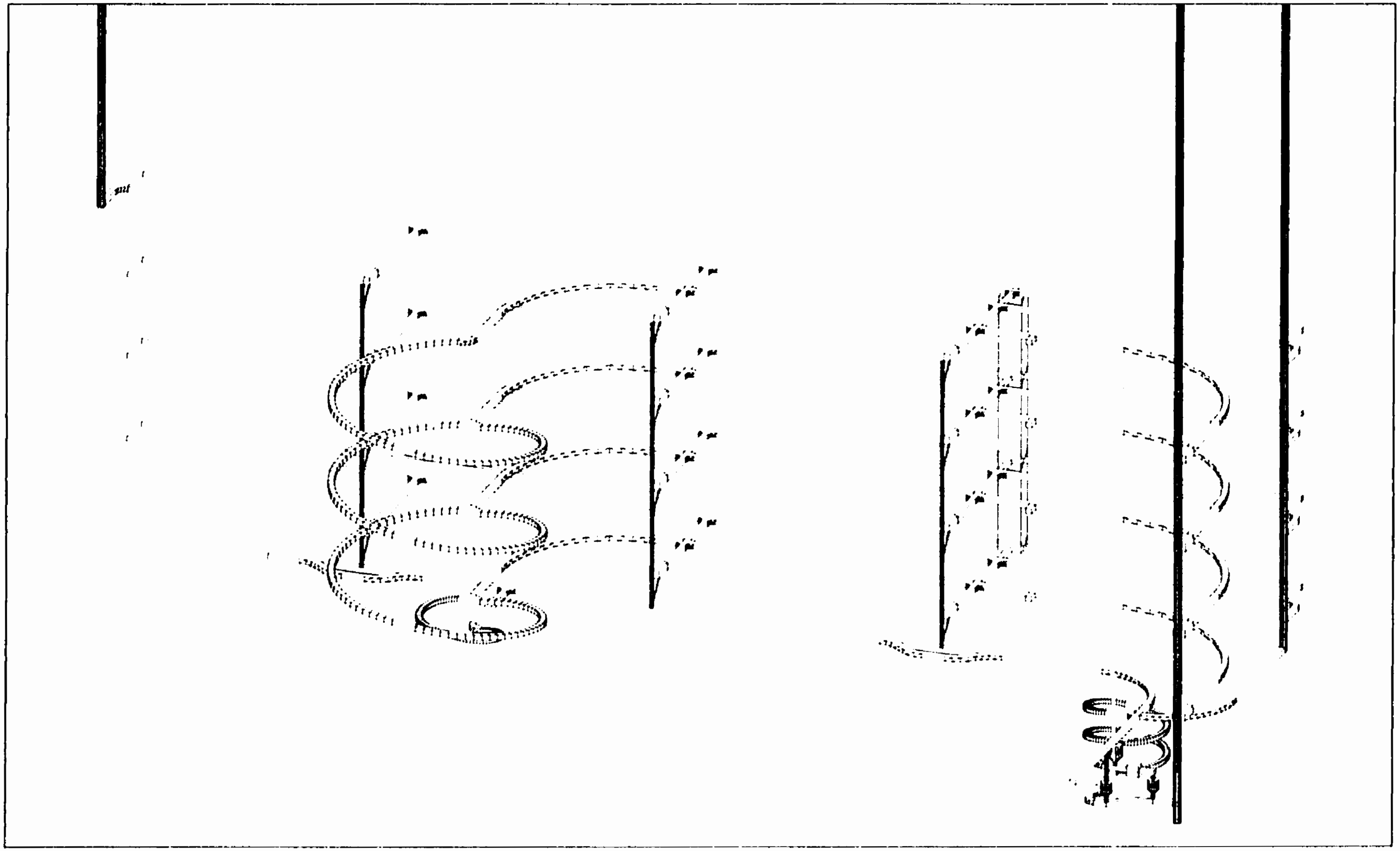

Figure 45: Fox River Project – Mine layout: large equipment, 15-m ore, 75-m levels

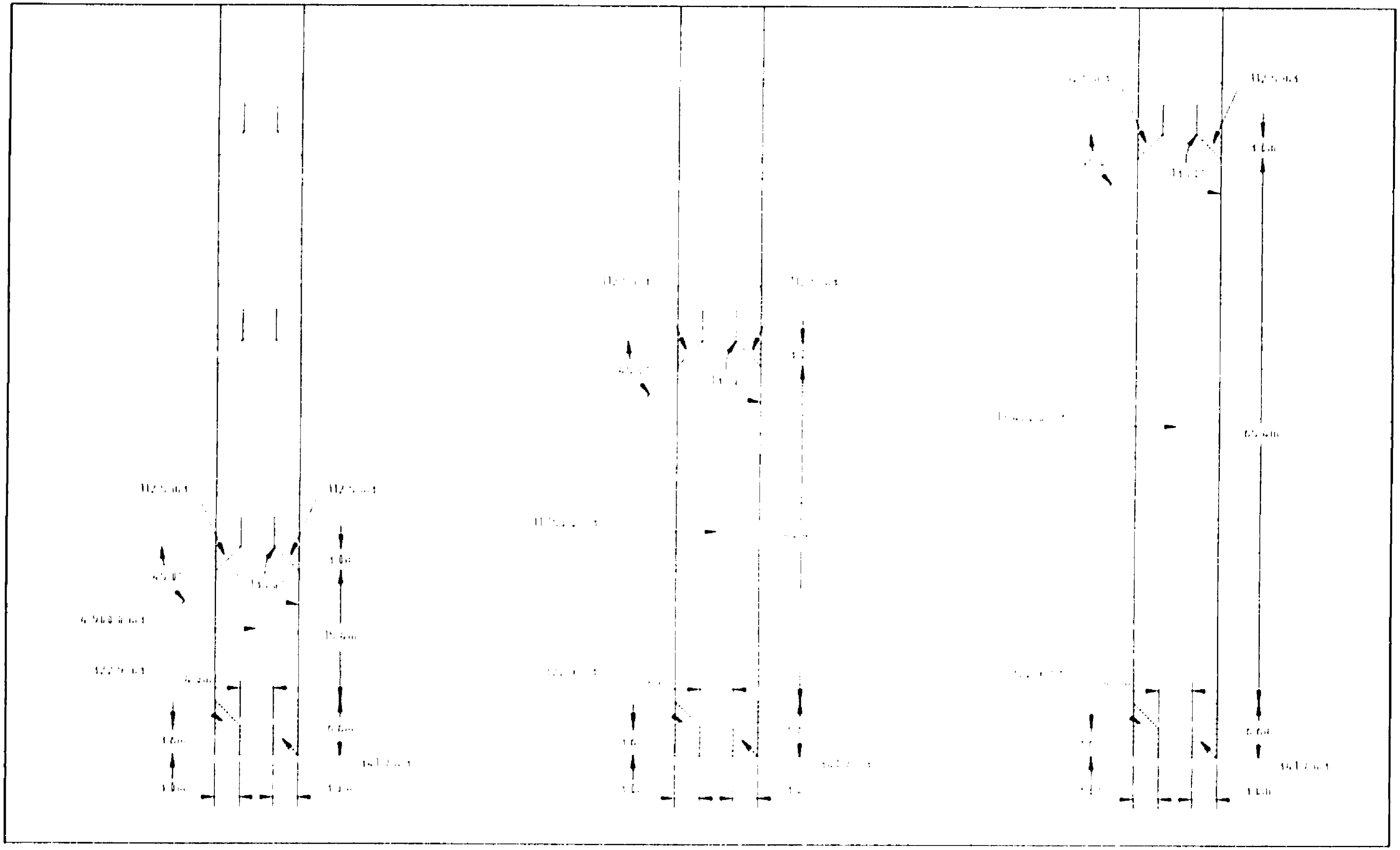

Figure 46: Fox River Project – Cross-sectional view – mid-size equipment, 10-m ore

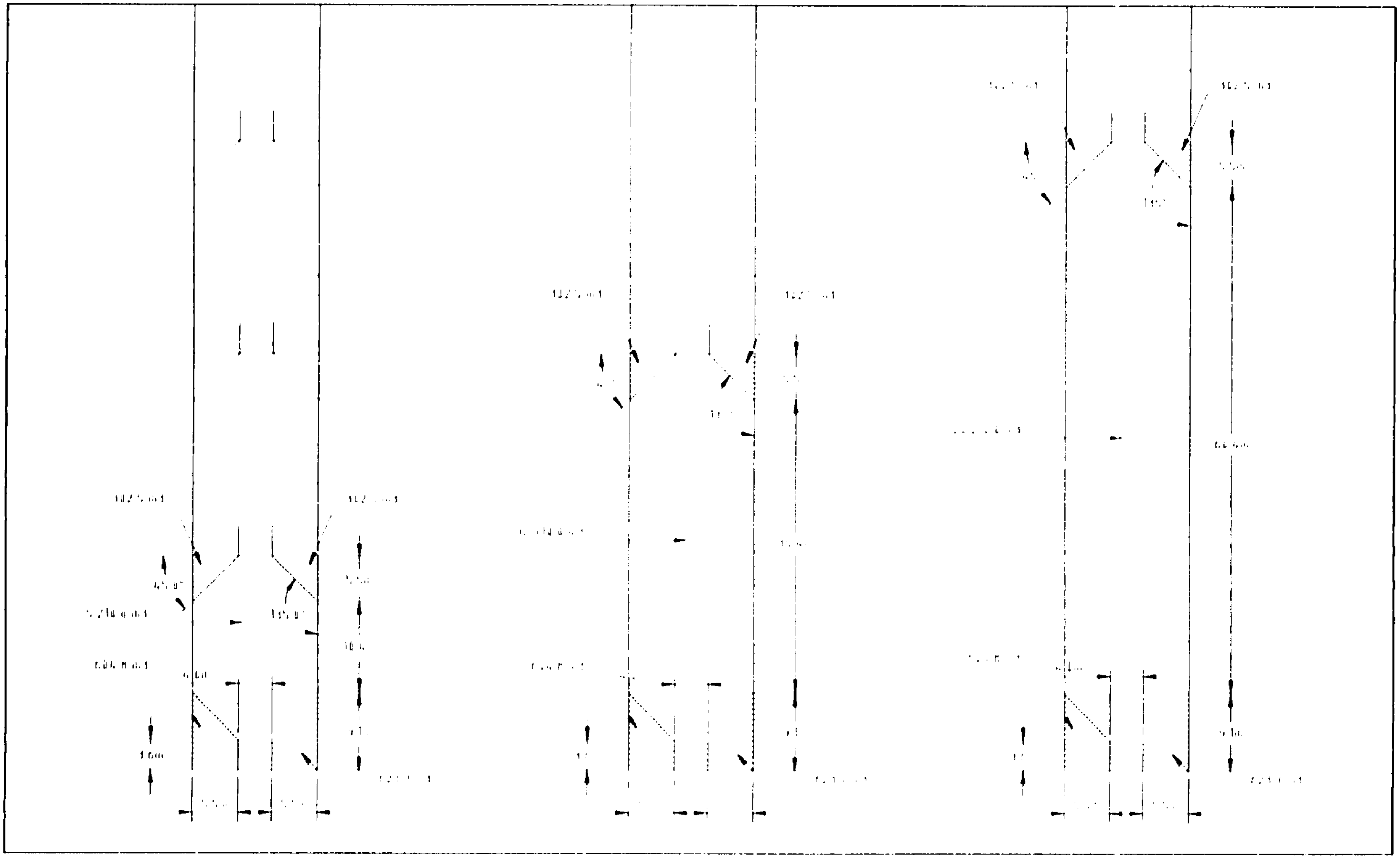

Figure 47: Fox River Project – Cross-sectional view – mid-size equipment, 15-m ore

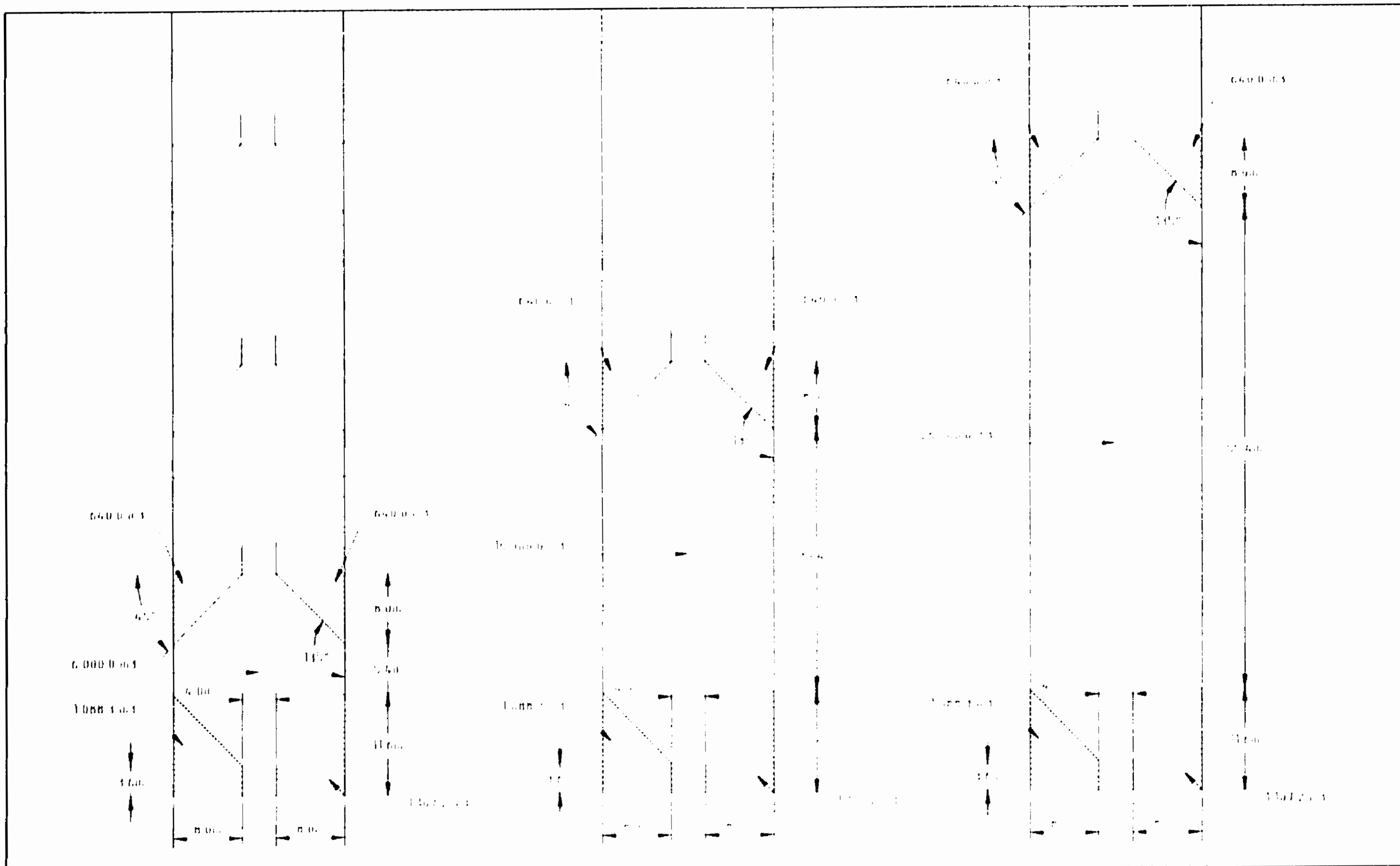

Figure 48: Fox River Project – Cross-sectional view – mid-size equipment, 20-m ore

The layout of stopes or mining blocks was determined by inter-level distance, orebody thickness and equipment size. As noted above, stopes in 10-m ore are five metres longer than in the case of 15- and 20-m ore. This is not only to account for the fact that, assuming that all other factors remain constant, slender stopes are more stable than wide ones, but also to increase the productive capacity of 10-m thick mining units and facilitate achieving reasonable production rates. Figure 46, Figure 47, and Figure 48 display cross-sectional views of stopes designed to exploit 15-m ore using mid-size equipment. Each section shows the corresponding undercut or sublevel, stope, and upper and lower wedges. It should be noted that this case study assumes that the lower wedges cannot be recovered by the proposed mining method. Pertinent dimensions and volumes are included in the drawings.

6.5.3 Estimation of Minable Reserves

Estimating minable reserves involved making assumptions regarding ore losses and dilution in the light of the mining method, equipment size, and level spacing. Three types of ore losses were considered: ore lost against backfill; oxidation of ore (sulphide ore) stuck against the stope walls; and, as noted above, ore left in the lower wedges of the stopes. The amount of ore lost against backfill and due to oxidation is a function of the wall area (the larger the area, the larger the ore losses), inter-level spacing (higher stopes have, generally, more irregularities), and equipment size (the use of large drilling equipment typically results in greater wall damage). In the case of oxidation of ore, and additional factor is the amount of time that the ore is left in the stope before mucking, which is directly related to stope size (i.e., the significance of this factor increased with orebody thickness). As depicted by Figure 46, Figure 47, and Figure 48, the volume of the lower stope wedges depends on orebody thickness and equipment size.

Table 35 presents the resulting ore losses for each of the 27 mining scenarios. Figure 49, Figure 50, and Figure 51 show graphically how total ore reserve recovery changes with inter-level spacing, equipment size, and orebody thickness. The following trends can be identified:

- With only two exceptions, the maximum ore reserve recovery for an equipment size/orebody thickness configuration is obtained with 50-metre levels.

- Regardless of orebody thickness, the highest ore recovery for 50- and 75-metre levels is achieved when small equipment is used. In the case of 25-metre levels, the highest recovery is obtained with mid-size equipment.
- Significantly higher ore recoveries are achieved in 10-metre ore, with the exception of mining with 75-metre levels where marginally higher recoveries are obtained in 15-metre ore.

Two different sources of dilution were taken into account: rock walls and backfill. In both cases, the amount of dilution was calculated as a function of wall area, inter-level distance and equipment size. Table 36 presents the results obtained for every mining scenario. Such results are graphically summarized in Figure 52, Figure 53, and Figure 54. It is, thus, evident that:

- Dilution increases with inter-level spacing.
- Dilution increases with equipment size.
- Dilution decreases with orebody thickness.

Once estimates for ore losses and dilution are obtained, the calculation of the corresponding minable reserves is straightforward. The results are shown in Table 36 as well as Figure 55 and Figure 56. Although the *in-situ* reserves were 12 million tonnes for every scenario, the minable reserves range from 9.6 million tonnes (20-m wide ore mined with small equipment and 25-m levels) to 13.2 million tonnes (10-m ore mined with large equipment and 75-m levels).

The minable ore reserve tonnage rapidly increases when inter-level spacing is raised from 25 m to 50 m, but it shrinks (or expands only marginally in the case of large equipment) when the distance between levels is increased to 75 metres. On the other hand, due to higher dilution, the ore reserve tonnage always increases when equipment size is increased. The rate of ore reserve tonnage increase is higher when equipment size changes from mid-size to large.

Finally, the minable reserves of individual mining blocks were calculated as shown in Table 37. Development ore, produced by the excavation of the sublevel (undercut) and crosscut, is not diluted. As noted above, only the upper wedges can be recovered with this method, but some of the ore is lost against backfill and due to oxidation. Also, both rock wall material and backfill dilute the wedges.

Table 35: Fox River Project – Ore losses and ore reserve recovery

Equipment Size	Orebody Width	Inter Level Spacing	Total Block Volume	Stope Specifications			Ore Losses										Total Ore Reserve Recovery	
				Strike Length	Ore Volume	F/W Height	Against Backfill			Wall Oxidation					Lower Wedge	Total Losses		
							Area	Width	Volume	Days	Area	Width	Volume					
	m	m	m^3	m	m^3	m	m^2	m	m^3		m^2	m	m^3		m^3	m^3	%	m^3
Small	10	25	6,250	25	4,838	18.2	203.8	0.20	28	24	1,142.7	0.10	79		746	853	86.35	2,590,650
		50	12,500	25	11,088	43.2	453.8	0.40	125	55	2,642.7	0.46	835		746	1,706	86.35	2,590,443
		75	18,750	25	17,338	68.2	703.8	0.60	291	87	4,142.7	1.08	3,072		746	4,109	78.09	2,342,598
	15	25	7,500	20	5,120	15.7	288.5	0.20	40	26	1,008.4	0.10	70		1,327	1,436	80.85	2,425,504
		50	15,000	20	12,620	40.7	663.5	0.40	183	63	2,383.4	0.49	810		1,327	2,320	84.53	2,535,959
		75	22,500	20	20,120	65.7	1,038.5	0.60	430	101	3,758.4	1.18	3,056		1,327	4,812	78.61	2,358,376
	20	25	10,000	20	5,870	13.2	360.8	0.20	50	29	1,078.9	0.10	74		2,324	2,448	75.52	2,265,459
		50	20,000	20	15,870	38.2	860.8	0.40	237	79	2,578.9	0.54	962		2,324	3,523	82.38	2,471,488
		75	30,000	20	25,870	63.2	1,360.8	0.60	563	129	4,078.9	1.32	3,719		2,324	6,606	77.98	2,339,354
Mid-Size	10	25	6,250	25	4,900	18.4	205.0	0.30	42	25	1,150.5	0.15	119		665	826	86.78	2,603,507
		50	12,500	25	11,150	43.4	455.0	0.60	188	56	2,650.5	0.57	1,040		665	1,893	84.86	2,545,745
		75	18,750	25	17,400	68.4	705.0	0.90	438	87	4,150.5	1.24	3,558		665	4,660	75.15	2,254,445
	15	25	7,500	20	5,210	15.9	290.8	0.30	60	26	1,012.3	0.15	105		1,229	1,393	81.42	2,442,649
		50	15,000	20	12,710	40.9	665.8	0.60	275	64	2,387.3	0.61	1,004		1,229	2,508	83.28	2,498,378
		75	22,500	20	20,210	65.9	1,040.8	0.90	646	101	3,762.3	1.36	3,523		1,229	5,397	76.01	2,280,367
	20	25	10,000	20	6,000	13.4	364.0	0.30	75	30	1,081.0	0.15	112		2,196	2,383	76.17	2,285,208
		50	20,000	20	16,000	38.4	864.0	0.60	358	80	2,581.0	0.67	1,187		2,196	3,740	81.30	2,439,046
		75	30,000	20	26,000	63.4	1,364.0	0.90	847	130	4,081.0	1.52	4,269		2,196	7,311	75.63	2,268,923
Large	10	25	6,250	25	4,872	18.3	202.4	0.40	56	24	1,136.3	0.20	157		627	840	86.57	2,596,954
		50	12,500	25	11,122	43.3	452.4	0.80	250	56	2,636.3	0.68	1,245		627	2,122	83.02	2,490,743
		75	18,750	25	17,372	68.3	702.4	1.20	581	87	4,136.3	1.43	4,069		627	5,277	71.86	2,155,659
	15	25	7,500	20	5,198	15.8	287.4	0.40	79	26	995.4	0.20	137		1,196	1,412	81.17	2,435,044
		50	15,000	20	12,698	40.8	662.4	0.80	365	63	2,370 4	0.73	1,198		1,196	2,759	81.60	2,448,121
		75	22,500	20	20,198	65.8	1,037.4	1.20	859	101	3,745.4	1.55	4,015		1,196	6,069	73.02	2,190,742
	20	25	10,000	20	5,998	13.3	359.9	0.40	99	30	1,059.8	0.20	146		2,163	2,408	75.92	2,277,577
		50	20,000	20	15,998	38.3	859.9	0.80	474	80	2,559.8	0.80	1,413		2,163	4,050	79.75	2,392,540
		75	30,000	20	25,998	63.3	1,359.9	1.20	1,125	130	4,059.8	1.73	4,855		2,163	8,143	72.86	2,185,726

Table 36: Fox River Project – Dilution and diluted ore reserves

Equipment Size	Orebody Width	Inter Level Spacing	Stope Specifications			Dilution									Diluted Reserves		
			Strike Length	Ore Volume	F/W Height	Hangingwall & Footwall			Backfill			Total Dilution			Mean SG	Per Block	Total Reserves
						Area	Width	Volume	Area	Width	Volume						
	m	m	m	m^3	m	m^2	m	m^3	m^2	m	m^3	m^3	tonnes	%		tonnes	tonnes
Small	10	25	25	4,838	18.2	455.0	1.00	455	203.8	0.60	122	577	1,610	6.9	3.88	23,198	11,135,164
		50	25	11,088	43.2	1,080.0	1.35	1,458	453.8	0.81	368	1,826	5,109	10.6	3.83	48,283	11,587,954
		75	25	17,338	68.2	1,705.0	1.90	3,240	703.8	1.14	802	4,042	11,323	16.2	3.74	69,888	11,182,082
	15	25	20	5,120	15.7	314.0	1.00	314	288.5	0.90	260	574	1,461	5.7	3.87	25,716	10,286,544
		50	20	12,620	40.7	814.0	1.35	1,099	663.5	1.22	806	1,905	4,909	8.8	3.81	55,628	11,125,641
		75	20	20,120	65.7	1,314.0	1.90	2,497	1,038.5	1.71	1,776	4,272	11,042	13.5	3.72	81,793	10,905,705
	20	25	20	5,870	13.2	264.0	1.00	264	360.8	1.20	433	697	1,658	5.2	3.86	31,864	9,559,183
		50	20	15,870	38.2	764.0	1.35	1,031	860.8	1.62	1,394	2,426	5,883	8.2	3.80	71,789	10,768,410
		75	20	25,870	63.2	1,264.0	1.90	2,402	1,360.8	2.28	3,103	5,504	13,410	12.5	3.70	106,984	10,698,402
Mid-Size	10	25	25	4,900	18.4	460.0	1.50	690	205.0	0.90	185	875	2,439	10.1	3.83	24,135	11,584,749
		50	25	11,150	43.4	1,085.0	2.03	2,197	455.0	1.22	553	2,750	7,697	15.4	3.75	50,126	12,030,268
		75	25	17,400	68.4	1,710.0	2.85	4,874	705.0	1.71	1,206	6,079	17,032	23.2	3.64	73,393	11,742,835
	15	25	20	5,210	15.9	318.0	1.50	477	290.8	1.20	349	826	2,129	8.0	3.83	26,555	10,622,117
		50	20	12,710	40.9	818.0	2.03	1,656	665.8	1.62	1,079	2,735	7,126	12.5	3.75	57,094	11,418,788
		75	20	20,210	65.9	1,318.0	2.85	3,756	1,040.8	2.28	2,373	6,129	16,015	19.0	3.63	84,426	11,256,766
	20	25	20	6,000	13.4	268.0	1.50	402	364.0	1.50	546	948	2,298	7.0	3.83	32,767	9,830,232
		50	20	16,000	38.4	768.0	2.03	1,555	864.0	2.03	1,750	3,305	8,165	11.2	3.74	73,206	10,980,904
		75	20	26,000	63.4	1,268.0	2.85	3,614	1,364.0	2.85	3,887	7,501	18,616	17.0	3.62	109,373	10,937,311
Large	10	25	25	4,872	18.3	456.3	2.50	1,141	202.4	1.50	304	1,444	4,029	15.7	3.74	25,670	12,321,825
		50	25	11,122	43.3	1,081.3	3.38	3,649	452.4	2.03	916	4,565	12,780	23.5	3.63	54,292	13,030,179
		75	25	17,372	68.3	1,706.3	4.75	8,105	702.4	2.85	2,002	10,107	28,318	34.4	3.49	82,209	13,153,510
	15	25	20	5,198	15.8	315.0	2.50	788	287.4	1.80	517	1,305	3,397	12.2	3.75	27,748	11,099,087
		50	20	12,698	40.8	815.0	3.38	2,751	662.4	2.43	1,610	4,360	11,471	19.0	3.64	60,434	12,086,747
		75	20	20,198	65.8	1,315.0	4.75	6,246	1,037.4	3.42	3,548	9,794	25,835	26.2	3.49	91,557	12,207,610
	20	25	20	5,998	13.3	265.0	2.50	663	359.9	2.10	756	1,418	3,499	10.3	3.76	33,867	10,160,081
		50	20	15,998	38.3	765.0	3.38	2,582	859.9	2.84	2,438	5,020	12,621	16.5	3.64	76,423	11,463,382
		75	20	25,998	63.3	1,265.0	4.75	6,009	1,359.9	3.99	5,426	11,435	28,879	24.8	3.49	116,308	11,630,759

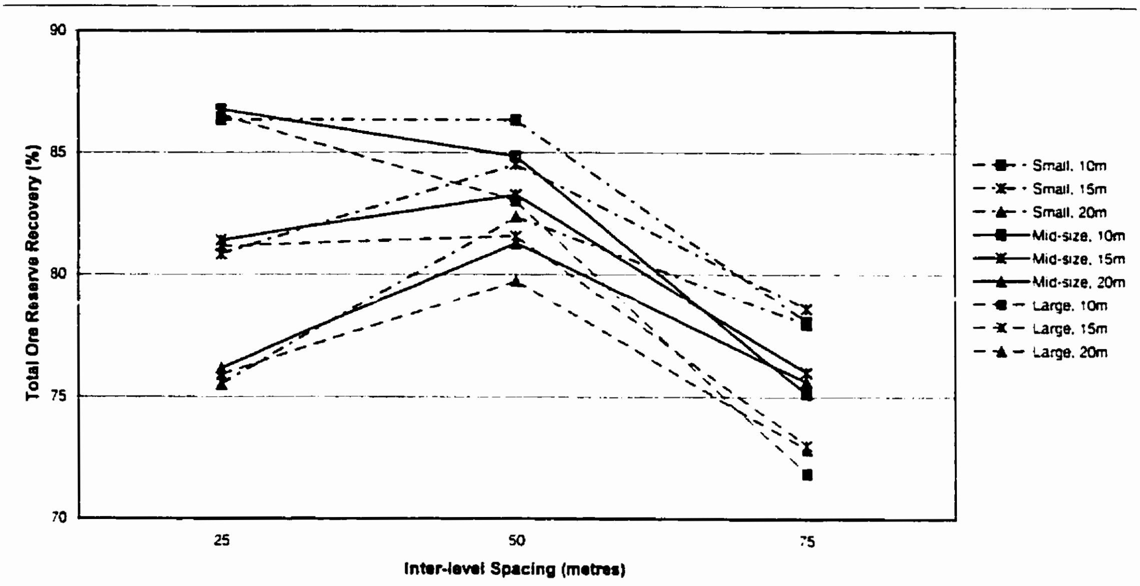

Figure 49: Fox River Project – Ore reserve recovery versus level spacing

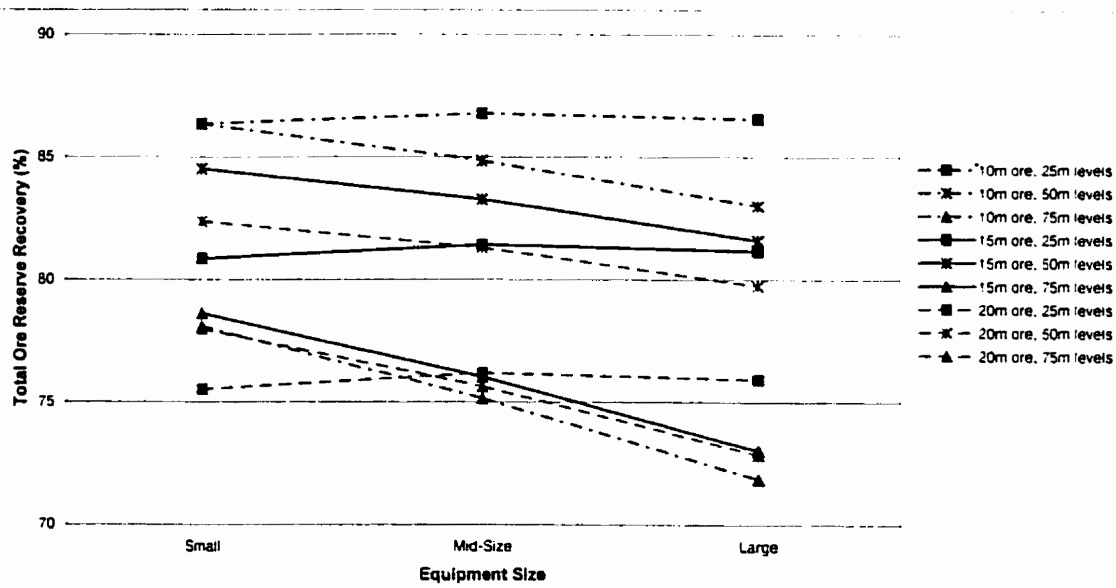

Figure 50: Fox River Project – Ore reserve recovery versus equipment size

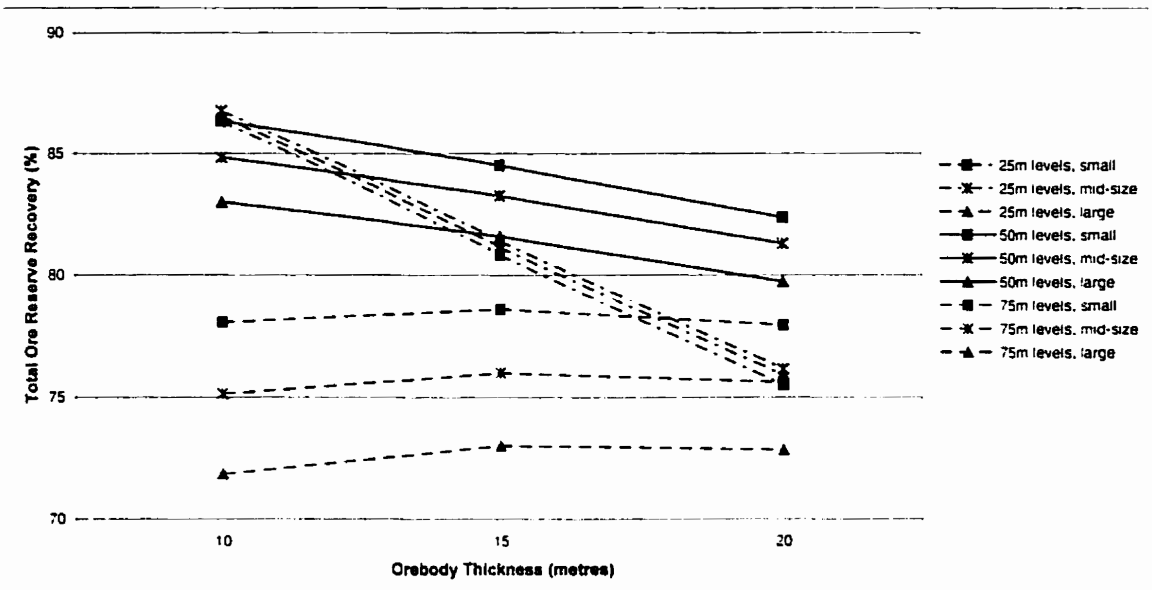

Figure 51: Fox River Project – Ore reserve recovery versus orebody thickness

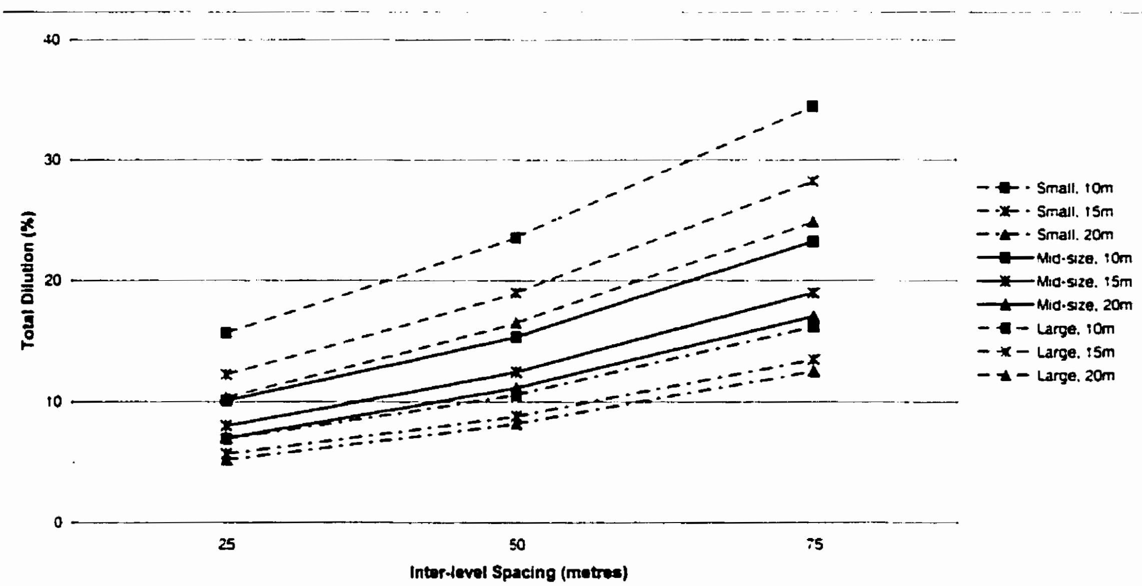

Figure 52: Fox River Project – Dilution versus inter-level spacing

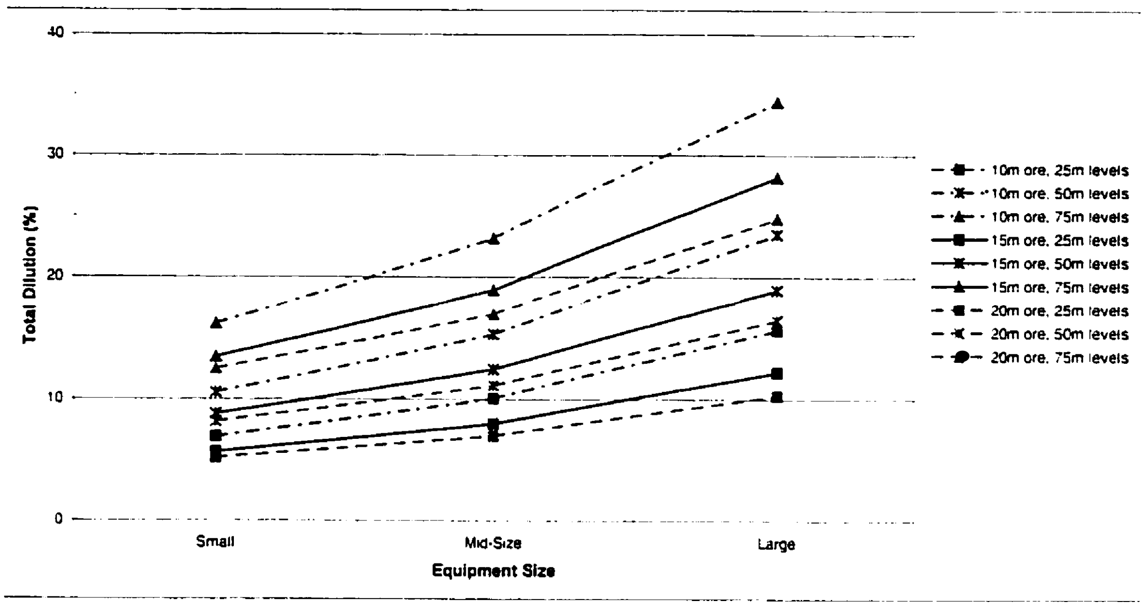

Figure 53: Fox River Project – Dilution versus equipment size

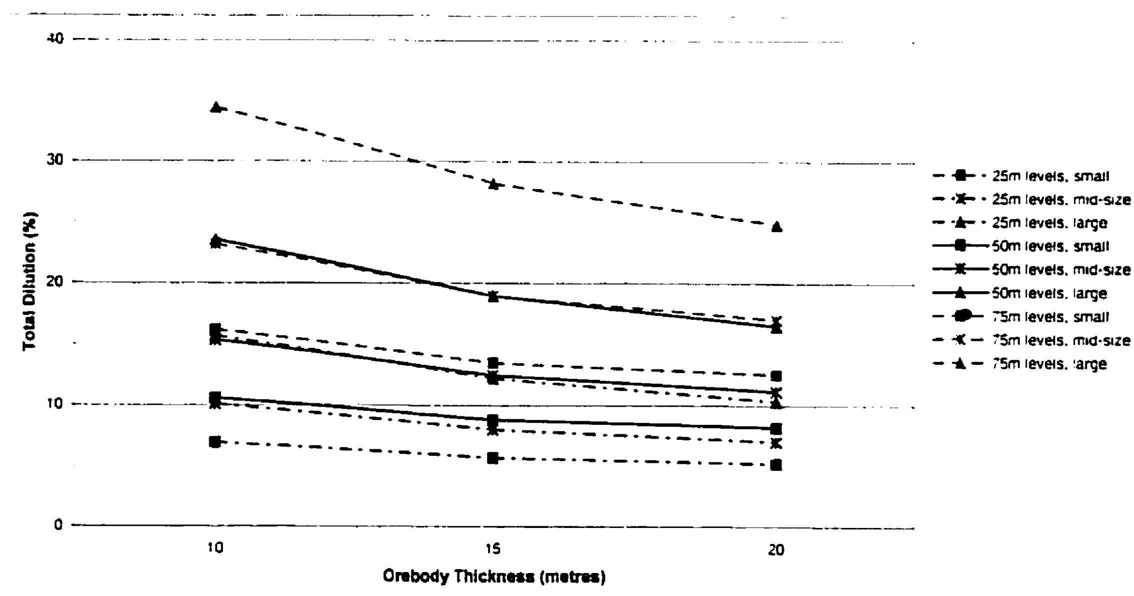

Figure 54: Fox River Project – Dilution versus orebody thickness

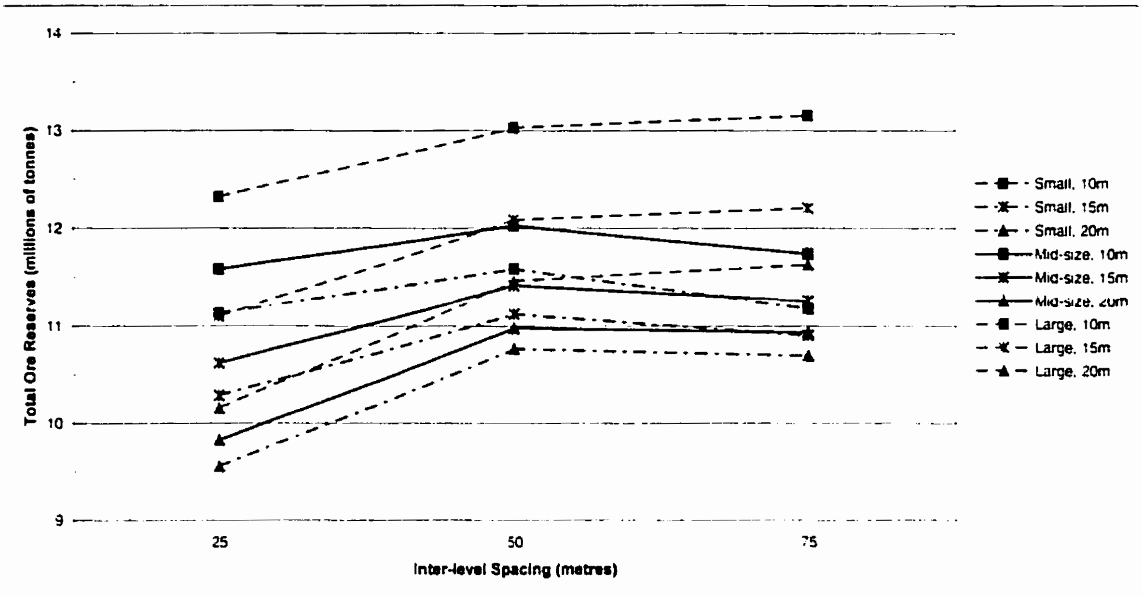

Figure 55: Fox River Project – Ore reserves versus inter-level spacing

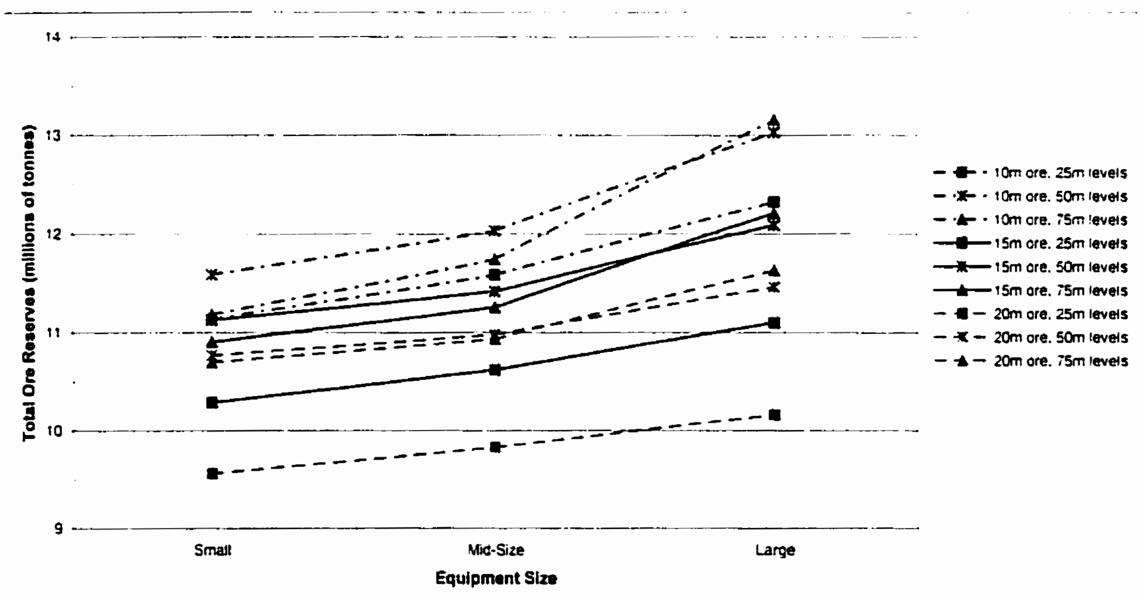

Figure 56: Fox River Project – Ore reserves versus equipment size

Table 37: Fox River Project – Mining block diluted ore production

Equipment Size	Orebody Width	Inter Level Spacing	Development tonnes			Stope tonnes						Upper Wedges tonnes						Total tonnes
							Losses		Dilution				Losses		Dilution			
	m	m	Sublevel	Crosscut	Total	Ore	Backfill	Oxidation	Walls	Backfill	Total	Ore	Backfill	Oxidation	Walls	Backfill	Total	
Small	10	25	1,296	344	1,640	19,352	104	268	1,125	232	20,337	1,024	8	47	240	12	1,221	23,198
		50	1,296	344	1,640	44,352	484	3,126	4,050	719	45,510	1,024	16	215	324	17	1,133	48,283
		75	1,296	344	1,640	69,352	1,140	11,782	9,263	1,581	67,274	1,024	25	505	456	23	974	69,888
	15	25	1,037	575	1,612	20,482	133	206	600	461	21,203	2,599	26	72	342	58	2,902	25,716
		50	1,037	575	1,612	50,482	680	2,887	2,835	1,533	51,283	2,599	52	354	462	79	2,734	55,628
		75	1,037	575	1,612	80,482	1,641	11,375	6,840	3,441	77,746	2,599	78	847	650	111	2,435	81,793
	20	25	1,037	805	1,842	23,482	145	189	300	704	24,152	5,379	54	109	492	161	5,870	31,864
		50	1,037	805	1,842	63,482	842	3,257	2,430	2,571	64,383	5,379	108	590	664	218	5,564	71,789
		75	1,037	805	1,842	103,482	2,091	13,435	6,270	5,898	100,124	5,379	161	1,411	935	307	5,018	106,984
Mid-Size	10	25	1,440	402	1,842	19,600	159	410	1,733	353	21,116	900	11	66	338	16	1,177	24,135
		50	1,440	402	1,842	44,600	732	3,910	6,136	1,084	47,178	900	22	250	456	22	1,106	50,126
		75	1,440	402	1,842	69,600	1,718	13,685	13,979	2,380	70,557	900	32	545	641	31	994	73,393
	15	25	1,152	674	1,826	20,840	204	315	936	625	21,882	2,420	36	104	495	73	2,848	26,555
		50	1,152	674	1,826	50,840	1,029	3,595	4,301	2,059	52,575	2,420	73	421	668	98	2,693	57,094
		75	1,152	674	1,826	80,840	2,475	13,154	10,328	4,608	80,147	2,420	109	937	941	138	2,452	84,426
	20	25	1,152	946	2,098	24,000	224	288	486	900	24,873	5,120	77	159	720	192	5,796	32,767
		50	1,152	946	2,098	64,000	1,276	4,040	3,694	3,240	65,617	5,120	154	706	972	259	5,491	73,206
		75	1,152	946	2,098	104,000	3,156	15,468	9,473	7,410	102,259	5,120	230	1,607	1,368	365	5,016	109,373
Large	10	25	1,800	448	2,248	19,488	211	547	2,906	585	22,220	756	12	80	516	23	1,202	25,670
		50	1,800	448	2,248	44,488	974	4,707	10,252	1,802	50,860	756	24	274	696	31	1,185	54,292
		75	1,800	448	2,248	69,488	2,289	15,704	23,334	3,961	78,790	756	36	571	980	43	1,172	82,209
	15	25	1,440	781	2,221	20,790	273	418	1,575	936	22,609	2,205	44	131	788	99	2,917	27,747
		50	1,440	781	2,221	50,790	1,374	4,312	7,189	3,085	55,378	2,205	88	480	1,063	134	2,834	60,433
		75	1,440	781	2,221	80,790	3,302	15,042	17,243	6,908	86,596	2,205	132	1,019	1,496	189	2,739	91,556
	20	25	1,440	1,114	2,554	23,990	301	381	825	1,259	25,393	4,805	96	204	1,163	252	5,919	33,866
		50	1,440	1,114	2,554	63,990	1,706	4,834	6,176	4,535	68,162	4,805	192	817	1,569	341	5,706	76,422
		75	1,440	1,114	2,554	103,990	4,214	17,649	15,818	10,373	108,318	4,805	288	1,770	2,209	479	5,435	116,307

6.5.4 Mine Development and Production Programs

Mine development programs covering the entire productive life of each mining scenario were assembled based on the respective three-dimensional layouts and ore reserves. Such programs were developed through the following steps:

- ***Calculation/measurement of sub-horizontal[132] development distances***

  Using the above-mentioned 3D mine layouts, and with the help of the AutoCAD program, the length corresponding to each development opening was either directly measured or calculated. Similarly, the tonnage of ore, waste, and backfill[133] produced by the excavation of each opening was obtained either by multiplying the respective length by its cross-sectional area, or directly from the AutoCAD layout. The level-by-level results are shown in Table 70 to Table 78 (one table for each equipment size/orebody width combination) and graphically summarized in Figure 57 and Figure 58. It can be seen that:

  - Regardless of orebody width and interlevel spacing, the total length of sub-horizontal development remains virtually constant when the size of equipment increases (Figure 57).
  - Development distance and the corresponding amount of waste generated depend on the number of levels to be excavated.
  - The amount of waste (rock plus backfill) produced significantly increases with equipment size (Figure 58).
  - The thicker the ore, the shallower the slope of the development-waste versus equipment-size curve. This is due to the fewer number of levels required to exploit thicker orebodies (Figure 58).

- ***Mining cycle calculations***

  Details about the mining cycle are needed to determine the productive capacity of a single mining block and the minimum speed of development that can support a determined production rate. Using operating data provided by equipment manufacturers and based on the assumptions made in Section 6.4, the duration of the mining cycle for each of the 27

[132] Neither the main ramp (that goes from the uppermost level to the lowest level) nor the auxiliary ramp (driven from the lowest level to the main haulage level) are included in this category.

[133] Once the backfill placed in a mined-out stope is cured, the corresponding sections of the crosscut and sublevel/undercut are re-excavated (in the backfill) in order to gain access to the top of the stopes located in the next level. See Pelley (1990) for a discussion of the pros and cons of backfill development.

scenarios was calculated. The results are shown in Table 79, Table 80, and Table 81. The following additional assumptions and calculations were made:

a. Drilling rates of 125, 150, and 180 metres/day were assumed for 25-, 50-, and 75-metre levels, respectively. This is to account for the fact that less set-up time and longer blastholes are required for mine designs that include longer inter-level distances.

b. Number of rounds blasted on each block. It was determined that there was a minimum tonnage/round that was practical from both blasting and mucking perspectives (about 4,500 tonnes) and that there was a maximum number of slices that could be efficiently blasted on 20- and 25-m long blocks (nine). As a result, the number of rounds/block fluctuated from 5 for 25-m levels, 6 for 50-m levels (with the exception of 20-m ore, which required 7 rounds), and 8 for 75-m levels (with the exception of 20-m ore, which needed 9 rounds).

c. The mucking rates for each equipment size were determined according to the mucking rate of the corresponding LHD, haulage distance, shift duration, efficiency, fixed cycle times, and mechanical availability. Table 38, for example, shows the calculations made for the large-equipment scenarios. It has been assumed that the project will operate two shifts per day and eight hours per shift (no overtime).

d. Backfill. The amount of backfill required was obtained by dividing the volume to be filled in each stope by the specific gravity of the backfill. Archibald (1999) provided backfill specifications: specific gravity of backfill = 1.75, backfill plant production rate = 250 tonnes per hour (142.9 m^3/hour). A backfill cure time of seven days was assumed for every case.

Figure 59 shows graphically how the length of the mining cycle increases as the interlevel spacing is raised from 25 to 75 metres. Larger cycles are the result of larger mining blocks. Total cycle times range from 47 days (large equipment mining 10-m thick ore with 25-m levels) to 279 days (small equipment exploiting 20-m thick ore with 75-m levels). It can be seen that the slopes of curves for small equipment are significantly more steep than those for mid-size and large equipment. This indicates that, due to its higher productivity, the increase in cycle length is not as significant for larger equipment.

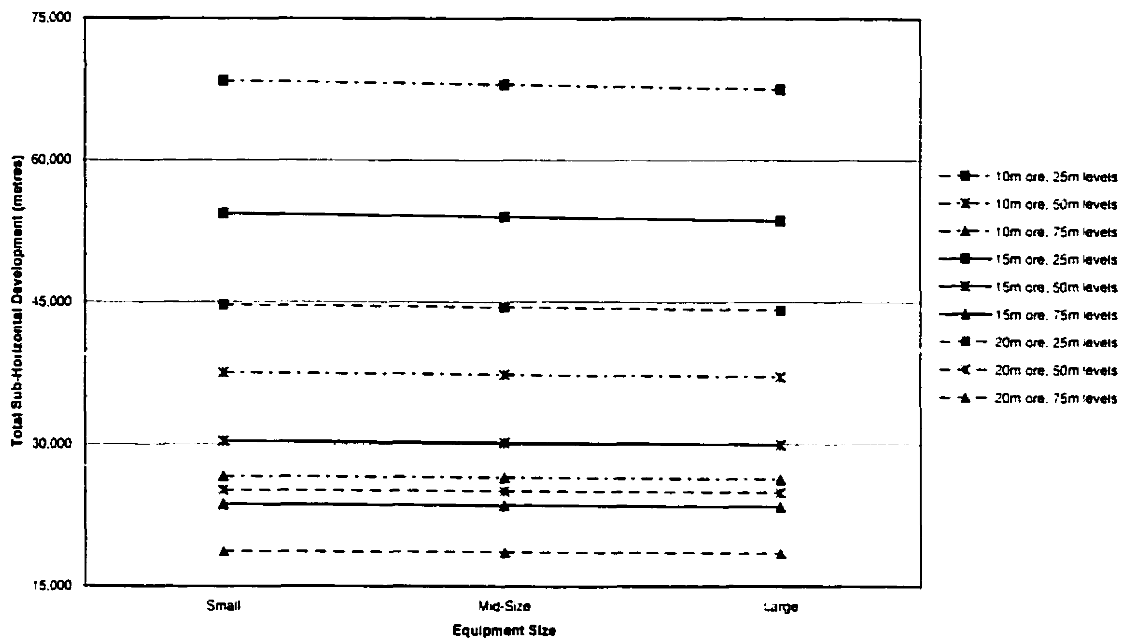

Figure 57: Fox River Project – Horizontal development versus equipment size

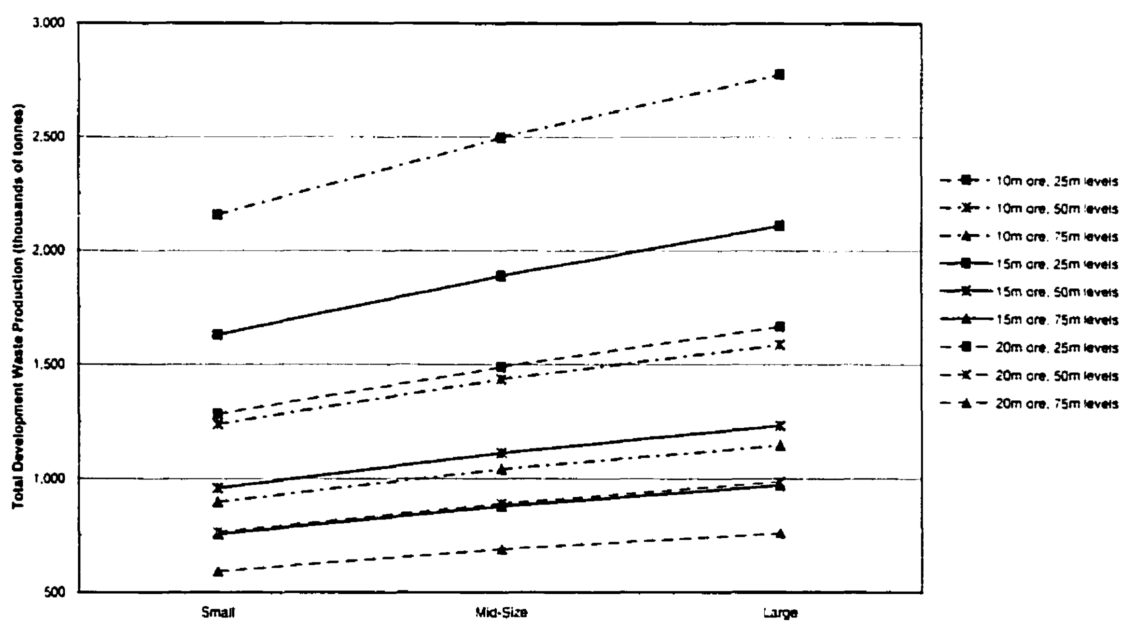

Figure 58: Fox River Project – Waste production versus equipment size

Table 38: Fox River Project – LHD cycle for large equipment cases

Capacity of loading equipment (LHD)	10.0 yd^3	15 tonnes
Input Parameters		
Effective working hours	6.0	hours/shift
Effective working minutes	55.0	minutes/hour
Mechanical availability	80.0%	
Average one way haulage distance	228.0	m
Average traveling speed, loaded	6.0	km/hour
Average traveling speed, empty	6.0	km/hour
Time to load LHD bucket	1.6	minutes
LHD travel time - loaded	2.3	minutes
Time to position LHD @ orepass	0.8	minutes
Time to dump LHD bucket	0.8	minutes
LHD travel time - empty	2.3	minutes
LHD time arranging muck at beginning of shift	30.0	minutes
Production Calculations		
Total effective working time	264.0	minutes
Effective mucking time	234.0	minutes
Total LHD cycle time to dump one bucket	7.8	minutes
Number of mucking cycles	30	
Production per shift	450.0	tonnes

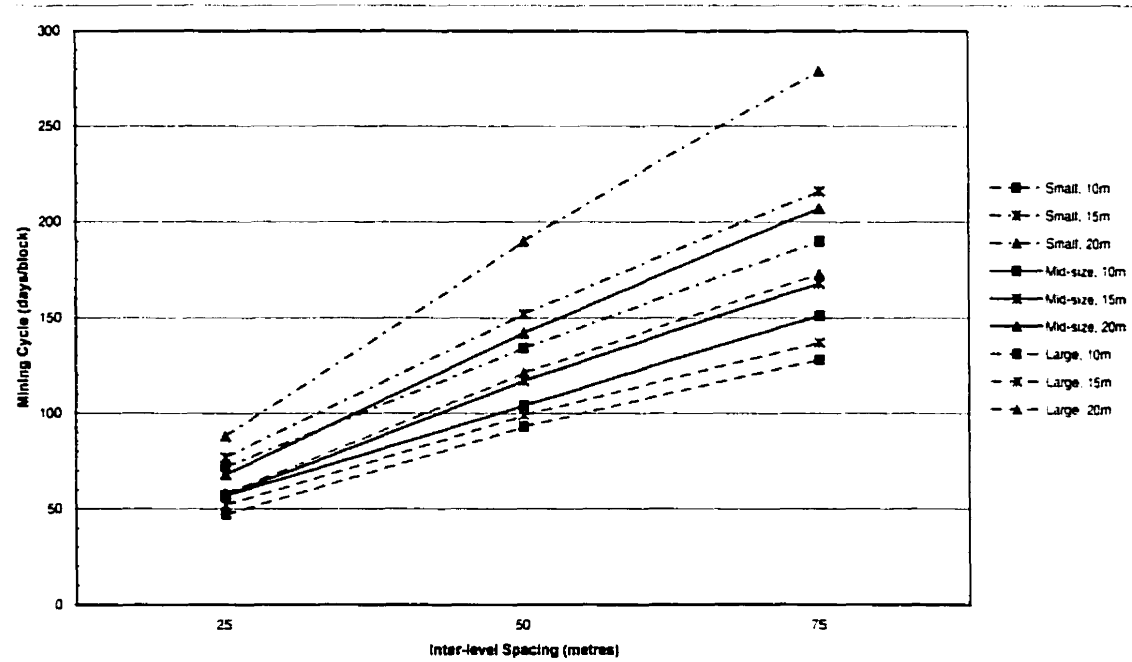

Figure 59: Fox River Project – Mining cycle versus interlevel spacing

- ***Determination of ore production rate***

A mining block produces ore due to both development and stoping activities (see Table 37). Therefore, the determination of the actual production rate of each mining scenario is based on assumptions made regarding the number or stopes to be exploited simultaneously, and the number of development crews to be kept in operation at the same time.

The number of active stopes depends principally on the mining sequence. In this case, a "*1-3-5*" single-lift mining sequence was employed (see Section 6.4.6). Since it uses primary and secondary stopes only, the maximum number of active stopes is equal to one half of the total number of stopes in a level. However, space and extraction routes are very restricted in underground mines, and there is a practical limit on the amount of equipment that can operate on the same production level simultaneously. On the other hand, the need to sustain a minimum production rate that justifies the large capital investment, coupled with the restrictions associated with a mine with very few producing blocks, puts pressure to bring as many stopes into simultaneous operation as possible.

An initial attempt with four simultaneous active stopes resulted in very low production rates. Thus, although four could be considered as a fair number of stopes, it was decided to continue the case study with five active stopes. Such a decision is supported by the fact that the mine layout includes three orepasses (i.e., traffic congestion in a level can be avoided with proper scheduling and equipment dispatch). It was expected that the corresponding increase in capital investment would be more than compensated by the larger production rates that would be thus achieved.

The resulting effective stope production rates are shown in Table 82, Table 83, and Table 84. The *number of days per level* (i.e., the number of days required to mine out an entire level) for each mining scenario was determined with the help of the computer program *MS Project 98*. Such numbers were obtained by assembling comprehensive production programs for one level using mining cycle data and, as noted above, assuming the simultaneous exploitation of five stopes and the 1-3-5 mining sequence. It was also assumed that, by mining two consecutive sets of primary stopes at the beginning of the sequence of every level, it would not be necessary to wait for the backfill of the first set to cure in order to continue mining succeeding sets.

Figure 60 shows the relationship between stope production rate and orebody thickness.[134] In the case of small and mid-size equipment, the curves are more or less clustered, indicating that, for a given orebody thickness, the difference in production rate between scenarios with 25-, 50-, and 75-m levels is not significant. They also present definite (but moderate) increasing trends: thicker orebodies resulted in slightly higher production rates. On the other hand, the behaviour of large equipment scenarios is quite distinct. First, production rate greatly increases with higher inter-level spacings, suggesting that large equipment can more adequately match the higher productive capacity of larger stopes. Second, the (generally positive) slopes of the curves are more pronounced, particularly in the case of 25-m interlevel spacing.[135] This trend suggests also that large equipment can take full advantage of thicker stopes.

- ***Matching development and production programs***

The previous calculation of stope production rates was strictly based on stope geometry, mining sequence, and assumptions regarding equipment and personnel productivity. It did not consider the impact of mine development. To reconcile those two important issues in this case study, it was assumed that enough resources (i.e., personnel and equipment) would be devoted to mine development in order to allow the production levels obtained above. The intensity and speed of mine development, in turn, are controlled by two main factors: advance rate (e.g., metres per round) and the number of active development crews.

Although important, the details of determining an optimum mine development speed are beyond the scope of this thesis. Furthermore, local factors typically affect such a speed,

[134] It is assumed that the mine operates seven days per week, 52 weeks per year.

[135] It should be noted that production rates obtained for 15-m ore mined with 75-m inter-level spacing are higher than expected. This is because the production-rate calculation procedure assumes that, under such scenarios, three complete levels with 75-m stopes will be extracted. The truth is, a 15-m thick orebody is only 200 metres high and, thus, there are only two sets of 75-m high stopes to be mined (the first level has 50-m stopes). Since the amount of development and production drilling in these scenarios are virtually the same, regardless of stope height, the overall production rate for the first level would be lower than for the last two.

making it difficult to define accurately a development rate that can be used for long-term planning. Thus, in order to simplify the analysis and eliminate the effect of development speed variability, it was decided to assume a single advance rate for all headings (levels, sublevels, crosscuts, and ramps) and all scenarios.[136] Such a rate was calculated as follows:

$$\text{rate} = \frac{3.6\ \dfrac{\text{m}}{\text{round}} \times 28\ \dfrac{\text{rounds}}{\text{month}}}{30\ \dfrac{\text{days}}{\text{month}}} = 3.36\ \frac{\text{m}}{\text{day}}$$

It is obvious that, as more development crews are deployed, the overall production rate can be thus increased (e.g., additional stopes could be brought into production simultaneously, not just five). However, space, scheduling, and financial limitations restrict the amount of development activity that an operation can undertake without affecting its ore production process. Thus, an "*optimum*" development rate that simultaneously fulfils short-term production needs and provides a degree of flexibility to the operation must be defined.

The number of simultaneous development crews was determined by assuming that, in addition to sustaining the corresponding stope production rate, the objective of the development program was to bring the mine into production in six years (i.e., to start producing from stopes at the beginning of the seventh year). The application of the same constraint to every scenario is validated by the fact that, at the beginning of the mine life, the amount of mine development is virtually identical for all of them (the main differences being the distance between levels, and the number and length of in-ore crosscuts). This was confirmed by the results shown in Table 85.

The mine life (and, thus, the intensity of mine development) of a scenario is a function of equipment size: production equipment can achieve a certain production rate, which, in turn, demands a corresponding development rate. The larger the equipment, the higher the

[136] It could be argued that large equipment excavating large development openings could achieve significantly higher development rates. However, this reasoning neglects to take into consideration ground-control issues. For instance, the requirement to rockbolt and mesh a heading right to the face severely restricts advance, regardless of equipment size and heading cross-sectional dimensions.

respective development rate (and the shorter the mine life, see Table 82, Table 83, and Table 84). This is graphically depicted by Figure 62, Figure 63, and Figure 64, which present the number of yearly development crews required by the production programs of small, mid-size, and large equipment mining scenarios, respectively, over their entire mine life. It can be seen that, as the size of production equipment increases and the mine life decreases, the development rate rapidly increases, demanding a larger number of development crews. It is interesting to note that, regardless of equipment size, the number of development crews required by the various orebody thickness/interlevel spacing combinations is virtually constant (see Table 85). As expected, the number of active development crews decreases towards the end of the mine life of every scenario.

- ***Total ore production program***

Total ore production programs for small, mid-size, and large equipment scenarios are shown in Table 39, Table 40, and Table 41, respectively. The higher concentration of both stoping and development activities in larger equipment scenarios is evident. Also, it can be seen that, as inter-level spacing is increased and the production capacity of single stopes rises, the contribution of ore from the corresponding development programs becomes irregular and less significant.

Changes of total effective ore production rate (i.e., the production rate that includes both stope and development ore) with orebody thickness are shown in Figure 61. As expected, the curves in this case are similar to those displayed in Figure 60. In fact, apart from the obvious increase in production rates (due to the added ore coming from development activities), small and mid-size equipment curves are virtually the same as their Figure 60 counterparts: they are also clustered and present the same general trend. Large equipment curves, on the other hand, are now more clustered, indicating that the addition of development ore balances to some extent the differences in stope production rate that result from using different inter-level spacings. Finally, it is interesting to note that, regardless of equipment size, 25-metre inter-level spacings result in the highest production rates for 20-metre thick ore.

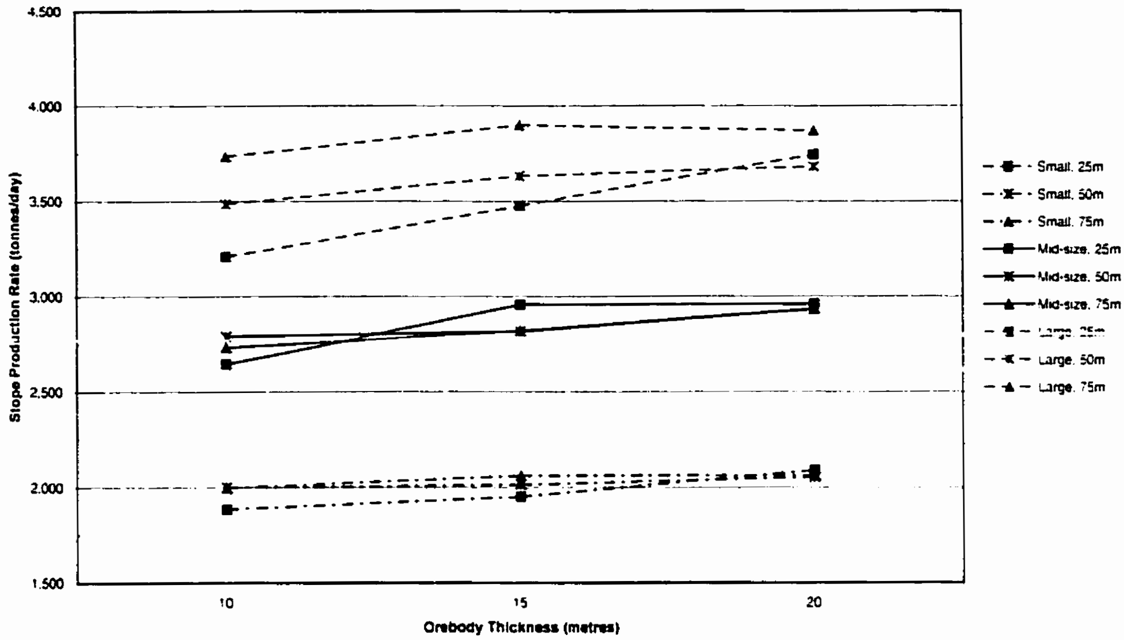

Figure 60: Fox River Project – Stope production rate versus orebody thickness

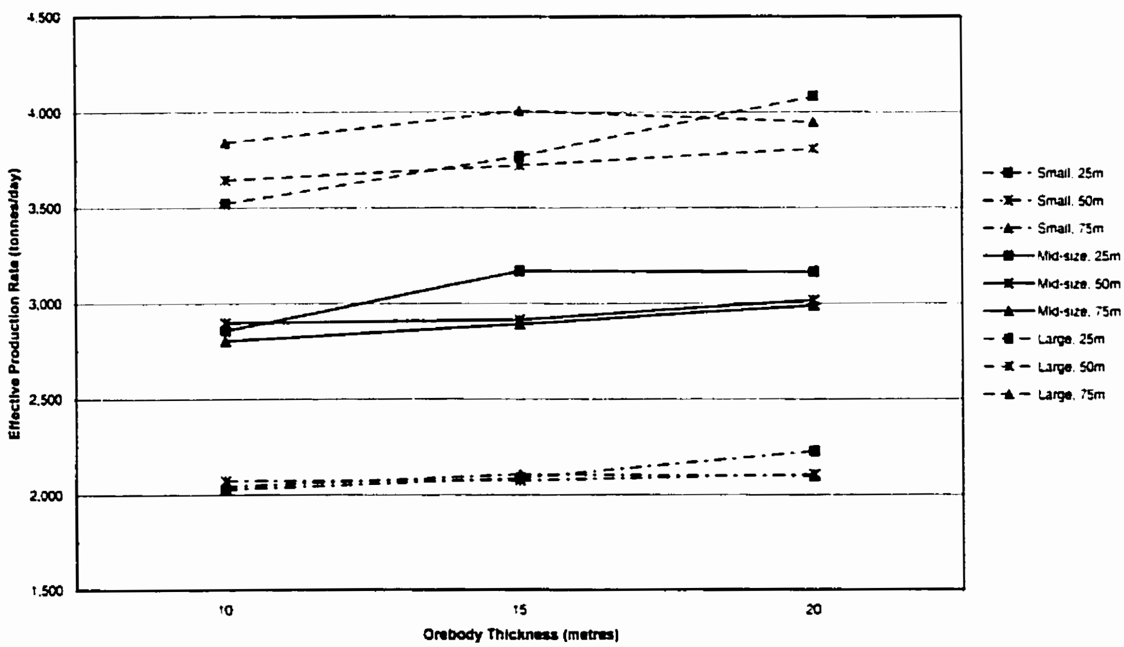

Figure 61: Fox River Project – Effective production rate versus orebody thickness

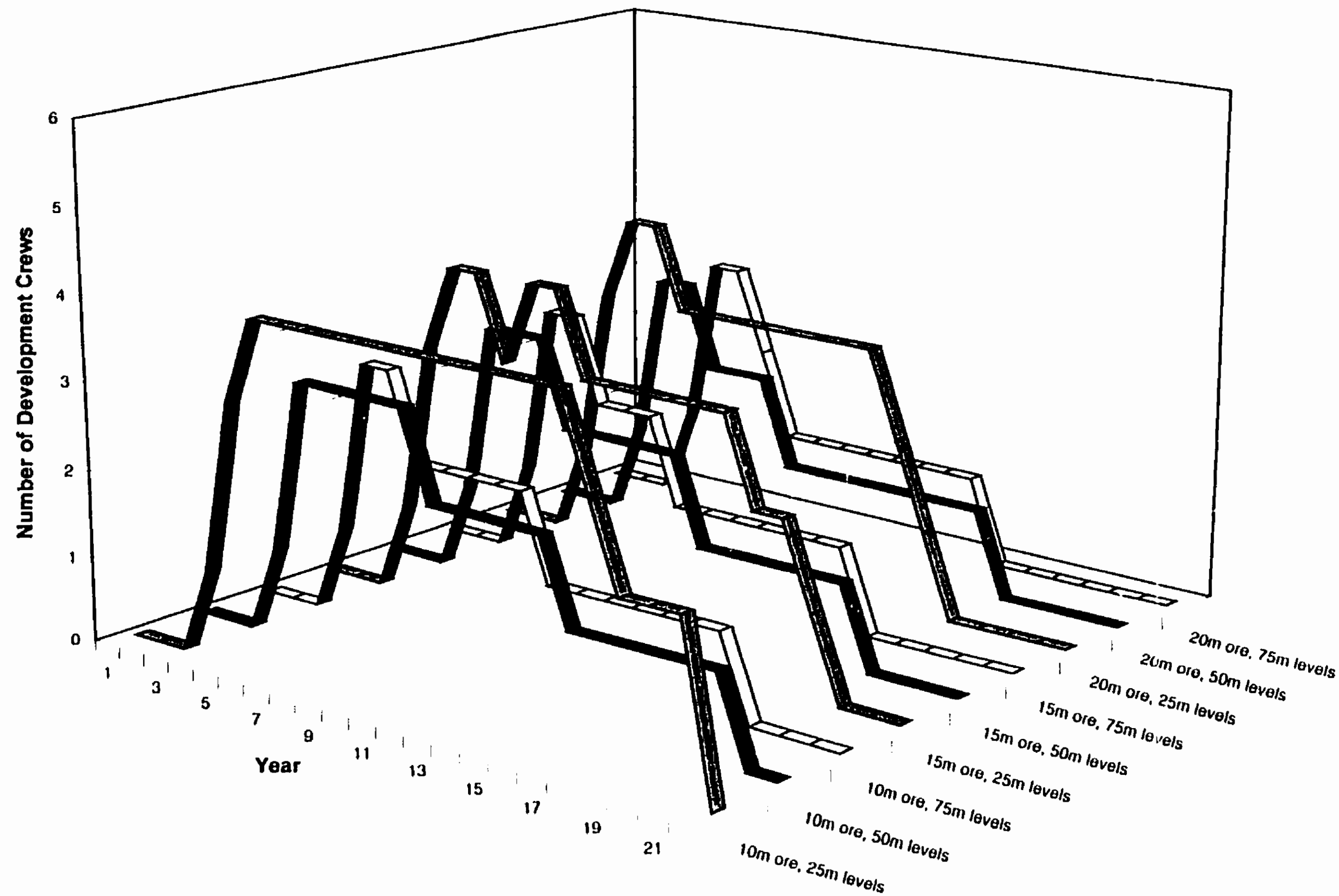

Figure 62: Fox River Project – Number of development crews – Small equipment

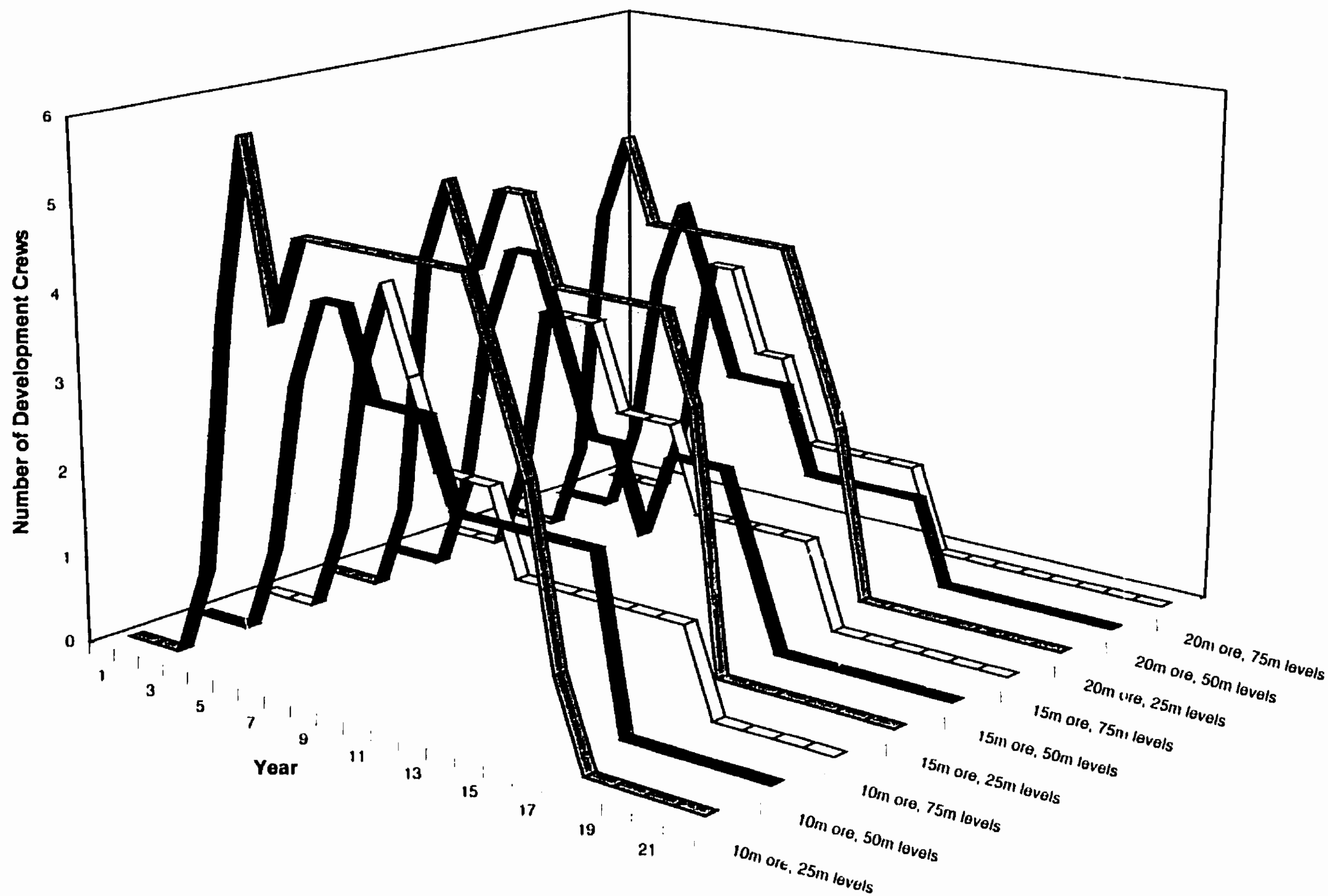

Figure 63: Fox River Project – Number of development crews – Mid-size equipment

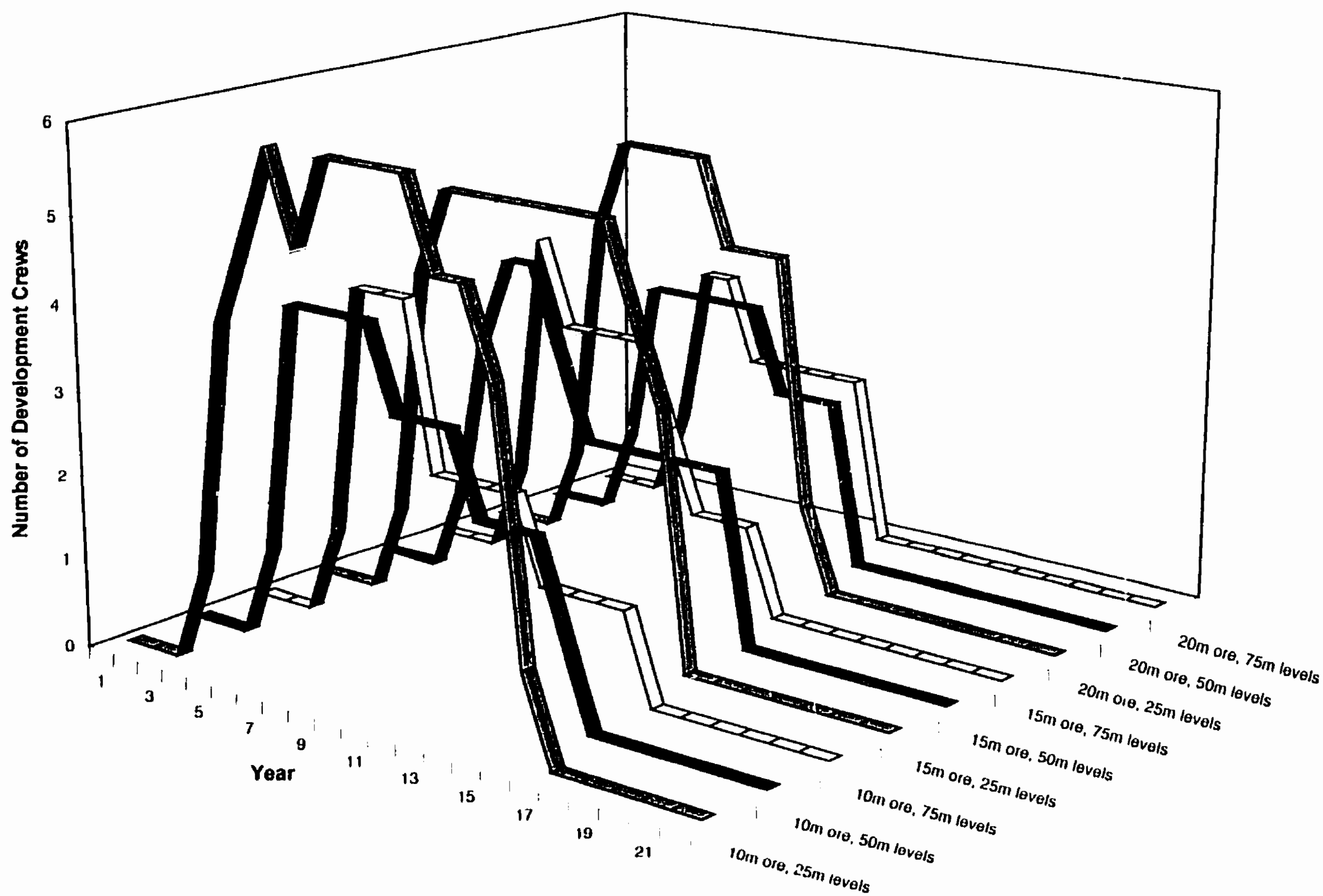

Figure 64: Fox River Project – Number of development crews – Large equipment

Table 39: Fox River Project – Ore production programs – Small equipment[137]

Thickness	Level	Source	Year																						Total
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	Stope	-	-	-	-	-	-	679	679	679	679	679	679	679	679	679	679	679	679	679	679	679	158	10,348
		Development	-	-	-	-	-	21	54	56	58	66	28	46	58	66	55	25	101	30	52	34	39	-	787
		Total	-	-	-	-	-	21	733	735	737	745	707	725	737	745	734	704	781	709	731	713	719	158	11,135
	50-m	Stope	-	-	-	-	-	-	721	721	721	721	721	721	721	721	721	721	721	721	721	721	721	384	11,194
		Development	-	-	-	-	-	17	33	34	15	41	25	9	59	21	29	36	8	20	8	24	14	-	394
		Total	-	-	-	-	-	17	754	754	736	762	745	730	780	742	750	757	728	741	729	744	734	384	11,588
	75-m	Stope	-	-	-	-	-	-	719	719	719	719	719	719	719	719	719	719	719	719	719	719	719	128	10,920
		Development	-	-	-	-	-	17	17	18	14	20	33	15	33	2	29	33	17	15	-	-	-	-	262
		Total	-	-	-	-	-	17	736	738	733	739	752	734	753	721	748	753	736	735	719	719	719	128	11,182
15-m	25-m	Stope	-	-	-	-	-	-	703	703	703	703	703	703	703	703	703	703	703	703	703	500	-	-	9,642
		Development	-	-	-	-	-	13	67	58	23	56	60	46	59	39	57	43	52	29	43	-	-	-	645
		Total	-	-	-	-	-	13	771	761	726	759	763	749	762	743	760	746	755	733	746	500	-	-	10,287
	50-m	Stope	-	-	-	-	-	-	726	726	726	726	726	726	726	726	726	726	726	726	726	726	645	-	10,803
		Development	-	-	-	-	-	15	33	12	20	26	37	28	37	34	15	33	18	3	11	-	-	-	322
		Total	-	-	-	-	-	15	759	738	745	752	762	753	762	760	741	759	744	728	737	726	645	-	11,126
	75-m	Stope	-	-	-	-	-	-	742	742	742	742	742	742	742	742	742	742	742	742	742	742	270	-	10,664
		Development	-	-	-	-	-	15	33	32	6	14	-	30	34	37	-	22	18	-	-	-	-	-	242
		Total	-	-	-	-	-	15	776	775	748	757	742	773	777	779	742	765	760	742	742	742	270	-	10,906
20-m	25-m	Stope	-	-	-	-	-	-	753	753	753	753	753	753	753	753	753	753	753	728	-	-	-	-	9,007
		Development	-	-	-	-	-	14	72	48	39	30	64	45	50	41	66	61	22	-	-	-	-	-	553
		Total	-	-	-	-	-	14	824	800	792	783	817	798	803	794	819	814	774	728	-	-	-	-	9,559
	50-m	Stope	-	-	-	-	-	-	740	740	740	740	740	740	740	740	740	740	740	740	740	740	136	-	10,492
		Development	-	-	-	-	-	14	39	39	-	19	12	17	26	31	35	-	24	20	-	-	-	-	276
		Total	-	-	-	-	-	14	779	779	740	759	752	757	766	771	774	740	764	760	740	740	136	-	10,768
	75-m	Stope	-	-	-	-	-	-	743	743	743	743	743	743	743	743	743	743	743	743	743	743	111	-	10,514
		Development	-	-	-	-	-	11	35	-	31	15	-	-	29	20	26	18	-	-	-	-	-	-	184
		Total	-	-	-	-	-	11	778	743	774	758	743	743	772	763	769	761	743	743	743	743	111	-	10,698

[137] All amounts in thousands of tonnes of ore.

Table 40: Fox River Project – Ore production programs – Mid-size equipment[138]

Thickness	Level	Source	Year																						Total
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	Stope	-	-	-	-	-	-	953	953	953	953	953	953	953	953	953	953	953	222	-	-	-	-	10,701
		Development	-	-	-	-	-	46	71	79	79	83	81	79	83	83	76	91	31	-	-	-	-	-	884
		Total	-	-	-	-	-	46	1,023	1,032	1,032	1,036	1,034	1,032	1,036	1,036	1,029	1,044	984	222	-	-	-	-	11,585
	50-m	Stope	-	-	-	-	-	-	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	536	-	-	-	-	11,588
		Development	-	-	-	-	-	19	38	37	38	39	60	46	16	27	78	43	-	-	-	-	-	-	442
		Total	-	-	-	-	-	19	1,043	1,042	1,043	1,044	1,065	1,051	1,021	1,031	1,082	1,048	1,005	536	-	-	-	-	12,030
	75-m	Stope	-	-	-	-	-	-	984	984	984	984	984	984	984	984	984	984	984	623	-	-	-	-	11,448
		Development	-	-	-	-	-	19	19	34	23	38	14	19	13	41	19	7	45	2	-	-	-	-	295
		Total	-	-	-	-	-	19	1,003	1,018	1,007	1,022	999	1,003	997	1,025	1,003	992	1,029	625	-	-	-	-	11,743
15-m	25-m	Stope	-	-	-	-	-	-	1,065	1,065	1,065	1,065	1,065	1,065	1,065	1,065	1,065	308	-	-	-	-	-	-	9,892
		Development	-	-	-	-	-	45	55	87	87	74	78	84	54	89	77	-	-	-	-	-	-	-	730
		Total	-	-	-	-	-	45	1,120	1,152	1,152	1,139	1,143	1,149	1,119	1,154	1,142	308	-	-	-	-	-	-	10,622
	50-m	Stope	-	-	-	-	-	-	1,014	1,014	1,014	1,014	1,014	1,014	1,014	1,014	1,014	1,014	913	-	-	-	-	-	11,054
		Development	-	-	-	-	-	17	38	42	42	37	23	38	63	42	21	-	-	-	-	-	-	-	365
		Total	-	-	-	-	-	17	1,053	1,056	1,056	1,051	1,037	1,053	1,077	1,056	1,035	1,014	913	-	-	-	-	-	11,419
	75-m	Stope	-	-	-	-	-	-	1,016	1,016	1,016	1,016	1,016	1,016	1,013	1,016	1,016	1,016	829	-	-	-	-	-	10,983
		Development	-	-	-	-	-	17	38	40	42	33	11	28	19	7	38	-	-	-	-	-	-	-	274
		Total	-	-	-	-	-	17	1,054	1,056	1,057	1,049	1,027	1,044	1,032	1,023	1,053	1,016	829	-	-	-	-	-	11,257
20-m	25-m	Stope	-	-	-	-	-	-	1,066	1,066	1,066	1,066	1,066	1,066	1,066	1,066	675	-	-	-	-	-	-	-	9,201
		Development	-	-	-	-	-	44	72	94	50	61	88	89	75	56	-	-	-	-	-	-	-	-	629
		Total	-	-	-	-	-	44	1,138	1,160	1,115	1,126	1,154	1,155	1,141	1,122	675	-	-	-	-	-	-	-	9,830
	50-m	Stope	-	-	-	-	-	-	1,056	1,056	1,056	1,056	1,056	1,056	1,056	1,056	1,056	1,056	106	-	-	-	-	-	10,666
		Development	-	-	-	-	-	16	45	44	19	50	41	44	17	39	-	-	-	-	-	-	-	-	315
		Total	-	-	-	-	-	16	1,101	1,100	1,075	1,106	1,097	1,100	1,073	1,095	1,056	1,056	106	-	-	-	-	-	10,981
	75-m	Stope	-	-	-	-	-	-	1,057	1,057	1,057	1,057	1,057	1,057	1,057	1,057	1,057	1,057	159	-	-	-	-	-	10,728
		Development	-	-	-	-	-	16	45	44	-	-	30	25	28	22	-	-	-	-	-	-	-	-	210
		Total	-	-	-	-	-	16	1,102	1,101	1,057	1,057	1,087	1,082	1,085	1,078	1,057	1,057	159	-	-	-	-	-	10,937

[138] All amounts in thousands of tonnes of ore.

Table 41: Fox River Project – Ore production programs – Large equipment[139]

Thickness	Level	Source	Year 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
10-m	25-m	Stope	-	-	-	-	-	-	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	847	-	-	-	-	-	-	11,243
		Development	-	-	-	-	-	26	119	103	127	109	111	117	100	109	122	37	-	-	-	-	-	-	1,079
		Total	-	-	-	-	-	26	1,274	1,258	1,282	1,264	1,266	1,272	1,256	1,264	1,277	884	-	-	-	-	-	-	12,322
	50-m	Stope	-	-	-	-	-	-	1,255	1,255	1,255	1,255	1,255	1,255	1,255	1,255	1,255	1,193	-	-	-	-	-	-	12,491
		Development	-	-	-	-	-	28	46	63	52	50	81	57	47	74	43	-	-	-	-	-	-	-	539
		Total	-	-	-	-	-	28	1,302	1,318	1,307	1,305	1,336	1,312	1,303	1,329	1,298	1,193	-	-	-	-	-	-	13,030
	75-m	Stope	-	-	-	-	-	-	1,345	1,345	1,345	1,345	1,345	1,345	1,345	1,345	1,345	688	-	-	-	-	-	-	12,794
		Development	-	-	-	-	-	26	47	21	43	45	47	45	24	24	38	-	-	-	-	-	-	-	360
		Total	-	-	-	-	-	26	1,392	1,366	1,388	1,390	1,392	1,390	1,369	1,369	1,383	688	-	-	-	-	-	-	13,154
15-m	25-m	Stope	-	-	-	-	-	-	1,115	1,115	1,115	1,115	1,115	1,115	1,115	1,115	1,115	173	-	-	-	-	-	-	10,210
		Development	-	-	-	-	-	45	137	52	100	111	92	116	62	134	40	-	-	-	-	-	-	-	888
		Total	-	-	-	-	-	45	1,252	1,167	1,215	1,226	1,207	1,231	1,177	1,249	1,156	173	-	-	-	-	-	-	11,099
	50-m	Stope	-	-	-	-	-	-	1,168	1,168	1,168	1,168	1,168	1,168	1,168	1,168	1,168	1,129	-	-	-	-	-	-	11,642
		Development	-	-	-	-	-	29	43	48	42	45	53	60	74	46	5	-	-	-	-	-	-	-	444
		Total	-	-	-	-	-	29	1,211	1,216	1,210	1,213	1,222	1,229	1,242	1,214	1,173	1,129	-	-	-	-	-	-	12,087
	75-m	Stope	-	-	-	-	-	-	1,404	1,404	1,404	1,404	1,404	1,404	1,404	1,404	639	-	-	-	-	-	-	-	11,874
		Development	-	-	-	-	-	22	47	82	52	40	36	-	41	12	-	-	-	-	-	-	-	-	333
		Total	-	-	-	-	-	22	1,452	1,487	1,456	1,445	1,440	1,404	1,446	1,417	639	-	-	-	-	-	-	-	12,208
20-m	25-m	Stope	-	-	-	-	-	-	1,348	1,348	1,348	1,348	1,348	1,348	1,303	-	-	-	-	-	-	-	-	-	9,394
		Development	-	-	-	-	-	13	149	116	105	120	119	119	25	-	-	-	-	-	-	-	-	-	766
		Total	-	-	-	-	-	13	1,497	1,465	1,453	1,468	1,468	1,467	1,329	-	-	-	-	-	-	-	-	-	10,160
	50-m	Stope	-	-	-	-	-	-	1,326	1,326	1,326	1,326	1,326	1,326	1,326	1,326	475	-	-	-	-	-	-	-	11,080
		Development		-	-	-	-	20	55	53	44	69	84	47	12	-	-	-	-	-	-	-	-	-	383
		Total	-	-	-	-	-	20	1,381	1,378	1,370	1,394	1,409	1,373	1,338	1,326	475	-	-	-	-	-	-	-	11,463
	75-m	Stope	-	-	-	-	-	-	1,394	1,394	1,394	1,394	1,394	1,394	1,394	1,394	225	-	-	-	-	-	-	-	11,375
		Development	-	-	-	-	-	34	47	46	7	51	51	18	-	-	-	-	-	-	-	-	-	-	255
		Total	-	-	-	-	-	34	1,441	1,440	1,400	1,445	1,445	1,412	1,394	1,394	225	-	-	-	-	-	-	-	11,631

[139] All amounts in thousands of tonnes of ore.

- ***Other pre-production development***

It is assumed that the early mine development of all scenarios is similar and follows the same schedule. This assumption is supported by the fact that, regardless of equipment size, orebody thickness, and inter-level spacing, initial mine development of every scenario complies with the following logical sequence:[140]

a. Sink production and return-air shafts.

b. Develop loading pocket level.

c. Drive loading pocket ramp.

d. Develop crusher station level.

e. Excavate ore and waste bins.

f. Excavate crusher station.

With the exception of level development, whose dimensions vary according to equipment size (see Table 34), all the above excavations have exactly the same specifications. Thus, it is safe to assume that construction time will be very similar in all scenarios, and the same schedule with a total duration of 44 months was used in all of them. Table 42, Table 43, and Table 44 list major pre-production mine development for small, mid-size, and large-equipment scenarios, respectively. It must be noted that, although the main production shaft has a diameter of six metres in every scenario, different diesel equipment requirements resulted in varying main ventilation shaft diameters. In fact, whereas a 3.5-metre shaft was needed in all small equipment scenarios, 4.0-metre and 5.5-metre ventilation shafts were required for mid-size and large-equipment cases. This highlights the iterative nature of scenario analysis: an initial estimate (see Section 6.5.1, Table 32, and Figure 42) indicated that about 1,000,000 cfm and a 6.0-metre ventilation shaft were required for ventilating Fox River.

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[140] The only major development task that is not included in the list is sinking a fresh-air ventilation shaft. This is because, as noted in Section 6.4.5, such a shaft is sunk to the top of the active mining area, i.e., it connects to the uppermost production level. Thus, since it must wait for such a level to be developed, the shaft is not part of the "*critical path*" of initial mine development.

Table 42: Fox River Project – Other pre-production development – Small equipment

	Length	Duration	Excavation/Construction Cost (US $)			Investment (US $)					Total (US $)
	m	months	total	per month	per year	Year 1	Year 2	Year 3	Year 4	Year 5	
Production Shaft (Diameter = 6.0 metres)	2,090	35	31,373,006	896,372	10,756,459	10,756,459	10,756,459	9,860,088	-	-	31,373,006
Return-Air Shaft	2,025	34	22,221,803	653,582	7,842,989	-	-	7,842,989	7,842,989	6,535,824	22,221,803
Fresh-Air Shaft 10-m ore	1,700	28	18,587,938	663,855	7,966,259	-	-	2,655,420	7,966,259	7,966,259	18,587,938
Fresh-Air Shaft 15-m ore	1,800	30	19,702,633	656,754	7,881,053	-	-	3,940,527	7,881,053	7,881,053	19,702,633
Fresh-Air Shaft 15-m ore, 75-m levels	1,775	30	19,423,662	647,455	7,769,465	-	-	3,884,732	7,769,465	7,769,465	19,423,662
Fresh-Air Shaft 20-m ore	1,850	31	20,261,153	653,586	7,843,027	-	-	4,575,099	7,843,027	7,843,027	20,261,153
Loading Pocket Level	204	2	295,395	147,697	N.A.	-	-	-	295,395	-	295,395
Loading Pocket Ramp	337	3	488,071	162,690	N.A.	-	-	-	488,071	-	488,071
Crusher Station Level	111	1	161,236	161,236	N.A.	-	-	-	161,236	-	161,236
Ore\Waste Bins	N.A.	2	182,100	91,050	N.A.	-	-	-	182,100	-	182,100
Crusher Station	N.A.	1	48,960	48,960	N.A.	-	-	-	48,960	-	48,960

Note: Return and fresh-air shafts have a 3.5-metre diameter.

Table 43: Fox River Project – Other pre-production development – Mid-size equipment

	Length	Duration	Excavation/Construction Cost (US $)			Investment (US $)					Total (US $)
	m	months	total	per month	per year	Year 1	Year 2	Year 3	Year 4	Year 5	
Production Shaft (Diameter = 6.0 metres)	2,090	35	32,575,215	930,720	11,168,645	11,168,645	11,168,645	10,237,925	-	-	32,575,215
Return-Air Shaft	2,025	34	24,382,699	717,138	8,605,658	-	-	8,605,658	8,605,658	7,171,382	24,382,699
Fresh-Air Shaft 10-m ore	1,700	28	20,392,789	728,314	8,739,767	-	-	2,913,256	8,739,767	8,739,767	20,392,789
Fresh-Air Shaft 15-m ore	1,800	30	21,616,701	720,557	8,646,680	-	-	4,323,340	8,646,680	8,646,680	21,616,701
Fresh-Air Shaft 15-m ore, 75-m levels	1,775	30	21,310,396	710,347	8,524,159	-	-	4,262,079	8,524,159	8,524,159	21,310,396
Fresh-Air Shaft 20-m ore	1,850	31	22,229,945	717,095	8,605,140	-	-	5,019,665	8,605,140	8,605,140	22,229,945
Loading Pocket Level	204	2	319,471	159,735	N.A.	-	-	-	319,471	-	319,471
Loading Pocket Ramp	337	3	527,851	175,950	N.A.	-	-	-	527,851	-	527,851
Crusher Station Level	111	1	174,378	174,378	N.A.	-	-	-	174,378	-	174,378
Ore\Waste Bins	N.A.	2	182,100	91,050	N.A.	-	-	-	182,100	-	182,100
Crusher Station	N.A.	1	48,960	48,960	N.A.	-	-	-	48,960	-	48,960

Note: Return and fresh-air shafts have a 4.0-metre diameter.

Table 44: Fox River Project – Other pre-production development - Large equipment

	Length	Duration	Excavation/Construction Cost (US $)			Investment (US $)					Total (US $)
	m	months	total	per month	per year	Year 1	Year 2	Year 3	Year 4	Year 5	
Production Shaft (Diameter = 6.0 metres)	2,090	35	33,758,380	964,525	11,574,302	11,574,302	11,574,302	10,609,777	-	-	33,758,380
Return-Air Shaft	2,025	34	30,424,785	894,847	10,738,159	-	-	10,738,159	10,738,159	8,948,466	30,424,785
Fresh-Air Shaft 10-m ore	1,700	28	25,438,524	908,519	10,902,225	-	-	3,634,075	10,902,225	10,902,225	25,438,524
Fresh-Air Shaft 15-m ore	1,800	30	26,968,069	898,936	10,787,227	-	-	5,393,614	10,787,227	10,787,227	26,968,069
Fresh-Air Shaft 15-m ore, 75-m levels	1,775	30	26,585,275	886,176	10,634,110	-	-	5,317,055	10,634,110	10,634,110	26,585,275
Fresh-Air Shaft 20-m ore	1,850	31	27,734,451	894,660	10,735,917	-	-	6,262,618	10,735,917	10,735,917	27,734,451
Loading Pocket Level	204	2	351,105	175,553	N.A.	-	-	-	351,105	-	351,105
Loading Pocket Ramp	337	3	580,119	193,373	N.A.	-	-	-	580,119	-	580,119
Crusher Station Level	111	1	191,645	191,645	N.A.	-	-	-	191,645	-	191,645
Ore\Waste Bins	N.A.	2	182,100	91,050	N.A.	-	-	-	182,100	-	182,100
Crusher Station	N.A.	1	48,960	48,960	N.A.	-	-	-	48,960	-	48,960

Note: Return and fresh-air shafts have a 5.5-metre diameter.

6.6 Operating Cost

In this case study, cost data are used to facilitate the evaluation of the benefits and drawbacks of every mining scenario from both economic and financial perspectives. It must be noted, however, that the objective of this section is **not** to obtain accurate cost items, but to provide a common basis for analysis. Furthermore, it is beyond the scope of this thesis to estimate the **total cost** of developing and carrying out every mining scenario. Only pertinent aspects of the mine development and mining processes will be considered, focusing on:

- major equipment-related costs (both capital and operational); and,
- personnel requirements.

Main sources of cost data are Western Mine Engineering (1996), Schumacher (1996), and the sponsors of this research project.[141] The operating cost of every major piece of equipment used for mine development and production was estimated from the following basic cost data:

- ***Operating Labour Cost***

  The effective hourly labour cost for all development and production activities was estimated assuming an average base hourly rate of CAN \$25.00, a WCB rate of 10.0% of the base hourly rate, fringe benefits equal to 65.0% of the base hourly rate, 5.0% of overtime work, and a 10.0% bonus. The calculation of such a cost (in 1996 US \$) is shown in Table 45.

- ***Major Operating Cost Components***

  Major components of the operating costs are: parts, maintenance labour, diesel fuel,[142] gasoline,[142] electric power,[142] lubricants,[142] and tires.[142] In order to maintain consistency and allow meaningful comparisons, the hourly rates corresponding to such cost components were obtained directly from Schumacher (1996) and Western Mine Engineering (1996).

- ***Supplies Cost***

  Includes supplies and consumables. Supplies are mostly used in activities such as drilling, blasting, ground support, and level upkeep.

[141] Cost data provided by the sponsors of this project were disguised in order to protect their confidential nature.

[142] Whenever applicable.

Table 45: Fox River Project – Estimation of general labour cost

	Labour Rate (US $ / hour)					
	Rate	W.C.B.	Fringe	O/T	Bonus	Total
Operating labour ($/man-hour)	16.67	1.67	10.83	0.83	1.67	31.67
Operating labour ($/man-shift)	133.33	13.33	86.67	6.67	13.33	253.33

6.6.1 Mine Development

Table 46, Table 47, Table 48, and Table 49 show capital and hourly operating costs for drilling, mucking, haulage, and ground support equipment, respectively. Operating costs for mine development were calculated using such data, together with the mine development programs elaborated in Section 6.5.4 and basic assumptions regarding consumables consumption, and work schedules. For instance, Table 50 and Table 51 show the per-metre cost of advancing main ramps with small and large equipment, respectively. On a more general level, Figure 65, Figure 66, and Figure 67 graphically illustrate how unit development cost (in US $ per metre of excavation) increases with the cross-sectional area of the opening in small, mid-size, and large scenarios, respectively. It can be seen that, even though the slope of the curves is remarkably similar, the cost of the excavation (expressed in US $/m^2) is significantly higher with larger equipment. This is in spite of the fact that it was assumed that larger equipment scenarios benefited from important economies of scale.

Based on the level-by-level horizontal development programs shown in Table 70 to Table 78 (see Appendix I), the corresponding yearly development programs and budgets were produced (see Table 52 to Table 57). Total (i.e., life-of-mine) development metres are very similar for scenarios having with the same combination of orebody thickness and inter-level spacing (see, for instance, Figure 57). However, and as direct consequence of higher per-metre rates, the tables also show that in such cases development cost increases by about 10.0% as equipment size increases from small to mid-size, and from mid-size to large.

Table 46: Fox River Project – Capital and operating costs – Drilling equipment

Equipment	Decription	Specifications	Unit Cost	Mining Scenario			Hourly Operating Costs (US $)						
			(US $)	Small	Medium	Large	Parts	Maint. Labour	Diesel Fuel	Electric Power	Lubricants	Tires	Total
Anfo Loader	Rubber-tired; articulated; diesel carrier	61-kw (82-hp) engine	191,500	1	1	1	3.86	2.78	1.78	-	1.18	0.37	9.97
Drifter	Hydraulic	32 mm to 43 mm diameter hole	42,500	1			0.86	0.85	-	-	0.24	-	1.95
Drifter	Hydraulic	38 mm to 51 mm diameter hole	53,200		1		1.07	1.06	-	-	0.30		2.43
Drifter	Hydraulic	up to 64 mm diameter hole	83,600			1	1.69	1.66	-	-	0.48	-	3.83
Drifter Feeds	45 mm - 51 mm diameter steel	4.0 m drill steel length, 3.7 m hole depth	28,500	1			0.57	0.57	-	-	0.16	-	1.30
Drifter Feeds	43 mm - 51 mm diameter steel	5.5 m drill steel length; 5.2 m hole depth	33,500		1	1	0.68	0.67	-	-	0.19	-	1.54
Drifter Feeds	Telescoping; 38 mm - 43 mm diameter steel	2.2 m - 4.0 m drill steel length; 3.8 m hole	48,400	1			0.98	0.96	-	-	0.28	-	2.22
Drifter Feeds	Telescoping; 43 mm - 51 mm diameter steel	3.1 m - 5.5 m drill steel length, 5.2 m hole	54,500		1	1	1.10	1.09	-	-	0.31	-	2.50
Jumbo Drill	1 boom; 4.7 m x 3.8 m max. face	41 kW; drills not included in price	295,000	1			4.76	4.21	-	2.30	1.69	0.10	13.06
Jumbo Drill	2 boom; 5.2 m x 7.8 m max. face	88 - 104 kW, drills not included in price	306,000		1		4.94	4.37	-	5.40	1.75	0.18	16.64
Jumbo Drill	2 boom; 6.3 m x 8.7 m max face	88 - 104 kW, drills not included in price	491,400			1	7.93	7.01	-	10.80	2.81	0.14	28.69
Longhole Drill	Rubber-tired; 45 mm - 76 mm diameter hole	Min. drift size: 3.3 m high x 2.4 m wide	376,300	1			6.07	5.37	-	2.39	2.15	0.10	16.08
Longhole Drill	Rubber-tired; 45 mm - 76 mm diameter hole	Min. drift size: 3.3 m high x 2.4 m wide	450,000		1		7.26	6.42	-	2.86	2.57	0.14	19.25
Longhole Drill	Rubber-tired; 45 mm - 76 mm diameter hole	Min. drift size: 3.3 m high x 2.4 m wide	550,000			1	8.87	7.85	-	3.49	3.14	0.18	23.54

Sources: Western Mine Engineering (1996), Schumacher (1996)

Table 47: Fox River Project – Capital and operating costs – Mucking equipment

Equipment	Decription	Specifications	Unit Cost	Mining Scenario			Hourly Operating Costs (US $)						
			(US $)	Small	Medium	Large	Parts	Maint. Labour	Diesel Fuel	Electric Power	Lubricants	Tires	Total
Loader	Diesel - hard rock LHD	2.7 m3; 104 kW engine	227,600	1			8.26	7.14	2.92	-	1.45	4.68	24.45
Loader	Diesel - hard rock LHD	4.6 m3; 138 kW engine	306,900		1		11.14	9.63	3.69	-	1.95	3.84	30.45
Loader	Diesel - hard rock LHD	5.4 m3; 172 kW engine	395,600			1	14.36	12.41	4.66	-	2.51	5.17	39.31

Sources: Western Mine Engineering (1996), Schumacher (1996)

Table 48: Fox River Project – Capital and operating costs – Haulage equipment

Equipment	Decription	Specifications	Unit Cost (US $)	Mining Scenario			Hourly Operating Costs (US $)						
				Small	Medium	Large	Parts	Maint. Labour	Diesel Fuel	Electric Power	Lubricants	Tires	Total
Truck	Diesel - rear dump; articulated	20-tonne (22-ton); 205-kw engine	338,800	1			6.84	4.93	5.61	-	2.23	2.88	22.49
Truck	Diesel - rear dump; articulated	27-tonne (30-ton); 207-kw engine	390,800		1		7.88	5.68	5.61	-	2.52	3.69	25.38
Truck	Diesel - rear dump; articulated	50-tonne (55-ton); 354-kw engine	498,800			1	10.06	7.24	9.83	-	3.35	5.44	35.92
Locomotive	Troley	13.6-tonne (15-ton)	215,000	1			4.33	7.58	-	6.28	1.23	-	19.42
Locomotive	Troley	18.0-tonne (20-ton)	275,000		1		5.54	9.70	-	8.37	1.57	-	25.18
Locomotive	Troley	22.7-tonne (25-ton)	342,000			1	6.90	12.06	-	10.04	1.95	-	30.95
Rail Car	Granby - side dump - still dumping	2.3-3.4 m3 - 14" wheels - 24-30" track	8,950				0.32	0.32	-	-	0.05	-	0.69
Rail Car	Granby - side dump - still dumping	3.4-4.5 m3 - 16" wheels - 30-36" track	10,810	1	1	1	0.39	0.39	-	-	0.06	-	0.84
Rail Car	Granby - side dump - still dumping	4.5-5.7 m3 - 16" wheels - 30-42 track	16,480				0.60	0.59	-	-	0.09	-	1.28

Sources Western Mine Engineering (1996), Schumacher (1996)

Table 49: Fox River Project – Capital and operating costs – Ground support equipment

Equipment	Decription	Specifications	Unit Cost (US $)	Mining Scenario			Hourly Operating Costs (US $)						
				Small	Medium	Large	Parts	Maint. Labour	Diesel Fuel	Electric Power	Lubricants	Tires	Total
Rock Bolter	Rubber-tired; articulated; diesel; one-boom	6.2 m - 7.0 m vertical reach, 2.4 m bolts	455,000	1			9.17	17.93	1.24	-	2.66	0.10	31.10
Rock Bolter	Rubber-tired; articulated; dieselr, one-boom	6.5 m - 7.3 m vertical reach, 2.4 m bolts	510,000		1		10.28	20.10	1.63	-	2.99	0.18	35.18
Rock Bolter	Rubber-tired; articulated; diesel; one-boom	8.5 m - 9.3 m vertical reach; 2.4 m bolts	580,000			1	11.69	22.86	1.63	-	3.39	0.18	39.75
Shotcreting	Truck mounted, remote controlled (truck n.i.)	3.8-19.1 m3 (5-25 yd3)/hr Robotic arm	43,900	1	1	1	0.89	0.64	4.55	-	0.47	0.26	6.81
Service Truck	Articulated, flat deck; 4.5-tonne; for shotcrete	61 kW (82 HP)	108,500	1	1	1	2.19	1.58	1.78	-	0.71	0.37	6.63
Scaler	Vehicle mounted; telescopic scaler	61 kW (82 HP)	284,000	1	1	1	5.73	4.12	1.78	-	1.71	0.37	13.71
Scissors lift	Rubber-tired; articulated; diesel carrier	61-kw (82-hp) engine	141,500	1	1	1	2.85	2.05	1.78	-	0.90	0.37	7.95

Sources Western Mine Engineering (1996), Schumacher (1996)

Table 50: Fox River Project – Development cost – 4.5 x 4.0 m ramp, small equipment

Cross-section (width x height)	4.5 m x 4.0 m		Crew size	5 miners
Average net advance rate	3.36 m/day			

	Rate	W.C.B.	Fringe	O/T	Bonus	Total
Operating labour ($/man-hour)	16.67	1.67	10.83	0.83	3.33	33.33
Operating labour ($/man-shift)	133.33	13.33	86.67	6.67	26.67	266.67
Operating labour cost ($/round)						1,428.57
Operating labour cost ($/metre)						396.83

Major Equipment	Jumbo	Bolter	Shotcrete	Scissors Lift	LHD	Truck	Train
Total operating cost (U.S. $/hour)	16.31	31.10	13.44	7.95	24.45	22.49	32.08
Hours of operation per round	4.00	2.00	1.50	1.00	6.50	-	1.00
Total operating cost (U.S. $/round)	65.24	62.20	20.16	7.95	158.93	-	32.08
Total operating cost (U.S. $/metre)	18.12	17.28	5.60	2.21	44.15	-	8.91

Total major equipment (U.S. $/metre)	96.27
Other service equipment @ 5% (U.S. $/metre)	4.81
Total equipment cost (U.S. $/metre)	101.08

Supplies	Unit Cost (U.S. $)		Unit/round		$/round	$/metre
Bolts, 1.8-m long, rebar	6.40	ea.	15	bolts	96.00	26.67
Bolts, 2.4-m long, rebar	8.17	ea.	29	bolts	236.83	65.79
Welded wire mesh, No. 6 galv.	12.33	/ sheet	10	sheets	123.33	34.26
Shotcrete	80.00	/ bag	10.00	bag	800.00	222.22
Anfo	1.25	/ kg	194.40	kg	242.86	67.46
Non-electric detonator c/w accessories	3.00	/ hole	38	holes	114.00	31.67
Button bit, 4.5-cm diameter	52.80	ea.	0.67	bits	35.39	9.83
Drilling rod, 4.2-m long, rope threaded	293.33	ea.	0.22	rods	65.54	18.21
Rigid ventilation duct, 0.91 m diameter	38.95	/ ft	11.81	feet	460.04	127.79
Water line, 5.0-cm diam., c/w fittings	1.47	/ ft	11.81	feet	17.32	4.81
Air line, 10.0-cm diam., c/w fittings	2.80	/ ft	11.81	feet	33.07	9.19
Drainage line, 5.0-cm diameter	1.47	/ ft	11.81	feet	17.32	4.81
Power cable, 4/0	9.50	/ ft	11.81	feet	112.20	31.17
Power cable, other	12.33	/ ft	11.81	feet	145.67	40.46
Switch	4,333.33	ea.	14.44	$/m	52.00	14.44
Water pump	1,500.00	ea.	4.29	$/m	15.43	4.29
Tools and miscellaneous materials	7.5%				192.53	53.48
Waste and/or rework	10.0%				275.95	76.65
Total supplies cost					3,035.50	843.20

	tonnes/m	$/ton	U.S. $/metre
Crushing	72.00	1.00	72.00
Hoisting	72.00	2.33	168.00

TOTAL DEVELOPMENT COST (U.S. $/metre)	1,581.10

Such an increase in development dollars is accompanied by a reduction in the development phase. For instance, it would take about 16 years to develop 15-m ore for mining with small equipment and 25-m levels. On the other hand, if large equipment were used, the deposit could be fully developed in about 11 years. However, the corresponding development budget would jump from US $79 million to US $93 million (about 18% higher). As this effect is coupled with a diminished ability to control dilution (see Table 36) it is evident that it has a significant impact on the economics of a scenario.

Table 51: Fox River Project – Development cost – 4.5 x 4.0 m ramp, large equipment

Cross-section (width x height)	5.3 m x 4.2 m	Crew size	5 miners
Average net advance rate	3.36 m/day		

	Rate	W.C.B.	Fringe	O/T	Bonus	Total
Operating labour ($/man-hour)	16.67	1.67	10.83	0.83	3.33	33.33
Operating labour ($/man-shift)	133.33	13.33	86.67	6.67	26.67	266.67
Operating labour cost ($/round)						1,428.57
Operating labour cost ($/metre)						396.83

Major Equipment	Jumbo	Bolter	Shotcrete	Scissors Lift	LHD	Truck	Train
Total operating cost (U.S. $/hour)	34.06	39.75	13.44	7.95	39.31	35.92	51.49
Hours of operation per round	2.50	2.00	1.50	2.00	3.50	-	0.50
Total operating cost (U.S. $/round)	85.15	79.50	20.16	15.90	137.59	-	25.75
Total operating cost (U.S. $/metre)	23.65	22.08	5.60	4.42	38.22	-	7.15

Total major equipment (U.S. $/metre)	101.12
Other service equipment @ 5% (U.S. $/metre)	5.06
Total equipment cost (U.S. $/metre)	106.18

Supplies	Unit Cost (U.S. $)		Unit/round		$/round	$/metre
Bolts, 1.8-m long, rebar	6.40	ea.	16	bolts	102.40	28.44
Bolts, 2.4-m long, rebar	8.17	ea.	32	bolts	261.33	72.59
Welded wire mesh, No. 6 galv.	12.33	/ sheet	11	sheets	135.67	37.69
Shotcrete	80.00	/ bag	11.00	bag	880.00	244.44
Anfo	1.25	/ kg	336.57	kg	420.47	116.80
Non-electric detonator c/w accessories	3.00	/ hole	44	holes	132.00	36.67
Button bit, 5.5-cm diameter	62.40	ea.	0.77	bits	47.80	13.28
Drilling rod, 4.2-m long, rope threaded	293.33	ea.	0.26	rods	74.91	20.81
Rigid ventilation duct, 1.21 m diameter	56.17	/ ft	11.81	feet	663.43	184.28
Water line, 5.0-cm diam., c/w fittings	1.47	/ ft	11.81	feet	17.32	4.81
Air line, 10.0-cm diam., c/w fittings	2.80	/ ft	11.81	feet	33.07	9.19
Drainage line, 5.0-cm diameter	1.47	/ ft	11.81	feet	17.32	4.81
Power cable, 4/0	9.50	/ ft	11.81	feet	112.20	31.17
Power cable, other	12.33	/ ft	11.81	feet	145.67	40.46
Switch	4,333.33	ea.	14.44	$/m	52.00	14.44
Water pump	1,500.00	ea.	4.29	$/m	15.43	4.29
Tools and miscellaneous materials	7.5%				233.33	64.81
Waste and/or rework	10.0%				334.44	92.90
Total supplies cost					3,678.79	1,021.89

	tonnes/m	$/ton	U.S. $/metre
Crushing	89.04	1.00	89.04
Hoisting	89.04	2.33	207.76

TOTAL DEVELOPMENT COST (U.S. $/metre)	1,821.69

6.6.2 Mining

Daily production rates are similar for scenarios using the same equipment size. Thus, both to estimate equipment requirements and compute yearly mining operating cost the average values were used: 2,100 tonnes/day, 3,000 tonnes/day, and 3,750 tonnes/day for small, mid-size and large equipment, respectively.

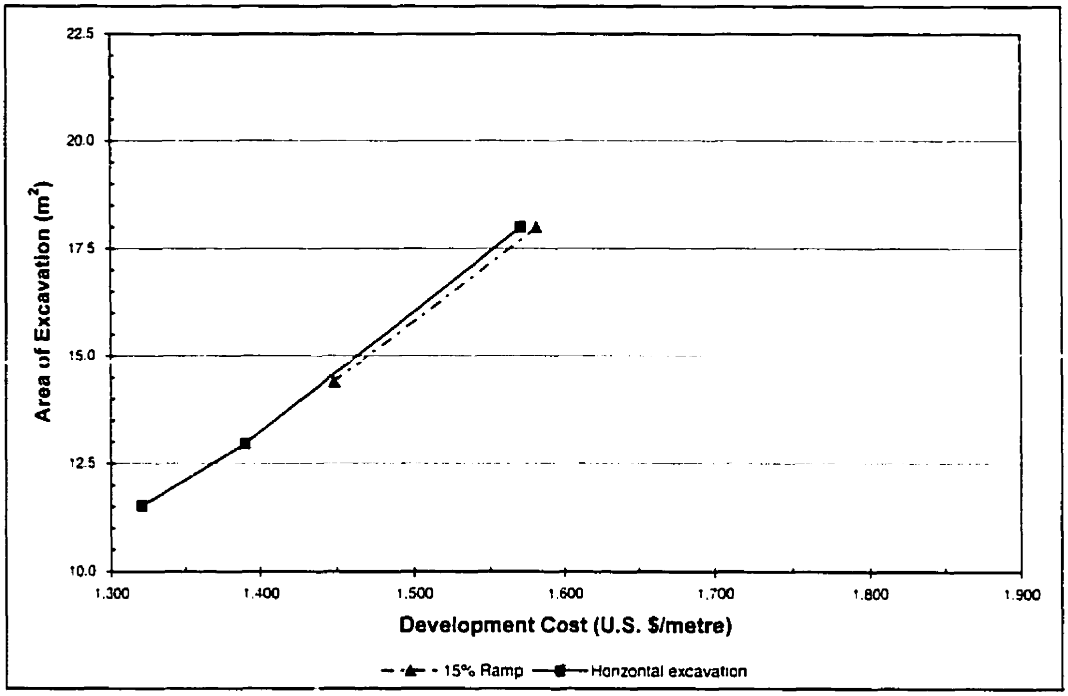

Figure 65: Fox River Project – Development cost - Small equipment

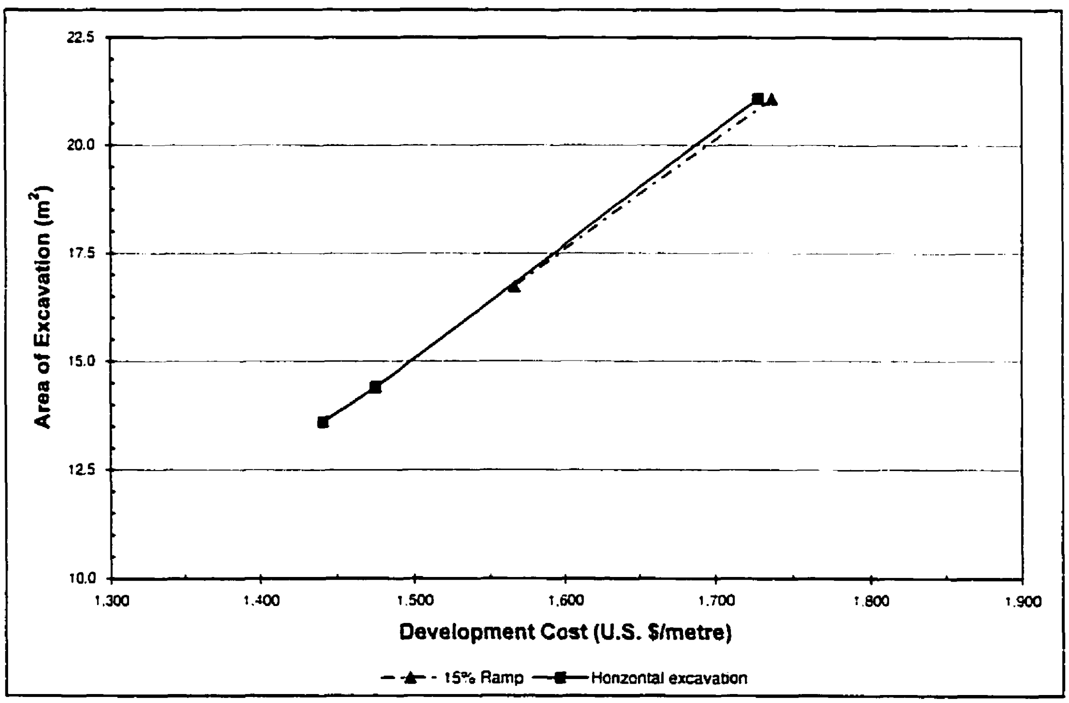

Figure 66: Fox River Project – Development cost – Mid-size equipment

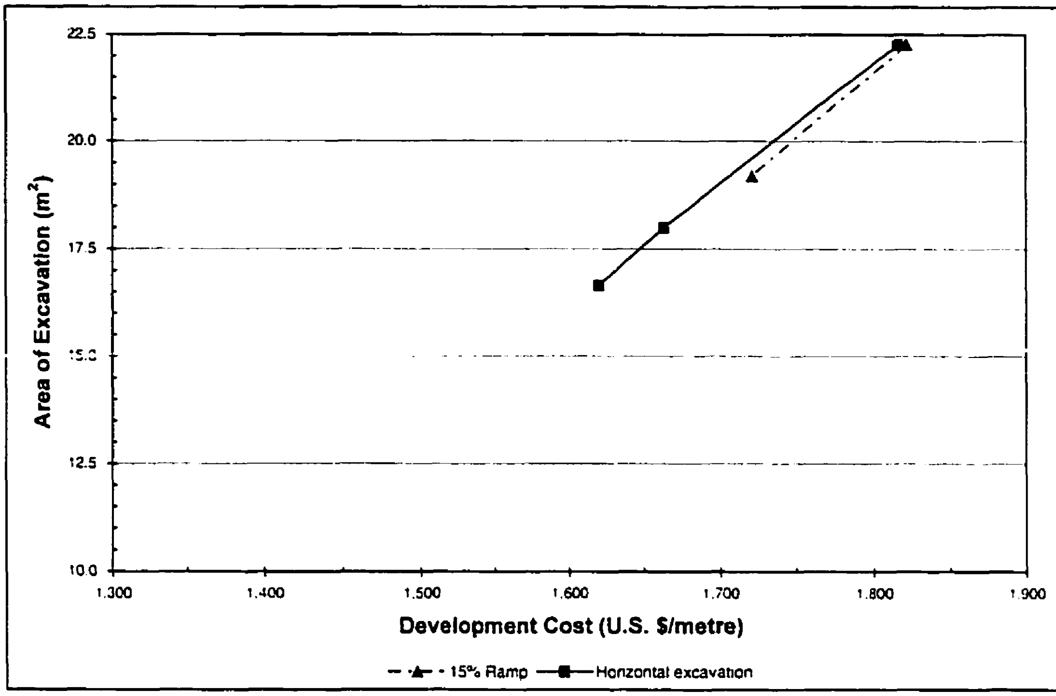

Figure 67: Fox River Project – Development cost – Large equipment

A procedure similar to that used for mine development was followed to estimate mining operating costs. Table 58, Table 59, and Table 60 show hourly and yearly operating costs for major pieces of mining equipment. It must be noted that truck and LHD requirements were calculated using actual production cycle data. Also, since there are five active stopes at any time (see Section 6.5.4) five longhole drills are required at all times in every scenario. Only one locomotive is needed by the operation, but the criticality of its operation makes it imperative that a stand-by unit is maintained at all times. Finally, one Anfo loader and two scissors lifts were included in every scenario to support stope operation.

6.6.3 Hoisting and Ventilation

As in the case of mining, hoisting and ventilation requirements were determined using average mining rates. Furthermore, in the case of hoisting average development waste production rates were used: 240 tonnes/day, 350 tonnes/day, and 450 tonnes/day for small, mid-size, and large equipment scenarios, respectively.

Table 52: Fox River Project – Horizontal development program – Small equipment

Ore Width	Level	Yearly Horizontal Development (metres)																						Total
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	-	-	-	806	4,051	4,539	4,605	4,667	4,684	4,481	4,687	4,600	4,623	4,788	4,478	4,748	4,724	3,475	2,368	2,399	1,885	-	70,607
	50-m	-	-	-	806	4,185	3,952	3,563	3,494	3,625	2,731	2,469	2,366	2,317	2,421	1,709	1,158	1,155	1,168	1,210	1,155	301	-	39,784
	75-m	-	-	-	806	4,152	3,785	2,419	2,308	2,285	2,420	2,314	1,412	1,210	1,206	1,066	1,210	1,210	1,053	-	-	-	-	28,856
15-m	25-m	-	-	-	806	4,018	4,632	4,356	4,142	4,604	4,776	4,193	3,493	3,555	3,551	3,563	3,551	2,522	2,400	1,752	-	-	-	55,914
	50-m	-	-	-	806	3,857	3,811	3,629	2,579	2,314	2,420	2,376	2,318	1,714	1,157	1,177	1,210	1,210	1,101	215	-	-	-	31,894
	75-m	-	-	-	806	3,950	3,683	2,824	2,314	2,371	1,806	1,134	1,210	1,167	1,210	1,210	1,167	552	-	-	-	-	-	25,403
20-m	25-m	-	-	-	806	3,816	4,706	4,532	3,493	3,460	3,507	3,525	3,585	3,617	3,604	3,112	2,415	1,618	114	-	-	-	-	45,913
	50-m	-	-	-	806	4,084	3,448	2,419	2,411	2,419	1,724	1,210	1,210	1,215	1,216	1,210	1,210	1,215	601	-	-	-	-	26,397
	75-m	-	-	-	806	3,219	3,405	2,320	1,192	1,223	1,206	1,209	1,168	1,220	1,210	1,215	500	-	-	-	-	-	-	19,892

Table 53: Fox River Project – Yearly development cost – Small equipment

Ore Width	Level	Yearly Development Cost (thousands of US $)																						Total
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	-	-	-	1,266	6,318	6,522	6,757	6,687	6,733	6,289	6,917	6,766	6,557	6,885	6,296	7,062	6,481	5,082	3,204	3,286	2,559	-	101,667
	50-m	-	-	-	1,266	6,578	5,801	5,190	5,078	5,423	3,929	3,608	3,415	3,245	3,511	2,338	1,565	1,578	1,602	1,646	1,564	415	-	57,750
	75-m	-	-	-	1,266	6,528	5,560	3,507	3,380	3,491	3,490	3,266	2,148	1,626	1,642	1,477	1,626	1,640	1,439	-	-	-	-	42,085
15-m	25-m	-	-	-	1,266	6,292	6,666	6,076	5,814	6,894	6,932	5,788	5,059	4,851	5,114	4,948	4,999	3,436	3,227	2,386	-	-	-	79,747
	50-m	-	-	-	1,266	6,062	5,688	5,130	3,856	3,389	3,303	3,408	3,280	2,305	1,564	1,611	1,619	1,626	1,497	299	-	-	-	45,903
	75-m	-	-	-	1,266	6,209	5,358	3,884	3,352	3,579	2,680	1,562	1,612	1,598	1,626	1,640	1,570	745	-	-	-	-	-	36,681
20-m	25-m	-	-	-	1,266	6,001	6,709	6,430	4,933	4,911	5,167	4,841	5,094	5,024	5,078	4,168	3,247	2,201	159	-	-	-	-	65,228
	50-m	-	-	-	1,266	6,420	4,981	3,251	3,380	3,799	2,518	1,626	1,633	1,634	1,648	1,626	1,640	1,634	808	-	-	-	-	37,863
	75-m	-	-	-	1,266	5,060	5,100	3,147	1,595	1,638	1,809	1,899	1,633	1,659	1,626	1,634	675	-	-	-	-	-	-	28,741

Table 54: Fox River Project – Horizontal development program – Mid-size equipment

Ore Width	Level	Yearly Horizontal Development (metres)																						Total
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	-	-	-	1,210	5,250	6,658	5,328	5,811	5,606	5,811	5,912	5,654	5,808	6,046	5,256	4,147	1,710	-	-	-	-	-	70,204
	50-m	-	-	-	806	4,179	4,658	4,616	3,971	3,458	3,462	2,798	2,352	2,230	2,300	2,378	2,230	134	-	-	-	-	-	39,573
	75-m	-	-	-	806	4,145	4,390	3,506	2,729	2,345	2,354	1,446	1,210	1,139	1,091	1,220	1,139	1,158	34	-	-	-	-	28,713
15-m	25-m	-	-	-	1,210	5,116	5,667	5,316	5,755	5,795	4,786	4,721	4,756	4,655	4,518	3,286	-	-	-	-	-	-	-	55,579
	50-m	-	-	-	806	4,179	4,592	4,834	3,136	2,363	2,347	1,558	2,363	2,350	2,218	974	-	-	-	-	-	-	-	31,718
	75-m	-	-	-	806	3,944	3,683	3,530	2,596	2,419	2,290	1,400	1,211	1,210	1,147	1,036	-	-	-	-	-	-	-	25,272
20-m	25-m	-	-	-	1,210	5,184	5,916	4,852	4,311	4,473	4,449	4,705	4,381	4,003	2,179	-	-	-	-	-	-	-	-	45,662
	50-m	-	-	-	806	4,179	4,355	3,615	2,943	2,381	2,304	1,192	1,203	1,210	1,186	893	-	-	-	-	-	-	-	26,267
	75-m	-	-	-	806	4,045	3,582	2,419	2,338	1,205	1,201	1,220	1,210	1,195	581	-	-	-	-	-	-	-	-	19,801

Table 55: Fox River Project – Yearly development cost – Mid-size equipment

Ore Width	Level	Yearly Development Cost (thousands of US $)																						Total
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	-	-	-	2,090	9,001	10,507	8,454	9,059	8,701	9,001	9,143	8,814	8,993	9,367	8,082	6,130	2,513	-	-	-	-	-	109,855
	50-m	-	-	-	1,393	7,224	7,552	7,351	6,378	5,502	5,440	4,240	3,508	3,584	3,394	3,489	3,257	198	-	-	-	-	-	62,509
	75-m	-	-	-	1,393	7,168	7,117	5,596	4,428	3,555	3,704	2,425	1,764	1,660	1,591	1,806	1,660	1,690	50	-	-	-	-	45,605
15-m	25-m	-	-	-	2,090	8,816	8,715	8,413	8,930	8,941	7,315	7,208	7,233	7,190	6,596	4,789	-	-	-	-	-	-	-	86,235
	50-m	-	-	-	1,393	7,224	7,430	7,681	4,904	3,439	3,688	2,440	3,435	3,445	3,230	1,416	-	-	-	-	-	-	-	49,725
	75-m	-	-	-	1,393	6,818	5,858	5,432	4,057	3,520	3,545	2,370	1,838	1,760	1,669	1,507	-	-	-	-	-	-	-	39,768
20-m	25-m	-	-	-	2,090	8,934	8,984	7,782	6,346	6,874	6,918	7,081	6,574	5,827	3,184	-	-	-	-	-	-	-	-	70,595
	50-m	-	-	-	1,393	7,224	7,015	5,580	4,571	3,664	3,347	1,728	1,747	1,753	1,729	1,300	-	-	-	-	-	-	-	41,051
	75-m	-	-	-	1,393	6,964	5,679	3,513	3,555	2,082	1,879	1,803	1,757	1,736	844	-	-	-	-	-	-	-	-	31,204

Table 56: Fox River Project – Horizontal development program – Large equipment

Ore Width	Level	Yearly Horizontal Development (metres)																						Total
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	-	-	-	806	4,537	5,966	6,728	6,627	6,996	6,779	7,073	6,724	5,777	5,785	4,669	1,234	-	-	-	-	-	-	69,701
	50-m	-	-	-	806	4,268	4,794	4,696	4,719	3,750	3,649	3,573	2,848	2,339	2,281	1,586	-	-	-	-	-	-	-	39,307
	75-m	-	-	-	806	4,336	4,794	4,591	2,331	2,430	2,382	2,368	1,296	1,210	1,220	769	-	-	-	-	-	-	-	28,531
15-m	25-m	-	-	-	806	4,403	5,802	5,972	5,805	5,938	5,729	4,777	4,728	4,706	4,764	1,730	-	-	-	-	-	-	-	55,160
	50-m	-	-	-	806	3,945	4,613	4,829	3,383	2,363	2,388	2,375	2,341	2,322	2,064	69	-	-	-	-	-	-	-	31,497
	75-m	-	-	-	806	4,336	4,189	3,581	3,434	3,027	2,088	1,210	1,122	1,144	170	-	-	-	-	-	-	-	-	25,106
20-m	25-m	-	-	-	1,210	4,993	5,879	5,920	5,943	5,920	5,246	4,769	4,237	1,230	-	-	-	-	-	-	-	-	-	45,348
	50-m	-	-	-	806	4,066	3,549	3,586	3,531	3,169	2,390	2,370	2,394	240	-	-	-	-	-	-	-	-	-	26,101
	75-m	-	-	-	806	3,256	3,652	2,522	2,353	2,379	2,386	1,977	354	-	-	-	-	-	-	-	-	-	-	19,685

Table 57: Fox River Project – Yearly development cost – Large equipment

Ore Width	Level	Yearly Development Cost (thousands of US $)																						Total
		1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
10-m	25-m	-	-	-	1,464	8,109	10,407	11,570	11,340	11,884	11,586	12,123	11,470	9,807	9,763	7,669	2,043	-	-	-	-	-	-	119,236
	50-m	-	-	-	1,464	7,754	8,393	8,115	8,112	6,490	6,219	6,074	4,787	3,859	3,743	2,605	-	-	-	-	-	-	-	67,614
	75-m	-	-	-	1,464	7,858	8,383	7,946	4,115	4,048	4,133	3,895	2,145	1,986	2,003	1,263	-	-	-	-	-	-	-	49,238
15-m	25-m	-	-	-	1,464	7,941	9,905	10,099	10,026	10,161	9,979	9,981	10,018	8,471	5,637	115	-	-	-	-	-	-	-	93,798
	50-m	-	-	-	1,464	7,167	8,082	8,320	5,759	3,906	4,121	3,930	3,848	3,795	3,381	115	-	-	-	-	-	-	-	53,891
	75-m	-	-	-	1,464	7,858	7,273	6,055	5,672	5,172	3,590	1,981	1,842	1,861	283	-	-	-	-	-	-	-	-	43,052
20-m	25-m	-	-	-	2,197	9,036	10,127	9,984	9,966	10,040	8,766	7,850	6,940	2,017	-	-	-	-	-	-	-	-	-	76,921
	50-m	-	-	-	1,464	7,387	6,101	6,107	6,048	5,346	3,926	3,873	3,908	398		-	-	-	-	-	-	-	-	44,561
	75-m	-	-	-	1,464	5,501	6,055	4,048	3,880	4,270	3,791	3,231	581	-	-	-	-	-	-	-	-	-	-	32,822

Table 58: Fox River Project – Mining operating cost – Small equipment

Equipment	Hourly Cost US $ / hour					Operating Hours per Unit				Operating Life		Number of Operating Units	Total Number of Units	Yearly Cost	
	Parts	Operating	Labour		Total	Availability	hours/shift	hours/day	hours/year	hours	years			US $/Unit	Total US $
			Maintenance	Operating											
Longhole Drill	6.17	4.54	5.37	31.67	47.75	75.0%	5.42	10.83	3,900.00	25,000	6	5	5	186,212	931,060
Anfo Loader	4.23	2.96	2.78	31.67	41.64	85.0%	5.42	10.83	3,900.00	30,000	8	1	1	162,383	162,383
LHD	12.94	4.37	7.14	31.67	56.12	80.0%	5.42	10.83	3,465.00	20,000	6	1	1	194,444	194,444
Truck	9.72	7.84	4.93	31.67	54.16	80.0%	5.42	10.83	3,862.80	25,000	6	1.4	2	209,196	296,833
Scissors Lift	3.22	2.68	2.05	31.67	39.62	85.0%	5.42	10.83	3,900.00	30,000	8	2	2	154,505	309,010
Locomotive	4.33	7.51	7.58	31.67	51.09	N.A.	6.00	12.00	4,320.00	35,000	8	1	2	220,694	220,694

Table 59: Fox River Project – Mining operating cost – Mid-size equipment

Equipment	Hourly Cost US $ / hour					Operating Hours per Unit				Operating Life		Number of Operating Units	Total Number of Units	Yearly Cost	
	Parts	Operating	Labour		Total	Availability	hours/shift	hours/day	hours/year	hours	years			US $/Unit	Total US $
			Maintenance	Operating											
Longhole Drill	9.54	6.03	8.54	31.67	55.78	75.0%	5.42	10.83	3,900.00	25,000	6	5	5	217,528	1,087,640
Anfo Loader	4.23	2.96	2.78	31.67	41.64	85.0%	5.42	10.83	3,900.00	30,000	8	1	1	162,383	162,383
LHD	14.98	5.84	9.63	31.67	62.12	80.0%	5.42	10.83	3,629.96	20,000	6	1	1	225,481	225,481
Truck	11.57	8.13	5.68	31.67	57.05	80.0%	5.42	10.83	3,888.00	25,000	6	1.5	2	221,797	338,854
Scissors Lift	3.22	2.68	2.05	31.67	39.62	85.0%	5.42	10.83	3,900.00	30,000	8	2	2	154,505	309,010
Locomotive	5.54	9.94	9.70	31.67	56.85	N.A.	6.00	12.00	4,320.00	35,000	8	1	2	245,578	245,578

Table 60: Fox River Project – Mining operating cost – Large equipment

Equipment	Hourly Cost US $ / hour					Operating Hours per Unit				Operating Life		Number of Operating Units	Total Number of Units	Yearly Cost	
	Parts	Operating	Labour		Total	Availability	hours/shift	hours/day	hours/year	hours	years			US $/Unit	Total US $
			Maintenance	Operating											
Longhole Drill	12.43	7.60	11.17	31.67	62.86	75.0%	5.42	10.83	3,900.00	25,000	6	5	5	245,166	1,225,829
Anfo Loader	4.23	2.96	2.78	31.67	41.64	85.0%	5.42	10.83	3,900.00	30,000	8	1	1	162,383	162,383
LHD	19.53	7.37	12.41	31.67	70.98	80.0%	5.42	10.83	2,475.00	20,000	8	1	1	175,667	175,667
Truck	15.50	13.18	7.24	31.67	67.59	80.0%	5.42	10.83	3,917.45	25,000	6	1.0	1	264,768	261,284
Scissors Lift	3.22	2.68	2.05	31.67	39.62	85.0%	5.42	10.83	3,900.00	30,000	8	2	2	154,505	309,010
Locomotive	6.90	11.99	12.06	31.67	62.62	N.A.	6.00	12.00	4,320.00	35,000	8	1	2	270,504	270,504

Hoisting calculations were carried out using the method described in Section 5.3. For example, Table 61 shows hoisting calculations for mid-size equipment. Based on hoist power requirements, hoisting operating costs for every equipment size were calculated as shown in Table 62.

Preliminary ventilation requirements were determined in Section 6.5.1 to provide initial estimates of ventilation shaft dimensions. At this stage, with more accurate estimates of diesel equipment usage, it is possible to refine such requirements. The results are shown in Figure 68, Figure 69, and Figure 70.

Table 61: Fox River Project – Hoisting system design – Mid-size equipment

	Ore	Waste	Other	Total
Daily Tonnage	**3,000**	**350**	**-**	**3,350**

Hoisting depth	2,075	metres	6,808	feet
Headframe height (sheaves)	49	metres	160	feet
Dumping height (from surface)	40	metres	131	feet
Total hoisting distance	**2,115**	**metres**	**6,939**	**feet**
Shifts (skipping)	2	per day		
Shift duration	12	hours		
Availability for ore/waste skipping	80.00	%		
Time available for ore/waste skipping	19.20	hours/day		
Hoist utilization	95.00	%		
Actual skipping time	18.24	hours/day		
Hourly Tonnage	**184**	**tonnes/hour**	**202**	**tons/hour**
Rope Specifications (flattened strand, fibre cored 6 x 27 UHT)				
Diameter	5.08	cm	2.00	inches
Breaking strength	189,948	kg	418,765	lbs
Weight	10.05	kg/metre	6.76	lbs/ft
Drum diameter	**4.1**	**metres**	**13.33**	**feet**
Drum face width, single layer of rope	**9.2**	**metres**	**30.21**	**feet**
Drum face width, two layers of rope	**2.6**	**metres**	**8.53**	**feet**

Skipping Cycle - Maximum speed	**15.0**	**m/sec**	**16.5**	**m/sec**
- Acceleration	**0.75**	**m/sec2**	**0.75**	**m/sec2**
	time (sec)	distance (m)	time (sec)	distance (m)
Accelerate to 0.5 m/sec	1.0	0.3	1.0	0.3
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Accelerate to maximum speed	19.6	149.9	21.6	181.4
Run at full speed	120.6	1,809.6	105.9	1,746.6
Decelerate to creep speed	19.6	149.9	21.6	181.4
Creep @ 0.5 m/sec	5.0	2.5	5.0	2.5
Decelerate to stop	1.0	0.3	1.0	0.3
Load/Dump	15.0	-	15.0	-
Total	**186.8**	**2,115.0**	**176.1**	**2,115.0**

Skip load	**9.5**	**tonnes**	**9.0**	**tonnes**
Skip load	10.5	tons	9.9	tons
Safety Factor at loading pocket	**5.16**		**5.28**	
Motor Power Requirements				
Self-ventilated d-c motor rms Power	**3,288**	**kW**	**3,691**	**kW**
Induction (a-c) motor rms Power	**3,443**	**kW**	**3,896**	**kW**

It is quite evident that, apart from large equipment scenarios, the initial estimate of about 1,000,000 cfm and 6.0-metre diameter shaft was too generous. The resulting fan power and yearly operating costs are 738 kW and US $420,712, 885 kW and US $511,485, and 1,035 kW and US $598,548 for small, mid-size, and large equipment scenarios, respectively.

Table 62: Fox River Project – Hoisting operating cost

	Operating Cost (US $/hour)						Total	
	Parts	Maintenance Labour	Electric Power	Lubricants	Operating Labour	Total	(US $/tonne)	(US $/day)
1,990-kW Service hoist	20.15	14.52	100.63	5.71	34.52	175.53	[redacted]	1,755.32
2,455-kW Production hoist	25.71	18.52	128.36	7.28	34.52	214.38	1.66	3,910.35
3,443-kW Production hoist	37.50	27.01	187.21	10.62	34.52	296.85	1.62	5,414.59
4,240-kW Production hoist	47.00	33.85	234.65	13.31	34.52	363.32	1.58	6,627.02

Source: Western Mine Engineering (1996)

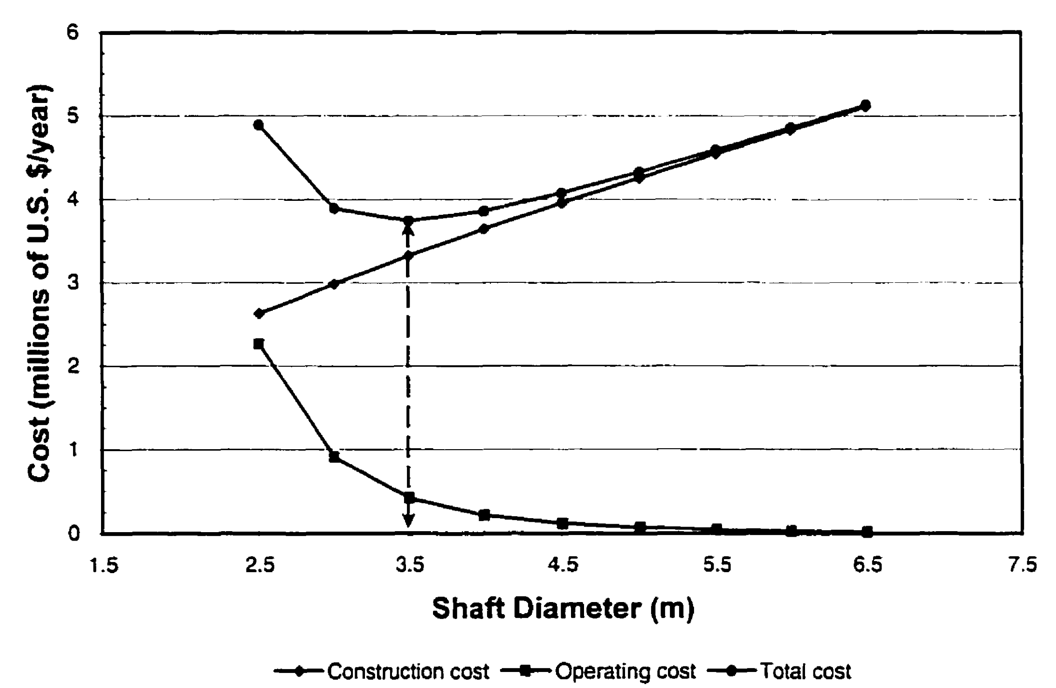

Figure 68: Fox River Project – Optimum ventilation shaft diameter – Small equipment

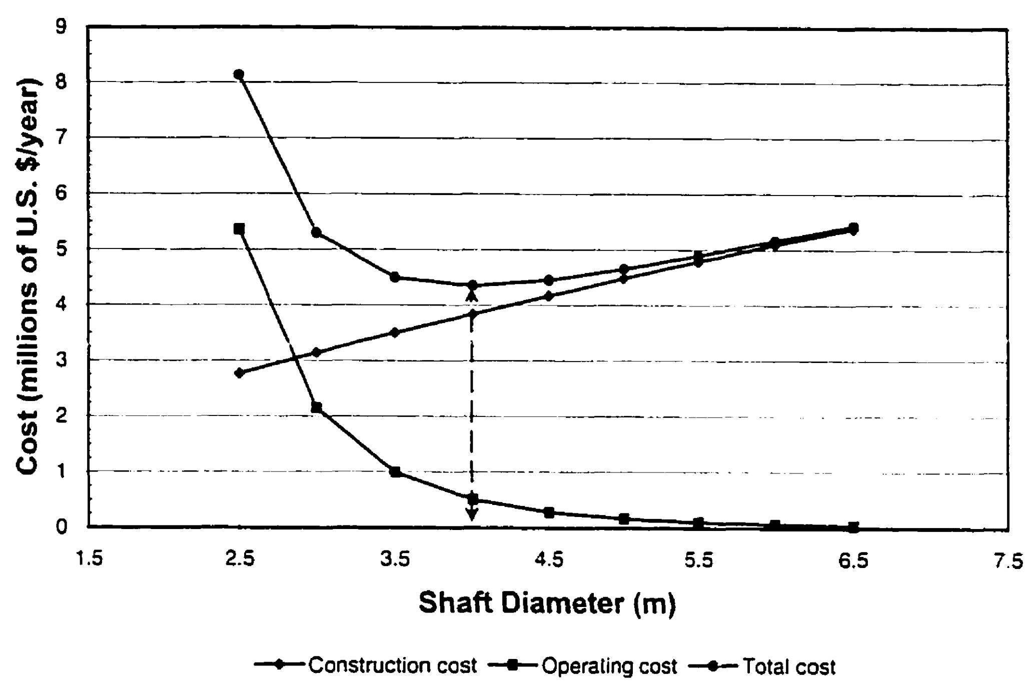

Figure 69: Fox River Project – Optimum ventilation shaft diameter – Mid-size equipment

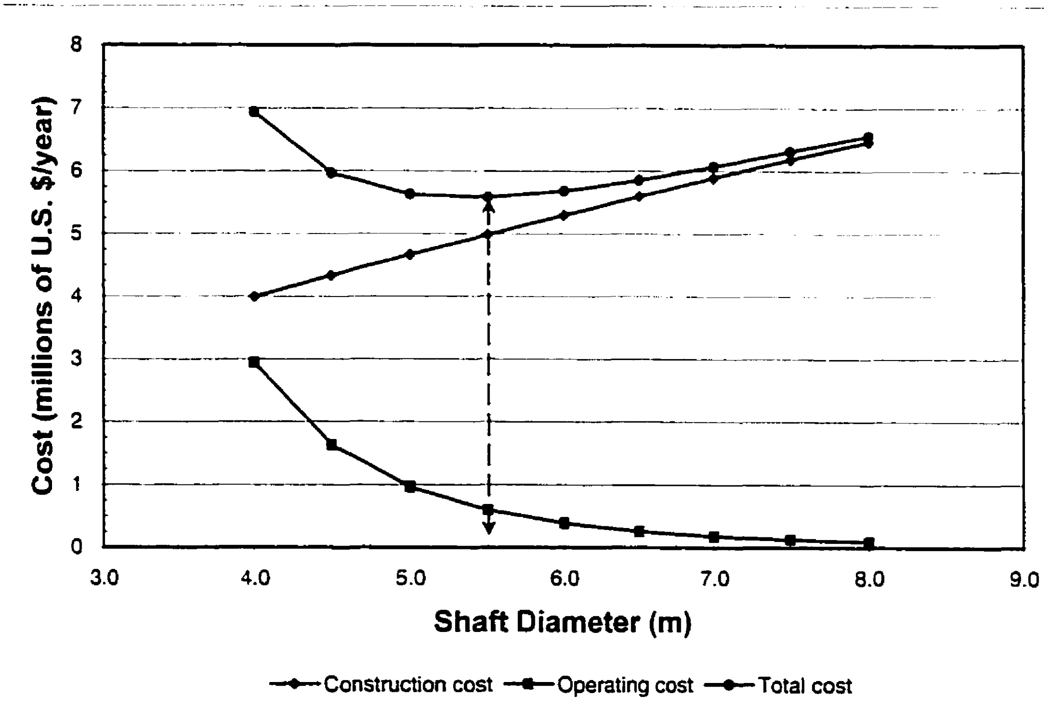

Figure 70: Fox River Project – Optimum ventilation shaft diameter – Large equipment

6.6.4 Backfill and Ore Processing

Instead of using first principles, backfill and ore processing operating costs were estimated using formulae provided by O'Hara and Suboleski (1992). For example, Table 62 and Table 63 show backfill plant and concentrator operating costs for large equipment scenarios. Similar tables were developed for small and mid-size equipment.

6.7 Capital Cost

This case study considers capital investment in pre-production mine development, mining and mine development equipment, and processing plant construction and equipment. Although some of such costs may be *expensed* in the year they were incurred instead of *capitalizing* them for subsequent depreciation, it is believed that in this way a more adequate return on the investment can be determined.

In addition to the excavation and support of all sub-horizontal and ramp development carried out during the first six years of every scenario (see Table 53, Table 55, and Table 57), mine development capital cost comprises the construction and implementation of all excavations listed in Table 42, Table 43, and Table 44, as well as and hoisting all the corresponding muck produced.

The investment programs for mining and mine development equipment reflect the corresponding ore production and underground development programs presented in Section 6.5.4. They are shown in Table 65 to Table 68. It must be noted that the investment programs in mining and development equipment consider replacing mobile units as they deteriorate over a certain period of time.

As in the case of operating costs, the estimation of the cost of constructing and furnishing the processing plant was carried out using the formulae developed by O'Hara and Suboleski (1992). For example, Table 69 shows the detailed calculation of the concentrator capital cost for small equipment scenarios using such formulae. It is important to note that, even within the small equipment cases, there was a difference in total capital cost of about US $ 1.7 million between the ones with the lowest and highest concentrator capital requirements.

Table 63: Fox River Project – Backfill plant operating cost – Large equipment

Activity	Formula	Scenario								
		10m ore-25m lvl	10m ore-50m lvl	10m ore-75m lvl	15m ore-25m lvl	15m ore-50m lvl	15m ore-75m lvl	20m ore-25m lvl	20m ore-50m lvl	20m ore-75m lvl
		3,882.6	4,017.8	4,232.3	3,722.2	3,733.9	4,431.2	4,500.1	4,238.2	4,380.7
Capital Cost (US $)	Backfill Plant = $\$4,500.00 \cdot T^{0.7}$	1,464,183.9	1,499,693.3	1,555,317.9	1,421,575.7	1,424,699.0	1,606,108.9	1,623,565.5	1,556,836.1	1,593,294.3
Operating Cost (US $/day)	Plant Operation = $\$12.00 \cdot T^{0.7}$	3,904.5	3,999.2	4,147.5	3,790.9	3,799.2	4,283.0	4,329.5	4,151.6	4,248.8
	Correction Factor = $(W_s/W)^{0.4}$	3,759.9	3,851.1	3,994.0	3,104.0	3,110.8	3,506.9	3,159.7	3,029.8	3,100.8

Source O'Hara and Suboleski, 1992

Table 64: Fox River Project – Concentrator operating cost – Large Equipment

Activity	Formula	Scenario								
		10m ore-25m lvl	10m ore-50m lvl	10m ore-75m lvl	15m ore-25m lvl	15m ore-50m lvl	15m ore-75m lvl	20m ore-25m lvl	20m ore-50m lvl	20m ore-75m lvl
		3,882.6	4,017.8	4,232.3	3,722.2	3,733.9	4,431.2	4,500.1	4,238.2	4,380.7
Primary Crushing	Primary Crushing = $\$2.00 \cdot T^{0.8}$	1,487.0	1,528.3	1,593.3	1,437.7	1,441.3	1,652.9	1,673.4	1,595.1	1,637.8
Fine Crushing	Fine Crushing = $\$12.60 \cdot T^{0.6}$	1,794.1	1,831.3	1,889.4	1,749.2	1,752.5	1,942.1	1,960.2	1,891.0	1,928.9
Grinding and Bins	Grinding and Bins = $\$4.90 \cdot T^{0.8}$	3,643.3	3,744.4	3,903.6	3,522.4	3,531.2	4,049.6	4,099.9	3,907.9	4,012.7
Processing Section	Processing Section = $\$40.00 \cdot T^{0.7}$	13,015.0	13,330.6	13,825.0	12,636.2	12,664.0	14,276.5	14,431.7	13,838.5	14,162.6
Electric Power	Electric Power = $\$164.00 \cdot T^{0.56}$	16,778.4	17,103.1	17,608.8	16,386.6	16,415.4	18,067.3	18,224.3	17,622.5	17,951.9
Total (US $/day)		36,717.8	37,537.8	38,820.1	35,732.2	35,804.5	39,988.5	40,389.5	38,855.0	39,693.9

Source O'Hara and Suboleski, 1992

Table 65: Fox River Project – Investment – Small mining equipment

Equipment	Yearly Investment (Thousands of US $)																						Total
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
Longhole Drill	-	-	-	-	-	1,882	-	-	-	-	-	1,882	-	-	-	-	-	1,882	-	-	-	-	5,645
Anfo Loader	-	-	-	-	-	192	-	-	-	-	-	-	-	192	-	-	-	-	-	-	-	-	383
LHD	-	-	-	-	-	228	-	-	-	-	-	228	-	-	-	-	-	228	-	-	-	-	683
Truck	-	-	-	-	-	678	-	-	-	-	-	678	-	-	-	-	-	678	-	-	-	-	2,033
Scissors Lift	-	-	-	-	-	283	-	-	-	-	-	-	-	283	-	-	-	-	-	-	-	-	566
Locomotive	-	-	-	-	-	430	-	-	-	-	-	-	-	430	-	-	-	-	-	-	-	-	860
Total	-	-	-	-	-	**3,691**	-	-	-	-	-	**2,787**	-	**905**	-	-	-	**2,787**	-	-	-	-	**10,169**

Table 66: Fox River Project – Investment – Mid-size mining equipment

Equipment	Yearly Investment (Thousands of US $)																						Total
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
Longhole Drill	-	-	-	-	-	2,782	-	-	-	-	-	2,782	-	-	-	-	-	-	-	-	-	-	5,564
Anfo Loader	-	-	-	-	-	192	-	-	-	-	-	-	-	192	-	-	-	-	-	-	-	-	383
LHD	-	-	-	-	-	307	-	-	-	-	-	307	-	-	-	-	-	-	-	-	-	-	614
Truck	-	-	-	-	-	782	-	-	-	-	-	782	-	-	-	-	-	-	-	-	-	-	1,563
Scissors Lift	-	-	-	-	-	283	-	-	-	-	-	-	-	283	-	-	-	-	-	-	-	-	566
Locomotive	-	-	-	-	-	550	-	-	-	-	-	-	-	550	-	-	-	-	-	-	-	-	1,100
Total	-	-	-	-	-	**4,895**	-	-	-	-	-	**3,871**	-	**1,025**	-	-	-	-	-	-	-	-	**9,790**

Table 67: Fox River Project – Investment – Large mining equipment

Equipment	Yearly Investment (Thousands of US $)																						Total
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
Longhole Drill	-	-	-	-	-	3,586	-	-	-	-	3,586	-	-	-	-	-	-	-	-	-	-	-	7,172
Anfo Loader	-	-	-	-	-	192	-	-	-	-	192	-	-	-	-	-	-	-	-	-	-	-	383
LHD	-	-	-	-	-	396	-	-	-	-	396	-	-	-	-	-	-	-	-	-	-	-	791
Truck	-	-	-	-	-	499	-	-	-	-	499	-	-	-	-	-	-	-	-	-	-	-	998
Scissors Lift	-	-	-	-	-	283	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	283
Locomotive	-	-	-	-	-	684	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	684
Total	-	-	-	-	-	**5,639**	-	-	-	-	**4,672**	-	-	-	-	-	-	-	-	-	-	-	**10,311**

Table 68: Fox River Project – Investment program - Development equipment

Equipment	Ore Width	Level	Yearly Investment (thousands of US $)																						Total
			1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	
Small	10-m	25-m	-	-	2,123	594	-	-	-	2,123	594	-	-	-	2,123	594	-	-	-	1 530	594	-	-	-	10,273
		50-m	-	-	2,123	594	-	-	-	2,123	-	-	-	-	2.123	-	-	-	-	2 123	-	-	-	-	9,086
		75-m	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	594	-	-	-	-	-	-	-	-	6,963
	15-m	25-m	-	-	2,123	594	-	-	-	2,123	594	-	-	-	2,123	594	-	-	-	-	-	-	-	-	8,150
		50-m	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	594	-	-	-	-	-	-	-	-	6,963
		75-m	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	594	-	-	-	-	-	-	-	-	6,963
	20-m	25-m	-	-	2,123	594	-	-	-	2,123	594	-	-	-	2,123	594	-	-	-	-	-	-	-	-	8,150
		50-m	-	-	2,123	594		-	-	1,530	594	-	-	-	1,530	594	-	-	-	-	-	-	-	-	6,963
		75-m	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	-	-	-	-	-	-	-	-	-	6,370
Mid-Size	10-m	25-m	-	-	2,393	700	700	-	-	2,393	700	700	-	-	2,393	-	700	-	-	-	-	-	-	-	10,677
		50-m	-	-	2,393	700	-	-	-	2,393	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	8,578
		75-m	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	7,878
	15-m	25-m	-	-	2,393	700	700	-	-	2,393	-	700	-	-	2 393	-	-	-	-	-	-	-	-	-	9,278
		50-m	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	7,878
		75-m	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	-	-	-	-	-	-	-	-	-	7,179
	20-m	25-m	-	-	2,393	700	700	-	-	1,693	700	700	-	-	1,693	-	-	-	-	-	-	-	-	-	8,578
		50-m	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	-	-	-	-	-	-	-	-	-	7,179
		75-m	-	-	2,393	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	-	-	-	-	-	5,485
Large	10-m	25-m	-	-	3,075	1,004	1,004	-	-	3,075	1,004	1,004	-	-	3,075	-	-	-	-	-	-	-	-	-	13,243
		50-m	-	-	3,075	1,004	-	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	-	-	-	-	-	11,234
		75-m	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	2,071	-	-	-	-	-	-	-	-	-	9,226
	15-m	25-m	-	-	3,075	1,004	1,004	-	-	3,075	1,004	-	-	-	3,075	-	-	-	-	-	-	-	-	-	12,239
		50-m	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	2,071	-	-	-	-	-	-	-	-	-	9,226
		75-m	-	-	3,075	1,004	-	-	-	3,075	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7,155
	20-m	25-m	-	-	3,075	1,004	1,004	-	-	3,075	-	1,004	-	-	-	-	-	-	-	-	-	-	-	-	9,163
		50-m	-	-	3,075	1,004	-	-	-	3,075	-	-	-	-	-	-	-	-	-		-	-	-	-	7,155
		75-m	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	-	-	-	-	-		-	-	-	-	7,155

Table 69: Fox River Project – Concentrator capital cost – Small equipment

Activity	Formula	Scenario								
		10m ore-25m lvl	10m ore-50m lvl	10m ore-75m lvl	15m ore-25m lvl	15m ore-50m lvl	15m ore-75m lvl	20m ore-25m lvl	20m ore-50m lvl	20m ore-75m lvl
		2,236.3	2,283.6	2,265.6	2,302.0	2,294.1	2,336.4	2,454.4	2,332.1	2,328.1
Mill Site Clearing (acres)	$A=0.05 \cdot T^{0.5}$	2.36	2.39	2.38	2.40	2.39	2.42	2.48	2.41	2.41
Clearing Costs	Clearing Costs = $\$2000 \cdot A^{0.9}$	4,339.0	4,380.1	4,364.5	4,395.9	4,389.2	4,425.3	4,524.6	4,421.7	4,418.3
Soil Stripping	Soil stripping = $\$1000 \cdot A^{0.8} \cdot D_u$	19,906.2	20,073.5	20,010.2	20,138.0	20,110.5	20,257.8	20,661.1	20,243.1	20,229.2
Mass Excavation	Mass Excavation = $\$200 \cdot Cu^{0.7}$	184,204.9	184,204.9	184,204.9	184,204.9	184,204.9	184,204.9	184,204.9	184,204.9	184,204.9
Excavation & Fill Compaction	Compaction = $\$850 \cdot Cd^{0.6} + \$75 \cdot Fc^{0.7}$	237,328.9	237,328.9	237,328.9	237,328.9	237,328.9	237,328.9	237,328.9	237,328.9	237,328.9
Concrete Foundations	Foundations = $\$30,000 \cdot T^{0.5}$	1,418,685.4	1,433,607.5	1,427,958.3	1,439,370.0	1,436,910.9	1,450,078.1	1,486,256.3	1,448,759.6	1,447,521.7
Building	Building = $\$27,000 \cdot T^{0.6}$	2,761,096.6	2,795,983.3	2,782,767.4	2,809,475.2	2,803,716.4	2,834,574.9	2,919,649.5	2,831,482.2	2,828,579.4
Gyratory Crusher	Gyratory Crusher = $\$63 \cdot T^{0.9}$	65,150.4	66,389.1	65,918.9	66,870.2	66,664.7	67,768.3	70,842.0	67,657.5	67,553.4
Primary Crushing Plant	Primary Crushing Plant = $\$15,000 \cdot T^{0.7}$	3,317,126.7	3,366,075.7	3,347,520.6	3,385,033.3	3,376,939.6	3,420,341.4	3,540,403.0	3,415,988.0	3,411,902.7
Fine Ore Crushing Plant	Fine Ore Crushing Plant = $\$18,000 \cdot T^{0.7}$	3,980,552.1	4,039,290.8	4,017,024.7	4,062,029.9	4,052,327.5	4,104,409.7	4,248,483.6	4,099,185.7	4,094,283.2
Grinding and Bins	Grinding and Bins = $\$18,700 \cdot T^{0.7}$	4,135,351.3	4,196,374.4	4,173,242.3	4,220,008.2	4,209,918.1	4,264,025.6	4,413,702.4	4,258,598.4	4,253,505.4
Processing Section	Processing Section = $\$30,100 \cdot T^{0.6}$	3,078,111.3	3,117,003.7	3,102,270.3	3,132,044.6	3,125,624.6	3,160,026.1	3,254,868.5	3,156,578.3	3,153,342.2
Tailings Storage	Tailings Storage = $\$20,000 \cdot T^{0.5}$	945,790.3	955,738.3	951,972.2	959,580.0	957,940.6	966,718.7	990,837.5	965,839.7	965,014.5
Water Supply	Water Supply = $\$14,000 \cdot T^{0.6}$	1,431,679.7	1,449,769.1	1,442,916.4	1,456,764.9	1,453,778.9	1,469,779.6	1,513,892.3	1,468,176.0	1,466,670.8
Peak Load (kW)	Peak Load = $165 \cdot T^{0.5}$	7,802.8	7,884.8	7,853.8	7,916.5	7,903.0	7,975.4	8,174.4	7,968.2	7,961.4
Substation	Substation = $\$580 \cdot (kW)^{0.8}$	753,749.5	760,085.3	757,688.3	762,528.6	761,486.2	767,063.4	782,335.6	766,505.4	765,981.4
Power Distribution	Power Distribution = $\$1,150 \cdot (kW)^{0.8}$	1,494,503.3	1,507,065.8	1,502,313.0	1,511,910.1	1,509,843.3	1,520,901.6	1,551,182.7	1,519,795.1	1,518,756.2
Other Facilities	Other Facilities = $\$10,000 \cdot T^{0.4}$	472,895.1	477,869.2	475,986.1	479,790.0	478,970.3	483,359.4	495,418.8	482,919.9	482,507.2
Total (US $)		24,300,470.9	24,611,239.7	24,493,487.2	24,731,482.9	24,680,154.7	24,955,263.7	25,714,592.0	24,927,684.3	24,901,799.7

6.8 Preparation of Cash Flows

Having determined the main components of the cash flow of every scenario in the previous sections of this chapter, it is now necessary to:

- calculate the unit value of the ore produced (in \$/tonne);[143]
- calculate the total ore production cost;[144] and,
- make assumptions regarding depreciation and tax regime.

The value of the ore was determined based on the ore production programs and ore recovery and dilution estimates. It was also necessary to make an assumption regarding the cost of smelting and refining the metal contained in the concentrates sold. Since it was already assumed that Fox River produced a base metal concentrate, it seemed adequate to consider a 10.0% treatment charge. This is in spite of the fact that, in reality, treatment charges depend on concentrate grade, contaminant content, tonnage sold, etc. However, this case study has only briefly considered the existence of the concentrator (basically, only to estimate its operating and capital costs) and, thus, it would not be appropriate to make assumptions regarding the quality of the concentrates produced by the various scenarios considered.

Total ore production costs were readily calculated from the cost data obtained in Sections 6.6 and 6.7.

Finally, and in order to simplify the calculations, it was decided to use straightforward depreciation and taxation systems: it was assumed to depreciate in five years, at 20.0% per year, and to apply a flat 40.0% tax rate.

The yearly cash flows corresponding to each of the 27 scenarios considered in this case study are included in Appendix I (see Table 86 to Table 112).

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143 The value of the ore is a function of waste and backfill dilution, which vary for each scenario.

144 This procedure was deemed to be more appropriate that estimating the unit ore production cost (in \$/tonne), which changes over the life of the mine due to the additional time required to deliver personnel and materials to the workplaces as the depth of mining increases. Furthermore, ore and waste extraction routes are also longer, and ventilation requirements increase due to higher resistance to airflow and heat generation. Since it was not possible to simulate such conditions in this case study, it would have been inadequate to use a single per-tonne production cost for the analysis of cash flows.

6.9 Analysis of the Results

The results obtained in this case study must be evaluated in the light of the objectives established in Section 6.1. Thus, it must be said outright that the second goal, i.e., the study of changes in cost structure as depth of mining increases, was not achieved. The reason for this is twofold:

1. The impact of depth is heavily dependent on local conditions.[145] In order to focus the analysis, the case study would have required many assumptions, turning it very specific and limiting the applicability of its results.
2. It would have required the addition of one more dimension to the three-dimensional problem already in place (the current "dimensions" being equipment size, orebody thickness, and inter-level spacing). This would have meant having to deal with at least 27 additional scenarios: one shallow and one deep case for every equipment size, thickness and level spacing combination. This was considered impractical.

On the other hand, and as seen in the first sections of this Chapter, the case study successfully addressed all the issues included as part of the first goal. The remainder of this section will discuss them critically, focusing on the economic aspects and the overall impact of the various mine development alternatives considered.

6.9.1 Economic Evaluation

As previously noted, the economic models of the mining scenarios developed in this case study constitute only approximations to what proper feasibility studies would have accomplished (with adequate data and resources). In fact, mine development and stoping (the main focus of this research project) were the areas in which detailed estimation of the time and resources involved was carried out. Thus, it is safe to assert that the estimated mine development and stoping costs are very close to what a feasibility study could have determined. On the other hand, the formulae used to produce some of the operating and capital cost estimates (see O'Hara and Suboleski,

[145] For example, in one of the operations visited as part of this research project, it was possible to observe moderate to strong rockburst activity at a depth of about 1,500 metres. However, in two other mines that exploited much wider base metal orebodies, such degree of damage was only found at a depth of more than 2,000 metres. Local factors such as in-situ stresses, stiffness of the rock, the presence (or absence) of water, etc., determine how soon depth-related ground control issues will be critical to an operation.

1992) do not have the same degree of certainty and, most probably, underestimate the reality of the deep mining operations being investigated. However, since costs estimated with the same methodology were used in all scenarios, it is believed that the procedure followed did not prevent from achieving results that allowed **relative** evaluations.

It is pertinent to note that the purpose of the Fox River case study was not to determine the mine development program that would result in the optimum production program. Nonetheless, it is difficult not to make a comparison that takes into account production cost and economic benefits achieved by all scenarios.[146] Therefore, the net present value (NPV) and internal rate of return (IRR) of the after-tax cash flow of every scenario were calculated. The results are graphically shown in Figure 71 to Figure 76.

Figure 71, Figure 73, and Figure 75 depict changes in Net Present Value (NPV) of the cash flows as the inter-level spacing of the mining scenarios increases from 25 m to 75 m. Figure 72, Figure 74, and Figure 76, on the other hand, present changes in Internal Rate of Return (IRR) with inter-level spacing. It should be noted that, in order to facilitate the visual analysis of the data, the scale of the *Y* axis of all plots of the same type (i.e., NPV and IRR plots) has been kept constant. In fact, the NPV scale goes from US $-15.0 million to US $35.0 million, whereas the IRR scale ranges from 7.5% to 15.0%. Not surprisingly (see Gentry and O'Neil, 1984, pp. 253-299) NPV and IRR plots of the same cash flows produce strikingly similar graphs.

Two main general trends can be observed:

1. With only one exception[147], this case study indicates that the profitability of mining thicker orebodies is higher. It is also evident that the use of larger mining equipment accentuates

[146] In hard rock mining, production cost is in most cases the dominant factor in any comparative analysis. Although not entirely correct (see discussion in Section 2.5.5) it must be remembered that, after all, cost is ***the*** competitive parameter in base- and precious metal mining. Nevertheless, an attempt was made in this section to use qualitative factors such as flexibility and efficiency to evaluate the resulting figures.

[147] Such an exception is the scenario in which 15-m ore is mined with 25-m levels and mid-size equipment. This scenario is a unique example of a situation in which a reduction in mine life, ore reserve recovery, and overall revenues, coupled with smaller level interval and higher mine development and concentrator expenditures, resulted in higher profitability. The increased operating and capital costs were more than compensated by mining higher grade ore at a faster rate over a shorter period (almost two years shorter, when compared with the 50-m level scenario).

the differences in profitability. In fact, the curves for small equipment scenarios are more clustered that those for large equipment.

2. Within a particular ore type, an increase in level spacing from 25 m to 50 m has, with only two exceptions, a beneficial effect on the economics of the operation. On the other hand, increasing the lift height from 50 m to 75 always has a negative impact. In both cases, the effect is more pronounced with larger equipment. In an extreme example, increasing the lift height from 50 m to 75 m when 10-m ore is mined with large equipment results in a NPV decrease of about US $21.0 million (the NPV drops from US $6.2 million to US $-14.7 million).

3. In all orebody thickness/inter-level spacing combinations, mid-size equipment resulted in the most profitable alternative. However, the difference in NPV (or IRR) between small and mid-size equipment scenarios is not as significant as the one observed between mid-size and large equipment.

Perhaps one of the most critical conclusions from the economic evaluation of the Fox River cash flows is that production cost **is not** the important issue when evaluating several alternatives for the exploitation of a certain deep orebody. Indeed, Figure 77, Figure 78, and Figure 79 show changes in production cost as inter-level spacing is increased from 25 to 75 metres.[148] The shape and general trend of the curves is similar to the one shown in Figure 20. They clearly indicate that as inter-level spacing increases from 25 m to 75 m, the production cost, expressed in dollars per tonne of ore produced, decreases accordingly. This information alone would suggest that significant economic benefits would be obtained from increasing the lift height. However, as discussed above, the reality is that 75-metre level intervals are the least economically attractive scenarios for Fox River.

There are three reasons for this inconsistency. First, the production cost in *dollars per tonne of ore* produced is not an adequate indicator of mining performance. Two major factors such as dilution and ore reserve recovery are not included in such a figure. Once they are factored in, scenarios resulting in high dilution and poor ore reserve recovery significantly increase their

[148] When looking at the absolute values plotted in these figures it should be kept in mind that operational and corporate overheads have not been considered. As noted in Section 5.7.3, overheads can account for up to 37% of total production cost.

corresponding production costs. A more appropriate (but not perfect) performance parameter would be production cost in dollars per ounce (or pound) of metal produced (see Section 2.5.5). Second, the production cost employed is the so-called *cash cost*, i.e., an operating cost that does not take into consideration the use of capital and other financial issues. This is incorrect since, for instance, indiscriminate use of capital (for example, the acquisition of very expensive equipment through heavy debt loads) can indeed lower the cash production cost even in terms of dollars per unit of metal content, but it may also bankrupt the company if the investment is not adequately recovered. Third, the time value of money is not accounted for. In other words, it is necessary to balance unit production cost with quantities produced over the entire life of the operation.

6.9.2 Mine Development

It is remarkable that the scenarios with the smallest investment in mine development (i.e., 75-metre inter-level spacing scenarios) and shorter mine lives resulted in the least economically attractive alternatives. This is a direct result of the loss of dilution control involved in increasing the distance between levels, which, in turn is a consequence of less efficient drilling and blasting practices. It is important to note that the in-situ value of the Fox River ore cannot be considered as high (see Section 6.4.2). In high-grade mines the impact of dilution or poor ore reserve recovery is even more pronounced, since the corresponding loss of revenue caused by dilution can be very considerable.

Not even the longer development periods that significantly extended the productive life of the 25-m level scenarios managed to completely eliminate the advantages of mining higher-grade material. In this regard, this case study clearly showed that mine development **is the** most important factor limiting the production rate of an underground mine. In fact, in spite of assuming the simultaneous operation of five stopes, the average ore production rate ranged from 2,100 tonnes/day with small equipment to 3,750 tonnes/day with large equipment (a 78.6% increase). The use of Taylor's Law would have resulted in production rates ranging from 2,800 to 4,000 tonnes/day, whereas the *seven-year minimum mine life* rule of thumb would have indicated a 5,000-tonne/day operation, ***regardless*** of equipment, lift height, or orebody thickness considerations (see Section 5.5.1). It is, thus, evident that rules of thumb, very popular (and useful) in open pit mining, have very limited applicability in deep underground mining.

The shape of the curves obtained indicates that there is a mine development scenario that can in fact maximize the profitability of an underground operation. In the case of Fox River, it is apparent that the optimum lift height must be between 25 and 50 metres (in fact, it may be closer to 50 m than to 25 m). The rapid deterioration of the economics of 75-m scenarios means that, unless the assumptions regarding the effects of the increase in level spacing are wrong, the chances of not achieving the maximum benefits by the exploitation of the orebody with such a development configuration are not very good.

The impact of equipment size is noticeable but not as evident as in the case of level spacing. Nonetheless, it is also clear that the size of the equipment cannot be increased (or decreased) without reaching a point at which the benefits of larger units are outweighed by the negative impact they have on grade control and flexibility. In Fox River, mid-size equipment results in better economic performance. However, it is the definition of *mid-size* that really matters. The equipment must be large enough to facilitate a cost-effective operation, while at the same time it should allow an optimum degree of selectivity.

It is believed that there is no point in trying to develop a formula to estimate/calculate the "right" development equipment configuration. As demonstrated by this case study, the efforts should be devoted to establishing a sound methodology for the evaluation of the various aspects involved in working out different mine development equipment scenarios. For instance, in addition to dilution and ore recovery, the Fox River scenarios have established the significant importance of hoisting, ventilation, and production rate both in terms of investment and operational requirements.

As noted in Section 4.4.4, the Canadian mining industry has already acknowledged the importance of balancing the smaller capital requirements in mine development resulting from larger lift heights, and the corresponding loss of dilution control and lower ore recovery. Therefore, the results of this case study are in accordance with real-life observations.

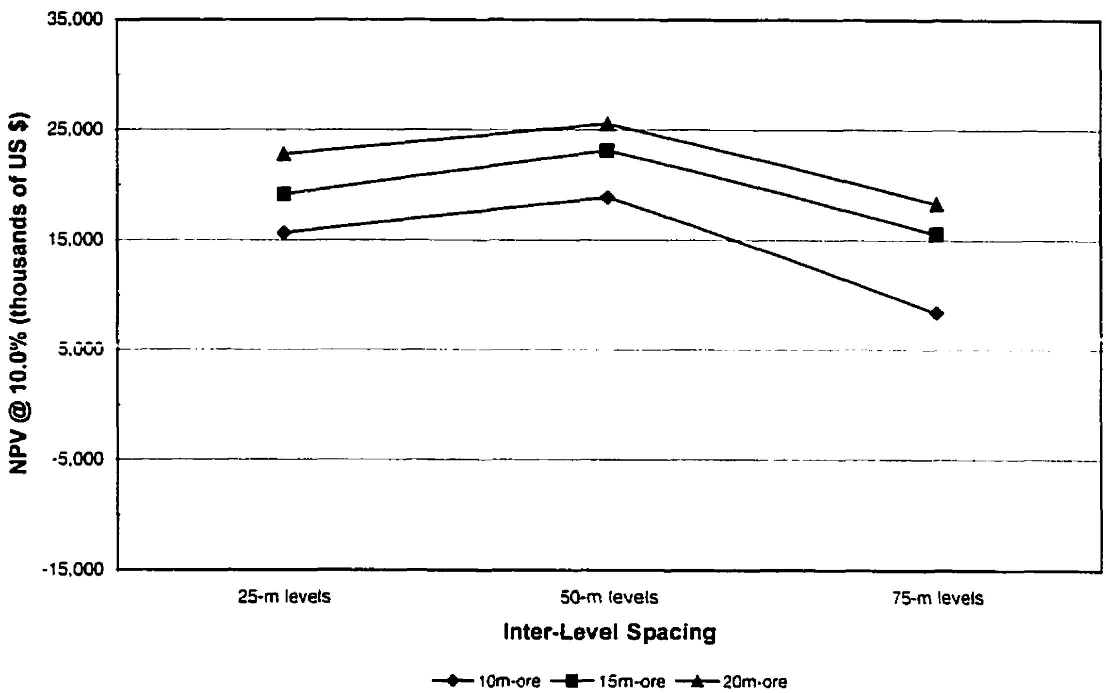

Figure 71: Fox River Project – NPV versus level spacing – Small equipment

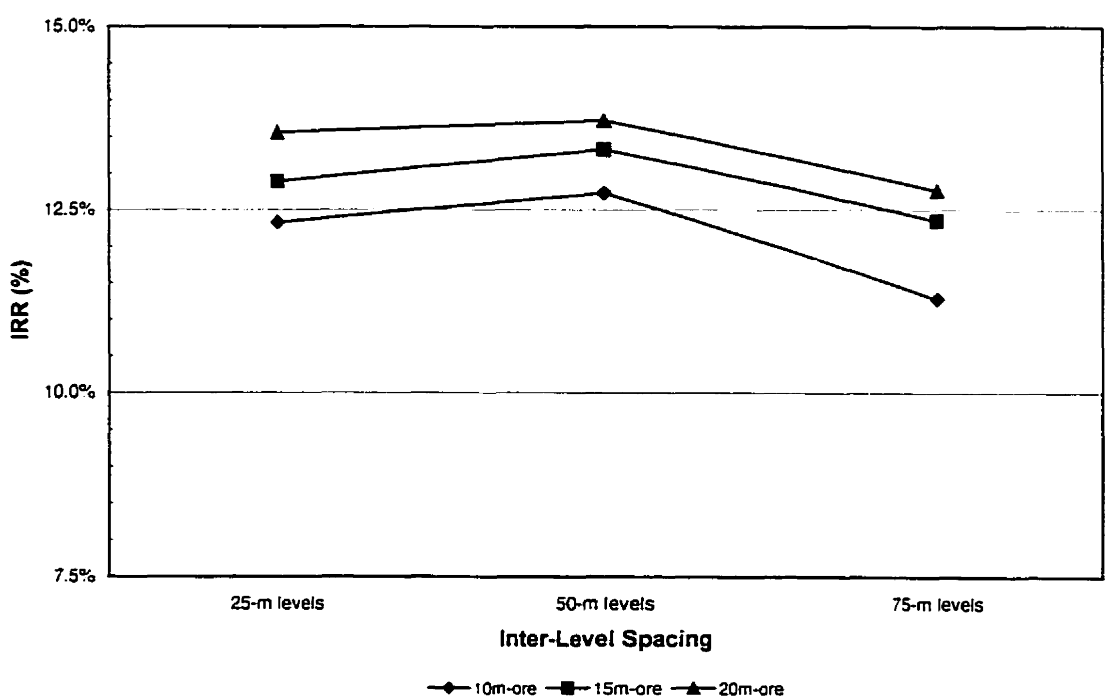

Figure 72: Fox River Project – IRR versus level spacing – Small equipment

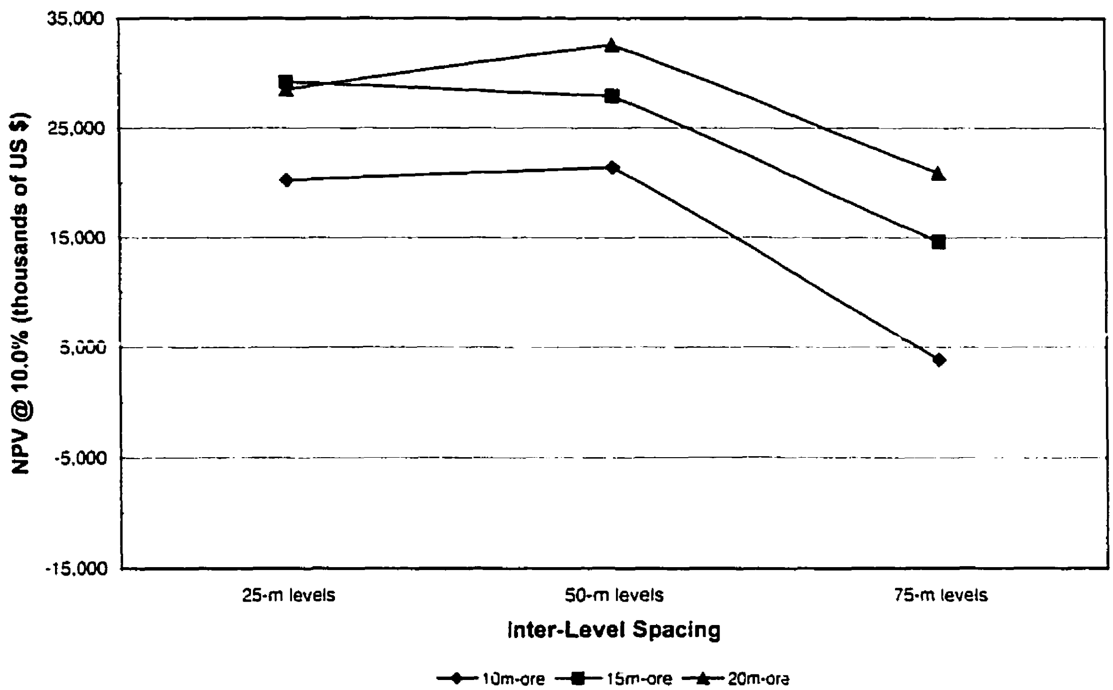

Figure 73: Fox River Project – NPV versus level spacing – Mid-size equipment

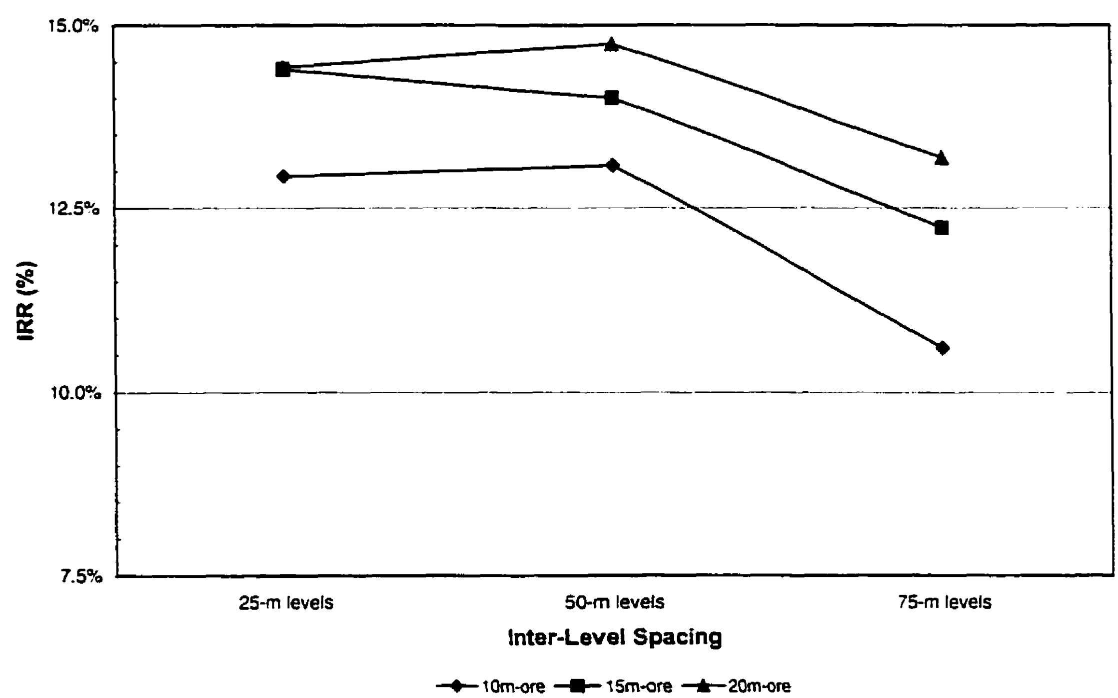

Figure 74: Fox River Project – IRR versus level spacing – Mid-size equipment

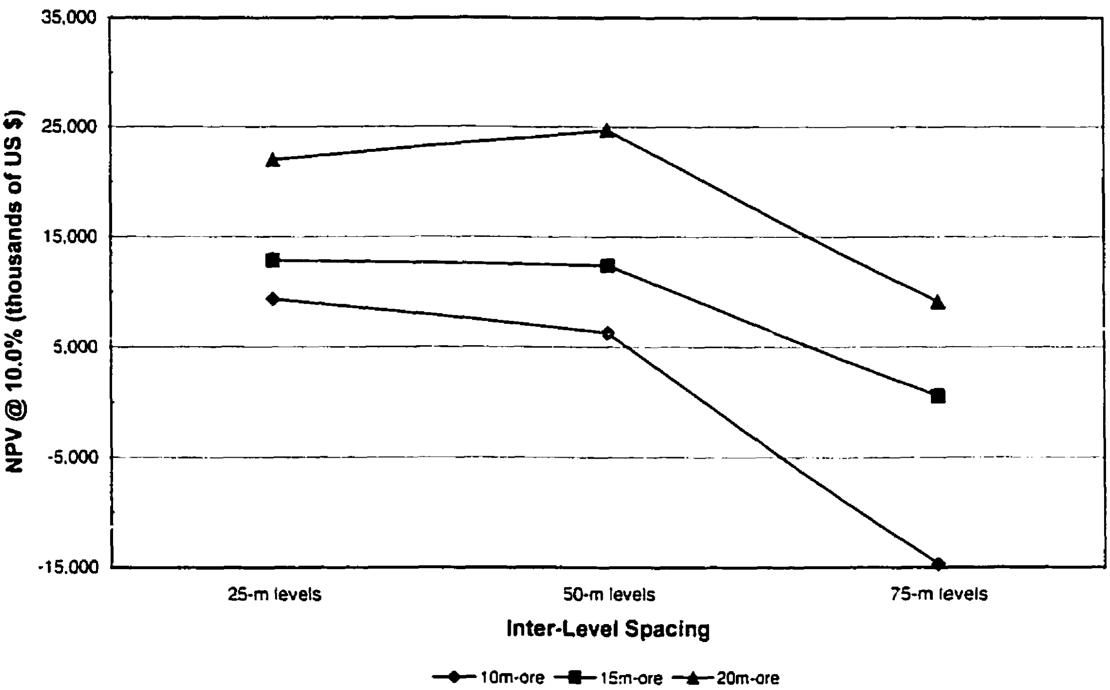

Figure 75: Fox River Project – NPV versus level spacing – Large equipment

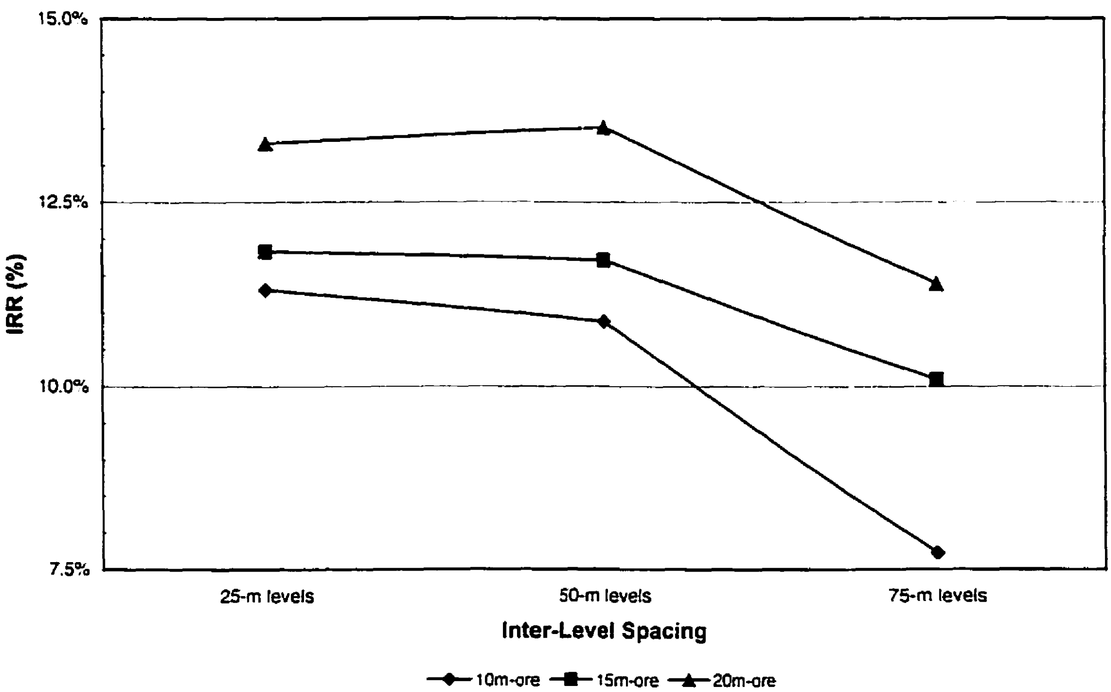

Figure 76: Fox River Project – IRR versus level spacing – Large equipment

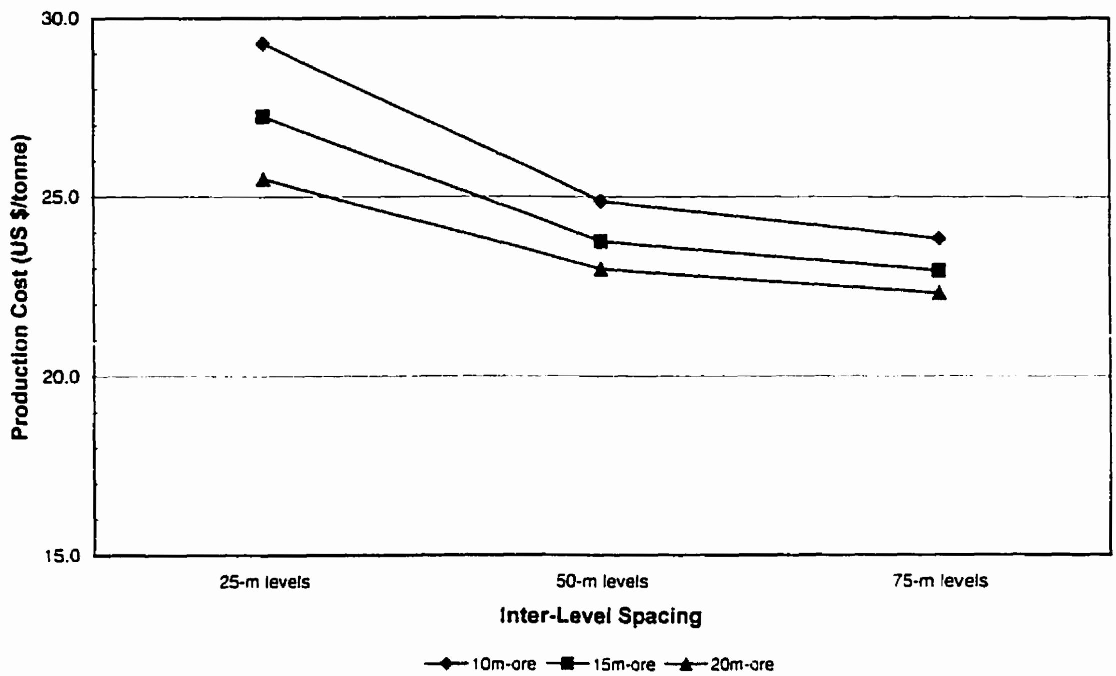

Figure 77: Fox River Project – Production cost vs level spacing – Small equipment

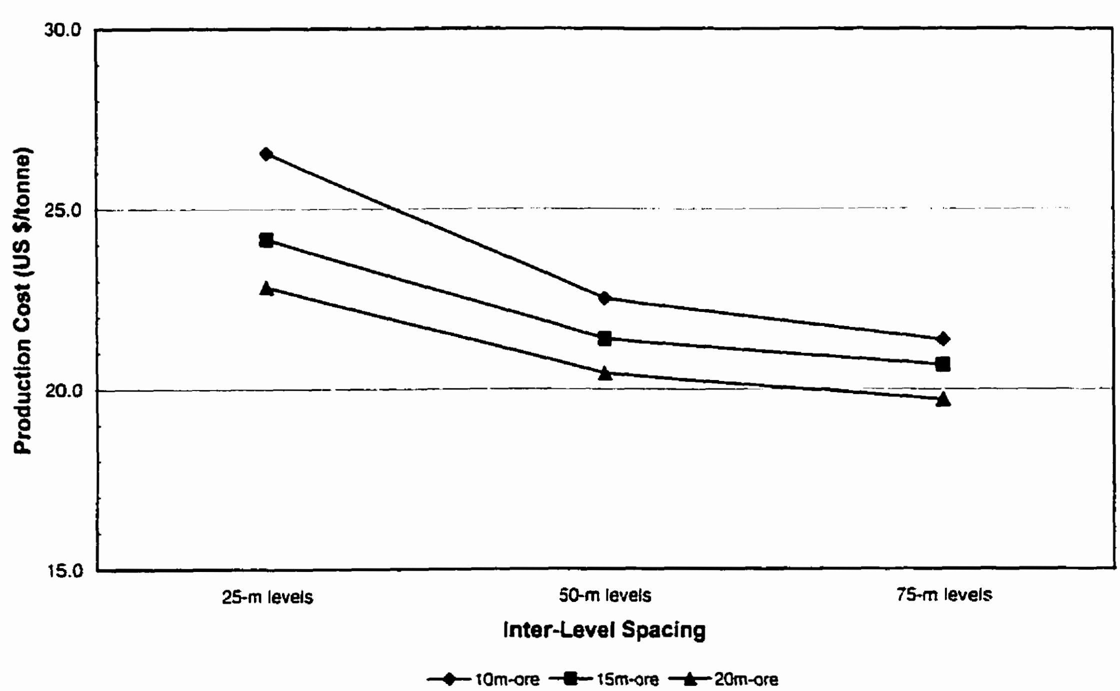

Figure 78: Fox River Project – Production cost vs level spacing – Mid-size equipment

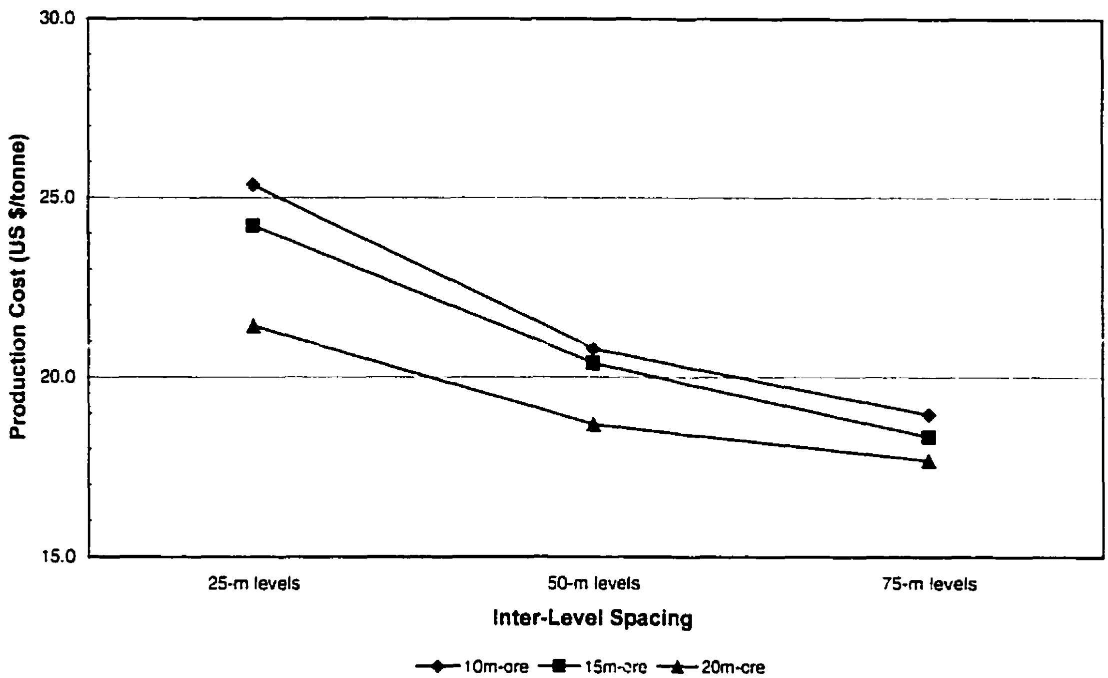

Figure 79: Fox River Project – Production cost vs level spacing – Large equipment

6.10 Summary and Conclusion

Twenty-seven mine development and mining scenarios constitute the Fox River case study. Their construction and evaluation has shown that changes in mine development regarding equipment size and inter-level spacing can have significant impact on the economics of a proposed mining operation. The use of the scenario analysis technique was particularly suitable for this case study, since it enabled the investigation of the effects of choices made at all stages in the process of constructing the scenarios.

Under the conditions of the Fox River case study, mid-size equipment and 50-metre inter-level spacings resulted in the most profitable mining configurations, regardless of orebody thickness. Such outcomes cannot be generalized, but they indicate that an *optimum* solution can be achieved by evaluating feasible alternatives using a procedure similar to the one presented in this Chapter. The very existence of an optimum solution, and the significant benefits that its identification and application could generate, make it imperative that mine operators carry out, as part of a standard mine investment analysis, the determination of its main characteristics. This should be done even when developing the so-called optimum is not practically feasible, since knowledge of the deviation from such an optimum could be very useful for future decision-making.

The successful application of the concept of *mining strategy* requires the development of tools that enable the mine operator to control key aspects of the production process (see Chapter 2). This case study has shown that one of such aspects is mine development. Therefore, additional research should focus on practical tools that facilitate the interactive and integrated evaluation of mine development alternatives. It is believed that significant improvements in underground mine design could be achieved by allowing the optimization of mine development design and planning.[149] The process of building the Fox River mine development scenarios with AutoCAD models, MS Excel spreadsheets, and MS Project schedules was slow, tedious, and inefficient. Following such a long and convoluted procedure, even when considering the prospective (and quite important) benefits that could be obtained from the results, would not be an acceptable proposition in real mining situations.

The analysis of the cash flows that resulted from the Fox River mining scenarios has two interesting implications for decision-making in underground operations. First, production cost by itself is not a good parameter for the evaluation of the economic performance of a mining operation or project. In its simplest form (i.e., when measured in dollars per tonne of ore processed) it does not account for dilution, the use of capital, and the time value of money. This case study has demonstrated that a much more reliable set of parameters can be obtained from cash flow analysis.

Second, mine development stands as the most important aspect of a deep underground mining project from both economic and operational perspectives. Long and expensive development periods call for careful consideration of their impact on mine life, production rate and several aspects of the mine operation such as dilution, ventilation, and muck transportation.

[149] To the author's knowledge, there was not a single piece of commercially available mining software at the beginning of year 2000 that could provide all the required capabilities.

7. Conclusion

The original proposal submitted to the sponsors of this research project had three well-defined objectives:

- to evaluate the pros and cons of using activity-based costing for decision-making in deep underground mining operations;
- to establish the constraints that most affect the profitability of ore extraction, particularly at depth; and,
- to develop the specifications for an integrated computer-aided mine design and planning system that facilitates the evaluation of the complex options facing mine operators.

The premise behind this proposal was that, while tremendous advancement was being achieved in, for instance, the areas of rock stress management and underground mine equipment design, there was not an integrated strategic approach to general mine design and planning. After consultations with the sponsors of the project, the overall objective of the research program was modified to become *"an investigation of the factors that most significantly affect mining at depth in order to determine their effect on the economic viability of deep mining in Ontario"* (see Section 1.2). This Chapter summarizes the main findings of this research with regards to the established goals.

7.1 Strategic Significance of the Factors

The theoretical evaluation and background research of the strategic significance of the five factors (see Chapter 4) coupled with the findings of the cases studies (see Chapters 5 and 6) make it possible to draw important general conclusions regarding their impact on deep underground hard-rock mining.

7.1.1 Vertical Ore/Waste Transport

Conventional vertical cable hoisting, which has been an integral part of Canadian mining for many decades, will continue to play its role in deep operations. This is because the other (theoretically) feasible alternatives for production hoisting have restrictions on the size of the material to be transported to surface (with the exception of trucking). Furthermore, there is always the issue of using so-called *unproven* technologies, such as vertical conveying and slurry pumping, in a very conservative industry such as mining.[150] Nonetheless, the use of vertical conveying for hoisting below the current bottom of an existing main shaft has significant potential, but requires primary crushing of the muck. In turn, this would call for a re-locatable crushing station if development is to just precede production at depth. As seen in Section 4.1.2, carrying out vertical conveying to surface would need several flights and transfers, a system not welcomed by most maintenance-conscious mine operators. It does not represent significant economic or operational advantages either.

Slurry pumping offers a great potential for deep hoisting in a single stage all the way to surface. However, it also requires underground secondary crushing, with the corresponding problems of increased heat generation and the need to centralize the hoisting function. On the other hand, the added potential for significant underground pre-concentration, which could result in important production cost reductions, and the continuous nature of the pumping process greatly enhance the attractiveness of this alternative.

7.1.2 Horizontal Ore/Waste Transport

The nature of the technologies currently used to carry out this task results in significant horizontal muck transport costs. As pointed out elsewhere, the potential exists to lower these costs (in dollars per tonne of material transported) by automation and/or tele-operation, which reduce the labour content and increase equipment utilization. However, the maintenance cost component may not be reduced significantly due to the increased complexity and sophistication of the machinery. From an operational view point, the need to excavate orepasses in competent ground, away from the orebody, could increase haulage distances significantly, thus overcoming the potential savings from the improved operation of mucking equipment.

[150] The author has seen internal reports produced by well established and forward-thinking Canadian underground mining companies in which, after thoroughly evaluating several technologies suitable for a particular task, the most promising and economically attractive solution is discarded due to its *unproven* and, thus, *risky* nature.

The mining industry must seriously investigate the use of more efficient technologies for horizontal muck transport. LHDs have very low payload-to-weight ratios and tend to suffer from low maintenance availability. Tracked conveyances do not have such drawbacks, but do not provide the flexibility inherent to trackless equipment and require a separate loading system. Similarly, conveying systems need primary crushing, unless current primary blasting practices are modified to reduce fragmentation. Furthermore, most alternative transport systems require that ore production be concentrated on intermediate levels through temporary orepasses to justify economically their high capital cost.

7.1.3 Ventilation

Most mining operations do not break down energy cost by use (i.e., hoisting, pumping, ventilation, etc.) and, thus, it is not possible to determine the exact ventilation cost. As seen in, for instance, Figure 68 to Figure 70, fan operating cost (i.e., energy consumption cost) can be very significant if the airways do not have the optimum cross-sectional area and shape. Nonetheless, at Lynx Brook, ventilation cost accounted for about 2.0% of mining cost (without energy cost), whereas in large equipment scenarios at Fox River it fluctuated between 3.0% and 3.5% (with energy cost included).

Adequate mine design, in particular mine development design, can result in significant savings in ventilation costs. Efficient mine sequencing also can have an important beneficial effect on ventilation. For instance, for a number of reasons most old operations such as those visited as part of this research project still ventilate areas (in fact, several levels) in which production activities had not taken place for several years. In deep operations, this not only increases ventilation demand but also endangers the future development of deep ore resources. Ventilation and general upkeep costs can be reduced drastically by properly decommissioning mined-out areas.

7.1.4 Mine Development

The examination of the cost structure of deep hard-rock mines has shown that the cost of mine development can be very high in this type of operations. Data provided by existing operations show that it can fluctuate between 20.0% and 25.0% of total mining cost, depending on the mining method. In mid-size equipment scenarios included in the Fox River case study, mine development varied between 42.0% and 53.0% of total operating mining cost. Even when considering that other components of the mining cost may have been underestimated (see Section 6.9.1), and that mine development in this case study included all costs related to such an activity (including ground support and hoisting the muck produced by mine development) these are very high percentages.

As the Fox River scenarios clearly indicate, development cost can be expected to increase at depth because of increased support requirements and the cost of removing the waste. To properly assess the alternatives, good cost data (such as that obtained with activity-based costing systems) and equipment productivity are required. For example, in one particular case, the cost of operating mucking equipment is charged against the level in which they work based on cumulative bucket capacity, thus precluding the differentiation between the various equipment sizes that are available in every level. This practice, not uncommon in mining operations, illustrates how poor data gathering can affect the ability to properly evaluate production cost with real (i.e., not estimated) data. Indeed, in this case it would be impossible to establish the operating efficiencies (and costs) related to different equipment working in the various faces/stopes available in a particular level.

7.1.5 Cost of Delivering Personnel and Supplies

With the exception of some papers on deep South African gold mines, there is scant information on underground personnel and materials distribution. Thus, it was difficult to address properly this important factor. In fact, cost information regarding distribution of personnel and supplies was not available readily at any of the operations that took part in this research project.

At one mine, underground material and personnel handling accounted for only 1.4% of total direct mining costs. Separating out the portion of the hoisting cost charged directly to the cages at the various shafts, it represented about 3% of direct costs. Other direct shaft maintenance items were certainly applicable to the service hoists but could not be broken down. These distribution costs however, considering the wide diversity of materials used underground, do not seem to be significant. Sometimes, these supplies are distributed to individual working locations, mainly on an allocated basis. Another practice is to distribute the supplies such as ground support devices and explosives to individual levels, and are not further allocated to individual job sites. The paper work to attempt to do this is likely not justified and, as shown in one particular case, is likely not accurate.

It is surprising to note the high proportion of costs attributable to general services. The total cost of general operating and maintenance consumables costs at Lynx Brook amounted to over 23% of the total direct mining cost. These costs are indicative of an operation spread over many levels, further demonstrating the need to more careful mine design and planning. To overcome these potential inefficiencies of distributing personnel and supplies to depth the entire mining system has to be examined. The cost advantages of such changes are qualitatively obvious, but unfortunately cannot be adequately quantified using existing cost data.

7.1.6 Summary

All of the examined factors are significant components of the mine production cost. Vertical ore and waste transport is particularly significant because it represents an area where significant cost increases and technological limitations will certainly take place with increased depth of mining. The capital cost of developing new conventional shafts to depth are enormous, as is the lead time. The remaining factors are very interrelated, with mine development being the key element linking all of them.

It is postulated that, until traditional hard-rock mine development technology (mobile drilling, mucking and haulage equipment coupled with traditional blasting techniques) is replaced by continuous/automated systems, it will maintain its position as the most strategic constituent of the mining process. This is not only due to the significant time and resources devoted to mine development, but also because of the impact it has on the entire production system. The Fox River case study has shown that different choices of mine development equipment and opening configurations can result in drastically different economic results. Furthermore, there is not a direct relationship between the amount of time and resources devoted to mine development and the resulting economic parameters. In other words, more intensive development (such as in the 25-m scenarios in the Fox River case study) can result in a more economically attractive operation, in spite of the higher direct capital and operating costs.

The future development of deep mining has to be examined in a strategic and integrated manner incorporating rock mechanics, production systems automation, and extraction sequences. With ore fragmentation playing such a critical role in hard-rock mining, new drilling and blasting systems are needed. It is important to consider the fact that systems that require continuous access to long horizontal slices of ore, such as some of the proposed mechanical excavation devices, will conflict with current mining sequencing practices that have proved to be very successful in dealing with the difficult ground conditions typical of deep mining. The level of innovation that, for instance, made mining 400-foot stopes at the Geco Mine economically possible is now required to help overcome the cost and technological problems of mining at depth.

7.2 Additional Comments

The cash flows resulting from the Fox River mining scenarios clearly indicate production cost in dollars per tonne of ore (an industry-accepted measure of mine operating performance) is not a good indicator of the economics of a mining operation. This is because such a parameter does not consider the effect of dilution, capital usage, and time value of money. On the other hand, even simple cash flow-based evaluation criteria such as NPV and IRR permit the comparative analysis of mine development alternatives. In fact, such criteria facilitate the determination of the mine development configuration and corresponding production plan that maximizes the return on the investment. Acknowledging the existence of an optimum mining scenario is critical to the underground mine evaluation process.

The problem (as demonstrated by both case studies) is that the preparation and evaluation of the underground mining scenarios required to perform cash flow analysis cannot be carried out without devoting significant resources to the process. In contrast, this has become a standard procedure in open pit mining, where even mathematical formulations are used to determine optimum pit limits and mining sequences. This is why, in spite of the potential benefits that can be achieved (particularly in deep mining conditions), it is believed that the industry will not adopt the methodology followed in the preparation of the case studies unless efficient computer-aided tools are available.

The mining industry should seriously consider the application of concepts successfully employed in other industries such as just-in-time production systems. Just-in-time mine development, for example, could reduce drastically level upkeep and support costs, but may also require re-design of current mining sequences. Returning to in-ore development, such as in the cases of Lynx Brook and Fox River, may offer potential savings but would reduce flexibility and increase supplies and personnel delivery costs. Changes such as these must be evaluated integrally, stressing the nced count, in the first place, with adequate mine design and planning tools.

The following comments are made in connection with the secondary objectives of this project.

7.2.1 Existing Cost Data

The cost data as presently collected is highly aggregated. Cost data must be compiled so that activity-based analysis can be performed routinely. This relates to direct production equipment costs, labour cost, maintenance cost, and distribution of overhead costs. The objective of this increased management effort would be to be able to project with an acceptable degree of certainty the evolution of the cost structure of an operation as depth of mining increases.

7.2.2 Specifications of a Computer-Assisted Planning System

The case studies carried out were painfully slow, in spite of the use of AutoCAD, a spreadsheet and other computer programs. Experienced mine planners familiar with their operation can become very proficient at accomplishing these tasks, but it is not believed that results obtained with existing (i.e., inadequate) tools could be useful for short or even medium term planning. If alternatives regarding mine development of sequencing are not evaluated promptly, the decisions come too late. Developing a complex three-dimensional numerical stress model, for example, is usually carried out for back-analysis as the mining extraction must proceed before the model is complete. While great strides have been made with existing software, more work is required in mine planning system development before these and other useful analyses can be carried out in a timely and efficient fashion.

In general, an effective underground mine design and planning tool would have to consider the fact that the process relies on large amounts of information that are distributed in three-dimensional space, consists of numerical, textual and graphical elements, and contains very complex data structures and relationships. It is beyond the scope of this thesis to provide specific software requirements, but in general it must be able to integrate all components of mining activities, provide a common database (which would contain data of several types), and allow simultaneous updates and calculations. Furthermore, it should allow easy development of graphic interfaces into currently available mining software, such as software use for numerical analysis of opening stability, geological modelling, ore reserve estimation, etc. Finally, it would be integrated with more general management software that provides information pertinent to business and corporate decision-making.

7.3 Future Work

Future research in the following areas is proposed:

- ***Computer-aided mine design and planning***

  The construction of the Fox River scenarios has provided important insight into the process of evaluating mine development alternatives. An extension of this research with the specific objective of designing and implementing mine development software would significantly benefit from the practical experience gained in the process.

- ***Cost collecting systems***

  An initial objective of this research, the study of alternative costing systems did not interest the potential sponsors from the mining industry. It is believed that this research has made it evident that decision-making in deep underground mining can take great advantage from proper compilation and analysis of the cost data produced by an on-going operation. The ability to evaluate changes in cut-off grade and general grade control policies can make a big difference not only in the current bottom-line, but in the future competitive position of Canadian mining industry. It is believed that activity-based costing, by concentrating on the main cost drivers of every stage in the production process, can greatly enhance decision-making at all levels. Additional research is still needed to facilitate the implementation of activity-based costing systems in mining corporations, a non-trivial and expensive task.

- ***Mine development equipment***

The strategic importance of mine development, stemming from the fact that it is through this activity that the actual production infrastructure of the mine is constructed, has been stressed throughout this research. Mine development is still dominated by traditional drilling and blasting practices and the use of mobile mucking and haulage equipment. The adoption of different development technologies would certainly affect the entire mining process, hopefully with resulting economic and strategic advantages. The real benefit of the new technology, however, can be determined only in an integrated manner, after investigating its impact on all aspects of the mining operation.

Further research in mine development equipment should focus on technologies that eliminate drilling and blasting, facilitate the implementation of efficient mining sequences at depth, make better use of energy, and do not impose additional ventilation requirements. The possibility of radically modifying the mine development system (maybe even the entire production process), not just the equipment or technologies involved, must be seriously considered. For example, the removal of inefficient LHDs from the mucking phase must also have beneficial impact on horizontal and vertical muck transport and/or ventilation (as a result, for instance, of better fragmentation and lower diesel equipment requirements). It is, thus, imperative that future research is provided with enough perspective to be able to evaluate the impact of its findings comprehensively.

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Appendix I – Fox River Project – Additional Tables

Table 70: Fox River Project – Horizontal development – Small equipment, 10-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m³	m	m³	m	m³	m	m³	m³	m	m³	m	m³	m	m³	m	m³	m	tonnes	tonnes	tonnes
	2,198	39,566	856	11,301			1,360	5,102	11,059			81	1,465	60	1,764	10	215	4,566		162,208	
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764	10	215	5,791	65,614	160,577	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
25	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764	10	215	5,791	65,614	160,577	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301			1,360	5,102	11,059			41	733	60	1,764	10	215	4,535	65,614	160,577	
	1,291	23,240										207	3,731					1,498		80,914	
	24,196	**435,519**	**11,128**	**146,915**	**9,416**	**124,312**	**17,680**	**66,332**	**143,770**	**4,400**	**50,688**	**777**	**13,988**	**780**	**22,932**	**40**	**860**	**68,417**	**787,371**	**1,851,205**	**306,251**
	2,198	39,566	856	11,301			1,360	5,102	11,059			81	1,465	60	1,764	10	215	4,566		162,208	
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764	10	215	5,791	65,614	160,577	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
50	2,209	39,755	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764	10	215	5,791	65,614	160,577	27,841
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301			1,360	5,102	11,059			41	733	60	1,764	10	215	4,535	65,614	160,577	
	1,291	23,240										207	3,731					1,498		80,914	
	14,809	**266,553**	**5,992**	**79,108**	**4,280**	**56,506**	**9,520**	**35,717**	**77,414**	**2,000**	**23,040**	**533**	**9,592**	**420**	**12,348**	**40**	**860**	**37,593**	**393,686**	**1,100,303**	**139,205**
	2,198	39,566	856	11,301			1,360	5,102	11,059			81	1,465	60	1,764	10	215	4,566		162,208	
	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764	10	215	5,791	65,614	160,577	27,841
75	1,565	28,161	856	11,301	856	11,301	1,360	5,102	11,059	400	4,608	41	733	60	1,764			5,137	65,614	125,150	27,841
	2,209	39,755	856	11,301			1,360	5,102	11,059			41	733	60	1,764	10	215	4,535	65,614	160,577	
	1,291	23,240										207	3,731					1,498		80,914	
	11,036	**198,637**	**4,280**	**56,506**	**2,568**	**33,903**	**6,800**	**25,512**	**55,296**	**1,200**	**13,824**	**452**	**8,127**	**300**	**8,820**	**30**	**645**	**26,665**	**262,457**	**814,576**	**83,523**

Table 71: Fox River Project – Horizontal development – Small equipment, 15-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m^3	m	m^3	m	m^3	m	m^3	m^3	m	m^3	m	m^3	m	m^3	m	m^3	m	tonnes	tonnes	tonnes
	2,198	39,566	820	10,886			1,950	9,258	13,824			81	1,465	60	1,764	10	215	5,120		170,502	
	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	2,209	39,755	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764	10	215	6,659	80,578	168,871	34,171
25	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	1,565	28,161	820	10,886	920	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	2,209	39,755	820	10,886			1,950	9,258	13,824			41	733	60	1,764	10	215	5,089	80,578	168,871	
	1,291	23,240										207	3,731					1,498		80,914	
	17,293	311,281	7,380	97,978	5,740	76,205	17,550	83,322	124,416	5,250	60,480	614	11,057	540	15,876	30	645	54,398	644,623	1,389,827	239,198
	2,198	39,566	820	10,886			1,950	9,258	13,824			81	1,465	60	1,764	10	215	5,120		170,502	
	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	2,209	39,755	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764	10	215	6,659	80,578	168,871	34,171
50	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	2,209	39,755	820	10,886			1,950	9,258	13,824			41	733	60	1,764	10	215	5,089	80,578	168,871	
	1,291	23,240										207	3,731					1,498		80,914	
	11,036	198,637	4,100	54,432	2,460	32,659	9,750	46,290	69,120	2,250	25,920	452	8,127	300	8,820	30	645	30,377	322,311	856,048	102,514
	2,198	39,566	820	10,886			1,950	9,258	13,824			81	1,465	60	1,764	10	215	5,120		170,502	
	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
75	1,565	28,161	820	10,886	820	10,886	1,950	9,258	13,824	750	8,640	41	733	60	1,764			6,005	80,578	133,445	34,171
	2,209	39,755	820	10,886			1,950	9,258	13,824			41	733	60	1,764	10	215	5,089	80,578	168,871	
	1,291	23,240										207	3,731					1,498		80,914	
	8,827	158,882	3,280	43,546	1,640	21,773	7,800	37,032	55,296	1,500	17,280	411	7,394	240	7,056	20	430	23,718	241,733	687,176	68,342

Table 72: Fox River Project – Horizontal development – Small equipment, 20-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m³	m	m³	m	m³	m	m³	m³	m	m³	m	m³	m	m³	m	m³	m	tonnes	tonnes	tonnes
	2,198	39,566	820	10,886			2,200	12,138	13,824			81	1,465	60	1,764	10	215	5,370		170,502	
	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
25	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
	2,209	39,755	820	10,886			2,200	12,138	13,824			41	733	60	1,764	10	215	5,339	92,098	168,871	
	1,291	23,240										207	3,731					1,498		80,914	
	13,520	**243,365**	**5,740**	**76,205**	**4,100**	**54,432**	**15,400**	**84,966**	**96,768**	**5,000**	**57,600**	**533**	**9,592**	**420**	**12,348**	**20**	**430**	**44,733**	**552,687**	**1,087,511**	**196,056**
	2,198	39,566	820	10,886			2,200	12,138	13,824			81	1,465	60	1,764	10	215	5,370		170,502	
	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
50	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
	2,209	39,755	820	10,886			2,200	12,138	13,824			41	733	60	1,764	10	215	5,339	92,098	168,871	
	1,291	23,240										207	3,731					1,498		80,914	
	8,827	**158,882**	**3,280**	**43,546**	**1,640**	**21,773**	**8,800**	**48,552**	**55,296**	**2,000**	**23,040**	**411**	**7,394**	**240**	**7,056**	**20**	**430**	**26,218**	**276,293**	**687,176**	**78,422**
	2,198	39,566	820	10,886			2,200	12,138	13,824			81	1,465	60	1,764	10	215	5,370		170,502	
	1,565	28,161	820	10,886	820	10,886	2,200	12,138	13,824	1,000	11,520	41	733	60	1,764			6,505	92,098	133,445	39,211
75	2,209	39,755	820	10,886			2,200	12,138	13,824			41	733	60	1,764	10	215	5,339	92,098	168,871	
	1,291	23,240										207	3,731					1,498		80,914	
	7,262	**130,721**	**2,460**	**32,659**	**820**	**10,886**	**6,600**	**36,414**	**41,472**	**1,000**	**11,520**	**370**	**6,662**	**180**	**5,292**	**20**	**430**	**18,712**	**184,196**	**553,732**	**39,211**

Table 73: Fox River Project – Horizontal development – Mid-size equipment, 10-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m³	m	m³	m	m³	m	m³	m³	m	m³	m	m³	m	m³	m	m³	m	tonnes	tonnes	tonnes
	2,194	46,221	840	12,224			1,360	6,194	13,056			81	1,715	60	1,764	10	215	4,545		188,914	
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764	10	215	5,755	73,671	187,005	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
25	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764	10	215	5,755	73,671	187,005	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224			1,360	6,194	13,056			41	858	60	1,764	10	215	4,515	73,671	187,005	
	1,289	27,161										207	4,368					1,496		94,587	
	24,176	**509,386**	**10,920**	**158,912**	**9,240**	**134,464**	**17,680**	**80,519**	**169,728**	**4,400**	**59,840**	**777**	**16,373**	**780**	**22,932**	**40**	**860**	**68,013**	**884,053**	**2,157,839**	**340,032**
	2,194	46,221	840	12,224			1,360	6,194	13,056			81	1,715	60	1,764	10	215	4,545		188,914	
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764	10	215	5,755	73,671	187,005	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
50	2,204	46,442	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764	10	215	5,755	73,671	187,005	30,912
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224			1,360	6,194	13,056			41	858	60	1,764	10	215	4,515	73,671	187,005	
	1,289	27,161										207	4,368					1,496		94,587	
	14,789	**311,602**	**5,880**	**85,568**	**4,200**	**61,120**	**9,520**	**43,356**	**91,392**	**2,000**	**27,200**	**533**	**11,228**	**420**	**12,348**	**40**	**860**	**37,382**	**442,026**	**1,282,291**	**154,560**
	2,194	46,221	840	12,224			1,360	6,194	13,056			81	1,715	60	1,764	10	215	4,545		188,914	
	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764	10	215	5,755	73,671	187,005	30,912
75	1,565	32,964	840	12,224	840	12,224	1,360	6,194	13,056	400	5,440	41	858	60	1,764			5,105	73,671	145,925	30,912
	2,204	46,442	840	12,224			1,360	6,194	13,056			41	858	60	1,764	10	215	4,515	73,671	187,005	
	1,289	27,161										207	4,368					1,496		94,587	
	11,020	**232,196**	**4,200**	**61,120**	**2,620**	**36,672**	**6,800**	**30,969**	**65,280**	**1,200**	**16,320**	**452**	**9,513**	**300**	**8,820**	**30**	**645**	**26,522**	**294,684**	**949,361**	**92,736**

Table 74: Fox River Project – Horizontal development – Mid-size equipment, 15-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m^3	m	m^3	m	m^3	m	m^3	m^3	m	m^3	m	m^3	m	m^3	m	m^3	m	tonnes	tonnes	tonnes
	2,194	46,221	800	11,680			1,950	11,146	16,320			81	1,715	60	1,764	10	215	5,095		198,706	
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	2,204	46,442	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764	10	215	6,615	91,302	196,797	38,290
25	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	2,204	46,442	800	11,680			1,950	11,146	16,320			41	858	60	1,764	10	215	5,065	91,302	196,797	
	1,289	27,161										207	4,368					1,496		94,587	
	17,278	**364,052**	**7,200**	**105,120**	**5,600**	**81,760**	**17,550**	**100,311**	**146,880**	**5,250**	**71,400**	**614**	**12,943**	**540**	**15,876**	**30**	**645**	**54,083**	**730,420**	**1,621,188**	**268,030**
	2,194	46,221	800	11,680			1,950	11,146	16,320			81	1,715	60	1,764	10	215	5,095		198,706	
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	2,204	46,442	800	11,080	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764	10	215	6,615	91,302	196,797	38,290
50	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	2,204	46,442	800	11,680			1,950	11,146	16,320			41	858	60	1,764	10	215	5,065	91,302	196,797	
	1,289	27,161										207	4,368					1,496		94,587	
	11,020	**232,196**	**4,000**	**58,400**	**2,400**	**35,040**	**9,750**	**55,728**	**81,600**	**2,250**	**30,600**	**452**	**9,513**	**300**	**8,820**	**30**	**645**	**30,202**	**365,210**	**998,321**	**114,870**
	2,194	46,221	800	11,680			1,950	11,146	16,320			81	1,715	60	1,764	10	215	5,095		198,706	
	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
75	1,565	32,964	800	11,680	800	11,680	1,950	11,146	16,320	750	10,200	41	858	60	1,764			5,965	91,302	155,717	38,290
	2,204	46,442	800	11,680			1,950	11,146	16,320			41	858	60	1,764	10	215	5,065	91,302	196,797	
	1,289	27,161										207	4,368					1,496		94,587	
	8,816	**185,753**	**3,200**	**46,720**	**1,600**	**23,360**	**7,800**	**44,582**	**65,280**	**1,600**	**20,400**	**411**	**8,656**	**240**	**7,056**	**20**	**430**	**23,687**	**273,907**	**801,524**	**76,680**

Table 75: Fox River Project – Horizontal development – Mid-size equipment, 20-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m³	m	m³	m	m³	m	m³	m³	m	m³	m	m³	m	m³	m	m³	m	tonnes	tonnes	tonnes
25	2,194	46,221	800	11,680			2,200	14,546	16,320			81	1,715	60	1,764	10	215	5,345		198,706	
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	2,204	46,442	800	11,680			2,200	14,546	16,320			41	858	60	1,764	10	215	5,315	104,902	196,797	
	1,289	27,161										207	4,368					1,496		94,587	
	13,510	**284,645**	**5,600**	**81,760**	**4,000**	**58,400**	**15,400**	**101,819**	**114,240**	**5,000**	**68,000**	**533**	**11,228**	**420**	**12,348**	**20**	**430**	**44,482**	**629,415**	**1,268,674**	**221,200**
50	2,194	46,221	800	11,680			2,200	14,546	16,320			81	1,715	60	1,764	10	215	5,345		198,706	
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	2,204	46,442	800	11,680			2,200	14,546	16,320			41	858	60	1,764	10	215	5,315	104,902	196,797	
	1,289	27,161										207	4,368					1,496		94,587	
	8,816	**185,753**	**3,200**	**46,720**	**1,600**	**23,360**	**8,800**	**58,182**	**65,280**	**2,000**	**27,200**	**411**	**8,656**	**240**	**7,056**	**20**	**430**	**25,087**	**314,707**	**801,524**	**88,480**
75	2,194	46,221	800	11,680			2,200	14,546	16,320			81	1,715	60	1,764	10	215	5,345		198,706	
	1,565	32,964	800	11,680	800	11,680	2,200	14,546	16,320	1,000	13,600	41	858	60	1,764			6,465	104,902	155,717	44,240
	2,204	46,442	800	11,680			2,200	14,546	16,320			41	858	60	1,764	10	215	5,315	104,902	196,797	
	1,289	27,161										207	4,368					1,496		94,587	
	7,252	**152,789**	**2,400**	**35,040**	**800**	**11,680**	**6,600**	**43,637**	**48,960**	**1,000**	**13,600**	**370**	**7,798**	**180**	**5,292**	**20**	**430**	**18,622**	**209,805**	**645,807**	**44,240**

Table 76: Fox River Project – Horizontal development – Large equipment, 10-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m^3	m	m^3	m	m^3	m	m^3	m^3	m	m^3	m	m^3	m	m^3	m	m^3	m	tonnes	tonnes	tonnes
	2,187	48,692	820	15,003			1,360	7,475	15,984			81	1,812	60	1,764	10	215	4,519		205,399	
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764	10	215	5,710	89,911	203,449	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
25	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764	10	215	5,710	89,911	203,449	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003			1,360	7,475	15,984			41	906	60	1,764	10	215	4,490	89,911	203,449	
	1,284	28,580										207	4,614					1,491		99,582	
	24,153	**537,646**	**10,660**	**195,039**	**9,020**	**165,033**	**17,680**	**97,172**	**207,792**	**4,400**	**73,260**	**777**	**17,298**	**780**	**22,932**	**40**	**860**	**67,510**	**1,078,932**	**2,359,584**	**417,013**
	2,187	48,692	820	15,003			1,360	7,475	15,984			81	1,812	60	1,764	10	215	4,519		205,399	
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764	10	215	5,710	89,911	203,449	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
50	2,199	48,948	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764	10	215	5,710	89,911	203,449	37,910
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003			1,360	7,475	15,984			41	906	60	1,764	10	215	4,490	89,911	203,449	
	1,284	28,580										207	4,614					1,491		99,582	
	14,763	**328,624**	**5,740**	**105,021**	**4,100**	**75,015**	**9,520**	**52,323**	**111,888**	**2,000**	**33,300**	**533**	**11,862**	**420**	**12,348**	**40**	**860**	**37,116**	**539,466**	**1,396,748**	**189,551**
	2,187	48,692	820	15,003			1,360	7,475	15,984			81	1,812	60	1,764	10	215	4,519		205,399	
	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764	10	215	5,710	89,911	203,449	37,910
75	1,565	34,837	820	15,003	820	15,003	1,360	7,475	15,984	400	6,660	41	906	60	1,764			5,066	89,911	160,473	37,910
	2,199	48,948	820	15,003			1,360	7,475	15,984			41	906	60	1,764	10	215	4,490	89,911	203,449	
	1,284	28,580										207	4,614					1,491		99,582	
	10,999	**244,840**	**4,100**	**75,015**	**2,460**	**45,009**	**6,800**	**37,374**	**79,920**	**1,200**	**19,980**	**452**	**10,050**	**300**	**8,820**	**30**	**645**	**26,341**	**359,644**	**1,032,826**	**113,731**

Table 77: Fox River Project – Horizontal development – Large equipment, 15-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m³	m	m³	m	m³	m	m³	m³	m	m³	m	m³	m	m³	m	m³	m	tonnes	tonnes	tonnes
	2,187	48,692	775	14,254			1,950	13,506	19,980			81	1,812	60	1,764	10	215	5,064		217,387	
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	2,199	48,948	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764	10	215	6,560	111,039	215,437	46,797
25	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	2,199	48,948	775	14,254			1,950	13,506	19,980			41	906	60	1,764	10	215	5,035	111,039	215,437	
	1,284	28,580										207	4,614					1,491		99,582	
	17,259	**384,188**	**6,975**	**128,284**	**5,425**	**99,776**	**17,550**	**121,554**	**179,820**	**5,250**	**87,413**	**614**	**13,674**	**540**	**15,876**	**30**	**645**	**53,643**	**888,310**	**1,782,609**	**327,580**
	2,187	48,692	775	14,254			1,950	13,506	19,980			81	1,812	60	1,764	10	215	5,064		217,387	
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	2,199	48,948	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764	10	215	6,560	111,039	215,437	46,797
50	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	2,199	48,948	775	14,254			1,950	13,506	19,980			41	906	60	1,764	10	215	5,035	111,039	215,437	
	1,284	28,580										207	4,614					1,491		99,582	
	10,999	**244,840**	**3,875**	**71,269**	**2,325**	**42,761**	**9,750**	**67,530**	**99,900**	**2,250**	**37,463**	**452**	**10,050**	**300**	**8,820**	**30**	**645**	**29,981**	**444,155**	**1,092,766**	**140,392**
	2,187	48,692	775	14,254			1,950	13,506	19,980			81	1,812	60	1,764	10	215	5,064		217,387	
	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
75	1,565	34,837	775	14,254	775	14,254	1,950	13,506	19,980	750	12,488	41	906	60	1,764			5,916	111,039	172,461	46,797
	2,199	48,948	775	14,254			1,950	13,506	19,980			41	906	60	1,764	10	215	5,035	111,039	215,437	
	1,284	28,580										207	4,614					1,491		99,582	
	8,800	**195,892**	**3,100**	**57,015**	**1,550**	**28,508**	**7,800**	**54,024**	**79,920**	**1,500**	**24,975**	**411**	**9,144**	**240**	**7,056**	**20**	**430**	**23,421**	**333,116**	**877,329**	**93,594**

Table 78: Fox River Project – Horizontal development – Large equipment, 20-m ore

Interval	Level		Sublevel				Access Crosscuts					Ventilation Drifts		Dump Station		Lunch Room		Total			
			Ore		Backfill		Distance	Ore	Waste	Backfill								Length	Ore	Waste	Backfill
m	m	m³	m	m³	m	m³	m	m³	m³	m	m³	m	m³	m	m³	m	m³	m	tonnes	tonnes	tonnes
25	2,187	48,692	775	14,254			2,200	17,668	19,980			81	1,812	60	1,764	10	215	5,314		217,387	
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	2,199	48,948	775	14,254			2,200	17,668	19,980			41	906	60	1,764	10	215	5,285	127,689	215,437	
	1,284	28,580										207	4,614					1,491		99,582	
	13,495	**300,403**	**5,425**	**99,776**	**3,875**	**71,269**	**16,400**	**123,679**	**139,860**	**5,000**	**83,250**	**533**	**11,862**	**420**	**12,348**	**20**	**430**	**44,168**	**766,133**	**1,394,711**	**270,408**
50	2,187	48,692	775	14,254			2,200	17,668	19,980			81	1,812	60	1,764	10	215	5,314		217,387	
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	2,199	48,948	775	14,254			2,200	17,668	19,980			41	906	60	1,764	10	215	5,285	127,689	215,437	
	1,284	28,580										207	4,614					1,491		99,582	
	8,800	**195,892**	**3,100**	**57,015**	**1,550**	**28,508**	**8,800**	**70,674**	**79,920**	**2,000**	**33,300**	**411**	**9,144**	**240**	**7,056**	**20**	**430**	**24,921**	**383,066**	**877,329**	**108,163**
75	2,187	48,692	775	14,254			2,200	17,668	19,980			81	1,812	60	1,764	10	215	5,314		217,387	
	1,565	34,837	775	14,254	775	14,254	2,200	17,668	19,980	1,000	16,650	41	906	60	1,764			6,416	127,689	172,461	54,082
	2,199	48,948	775	14,254			2,200	17,668	19,980			41	906	60	1,764	10	215	5,285	127,689	215,437	
	1,284	28,580										207	4,614					1,491		99,582	
	7,235	**161,056**	**2,325**	**42,761**	**775**	**14,254**	**6,600**	**53,005**	**59,940**	**1,000**	**16,650**	**370**	**8,238**	**180**	**5,292**	**20**	**430**	**18,505**	**255,378**	**704,868**	**54,082**

Table 79: Fox River Project – Mining cycle – Small equipment

Ore Width	Interval	Explosive		Blasthole Data			Drilling			Blasting			Mucking		Backfill				Total
		P.F.	Usage	Diameter	%Loaded	Net Load	Length	Rate	Time	Loading	Rounds	Tonnage	Rate	Time	Volume	Backfill	Cure	Total Time	
m	m	kg/tonne	kg/block	mm	%	kg/m	m/block	m/day	days	days	#	tonne/round	tonne/day	day/round	m^3	day/block	day/block	day/block	day/block
10	25	0.40	8,150	64	70.0%	2.03	4,022	125	32	1	5	4,312	450	10	5,974	3	7	10	72
	50	0.40	18,150	64	70.0%	2.03	8,956	150	60	2	6	7,774	450	17	12,619	6	7	13	134
	75	0.40	28,150	64	70.0%	2.03	13,890	180	77	2	8	8,531	450	19	18,683	8	7	15	190
15	25	0.40	9,232	64	70.0%	2.03	4,555	125	36	1	5	4,821	450	11	6,637	3	7	10	77
	50	0.40	21,232	64	70.0%	2.03	10,476	150	70	2	6	9,003	450	20	14,585	6	7	13	152
	75	0.40	33,232	64	70.0%	2.03	16,397	180	91	2	8	10,023	450	22	21,960	10	7	17	216
20	25	0.40	11,544	64	70.0%	2.03	5,696	125	46	1	5	6,004	450	13	8,248	4	7	11	88
	50	0.40	27,544	64	70.0%	2.03	13,591	150	91	2	7	9,992	450	22	18,902	8	7	15	190
	75	0.40	43,544	64	70.0%	2.03	21,485	180	119	2	9	11,682	450	26	28,898	13	7	20	279

Table 80: Fox River Project – Mining cycle – Mid-size equipment

Ore Width	Interval	Explosive		Blasthole Data			Drilling			Blasting			Mucking		Backfill				Total
		P.F.	Usage	Diameter	%Loaded	Net Load	Length	Rate	Time	Loading	Rounds	Tonnage	Rate	Time	Volume	Backfill	Cure	Total Time	
m	m	kg/tonne	kg/block	mm	%	kg/m	m/block	m/day	days	days	#	tonne/round	tonne/day	day/round	m^3	day/block	day/block	day/block	day/block
10	25	0.50	10,250	89	70.0%	3.92	2,615	125	21	1	5	4,459	660	7	6,298	3	7	10	57
	50	0.50	22,750	89	70.0%	3.92	5,805	150	39	2	6	8,047	660	12	13,357	6	7	13	104
	75	0.50	35,250	89	70.0%	3.92	8,994	180	50	2	8	8,944	660	14	20,169	9	7	16	151
15	25	0.50	11,630	89	70.0%	3.92	2,967	125	24	1	5	4,946	660	7	6,933	3	7	10	57
	50	0.50	26,630	89	70.0%	3.92	6,795	150	45	2	6	9,211	660	14	15,227	7	7	14	117
	75	0.50	41,630	89	70.0%	3.92	10,622	180	59	2	8	10,325	660	16	23,232	10	7	17	168
20	25	0.50	14,560	89	70.0%	3.92	3,715	125	30	1	5	6,134	660	9	8,565	4	7	11	68
	50	0.50	34,560	89	70.0%	3.92	8,818	150	59	2	7	10,158	660	15	19,565	9	7	16	142
	75	0.50	54,560	89	70.0%	3.92	13,921	180	77	2	9	11,919	660	18	30,190	13	7	20	207

Table 81: Fox River Project – Mining cycle – Large equipment

Ore Width	Interval	Explosive		Blasthole Data			Drilling			Blasting			Mucking		Backfill				Total
		P.F.	Usage	Diameter	%Loaded	Net Load	Length	Rate	Time	Loading	Rounds	Tonnage	Rate	Time	Volume	Backfill	Cure	Total Time	
m	m	kg/tonne	kg/block	mm	%	kg/m	m/block	m/day	days	days	#	tonne/round	tonne/day	day/round	m³	day/block	day/block	day/block	day/block
10	25	0.60	12,146	127	70.0%	7.98	1,522	125	12	1	5	4,685	900	5	6,855	3	7	10	47
	50	0.60	27,146	127	70.0%	7.98	3,402	150	23	2	6	8,674	900	10	14,944	7	7	14	93
	75	0.60	42,146	127	70.0%	7.98	5,281	180	29	2	8	9,995	900	11	23,580	10	7	17	128
15	25	0.60	13,798	127	70.0%	7.98	1,729	125	14	1	5	5,105	900	6	7,392	3	7	10	52
	50	0.60	31,798	127	70.0%	7.98	3,984	150	27	2	6	9,702	900	11	16,601	7	7	14	99
	75	0.60	49,798	127	70.0%	7.98	6,240	180	35	2	8	11,167	900	12	26,225	11	7	18	137
20	25	0.60	17,277	127	70.0%	7.98	2,165	125	17	1	5	6,262	900	7	9,010	4	7	11	58
	50	0.60	41,277	127	70.0%	7.98	5,172	150	34	2	7	10,553	900	12	20,970	9	7	16	121
	75	0.60	65,277	127	70.0%	7.98	8,179	180	45	2	9	12,639	900	14	33,292	15	7	22	173

Table 82: Fox River Project – Ore production rate -- Small equipment

Ore Width	Interval	Development Tonnage			Diluted Stope Tonnage			Active Stopes	Overall Mining Cycle		Effective Stope Production Rate				Mine Life
		Block	Level	Total	Block	Level	Mine								
m	m	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes		days/level	months/level	tonnes/day	tonnes/month	tonnes/year	levels/year	years
10	25	1,640	65,614	787,371	21,558	862,314	10,347,772	5	457	15.2	1,887	56,607	688,719	0.8	15.0
	50	1,640	65,614	393,686	46,643	1,865,710	11,194,258	5	932	31.1	2,002	60,055	730,670	0.4	15.3
	75	1,640	65,614	262,457	68,248	2,729,905	10,919,618	5	1,366	45.5	1,998	59,954	729,440	0.3	15.0
15	25	1,612	80,578	644,623	24,105	1,205,238	9,641,904	5	617	20.6	1,953	58,602	712,985	0.6	13.5
	50	1,612	80,578	322,311	54,017	2,700,830	10,803,321	5	1,340	44.7	2,016	60,466	735,674	0.3	14.7
	75	1,612	80,578	241,733	80,181	4,009,059	10,663,971	5	1,944	64.8	2,062	61,868	752,730	0.2	14.2
20	25	1,842	92,098	552,587	30,022	1,501,097	9,006,583	5	718	23.9	2,091	62,720	763,093	0.5	11.8
	50	1,842	92,098	276,293	69,947	3,497,370	10,492,110	5	1,702	56.7	2,055	61,646	750,023	0.2	14.0
	75	1,842	92,098	184,196	105,142	5,257,101	10,514,202	5	2,547	84.9	2,064	61,921	753,373	0.1	14.0

Table 83: Fox River Project – Ore production rate – Mid-size equipment

Ore Width	Interval	Development Tonnage			Diluted Stope Tonnage			Active Stopes	Overall Mining Cycle		Effective Stope Production Rate				Mine Life
		Block	Level	Total	Block	Level	Mine								
m	m	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes		days/level	months/level	tonnes/day	tonnes/month	tonnes/year	levels/year	years
10	25	1,842	73,671	884,053	22,293	891,732	10,700,781	5	337	11.2	2,646	79,383	965,822	1.1	11.1
	50	1,842	73,671	442,026	48,285	1,931,381	11,588,284	5	692	23.1	2,791	83,730	1,018,720	0.5	11.4
	75	1,842	73,671	294,684	71,551	2,862,045	11,448,179	5	1,047	34.9	2,734	82,007	997,752	0.3	11.5
15	25	1,826	91,302	730,420	24,729	1,236,465	9,891,717	5	418	13.9	2,958	88,741	1,079,688	0.9	9.2
	50	1,826	91,302	365,210	55,268	2,763,397	11,053,588	5	981	32.7	2,817	84,508	1,028,175	0.4	10.8
	75	1,826	91,302	273,907	82,600	4,129,987	10,982,858	5	1,464	48.8	2,821	84,631	1,029,676	0.2	10.7
20	25	2,098	104,902	629,415	30,669	1,533,472	9,200,832	5	518	17.3	2,960	88,811	1,080,535	0.7	8.5
	50	2,098	104,902	314,707	71,108	3,555,401	10,666,204	5	1,212	40.4	2,933	88,005	1,070,727	0.3	10.0
	75	2,098	104,902	209,805	107,275	5,363,755	10,727,511	5	1,827	60.9	2,936	88,075	1,071,577	0.2	10.0

Table 84: Fox River Project – Ore production rate – Large equipment

Ore Width	Interval	Development Tonnage			Diluted Stope Tonnage			Active Stopes	Overall Mining Cycle		Effective Stope Production Rate				Mine Life
		Block	Level	Total	Block	Level	Mine								
m	m	tonnes	tonnes	tonnes	tonnes	tonnes	tonnes		days/level	months/level	tonnes/day	tonnes/month	tonnes/year	levels/year	years
10	25	2,248	89,911	1,078,932	23,423	936,909	11,242,907	5	292	9.7	3,209	96,258	1,171,136	1.3	9.6
	50	2,248	89,911	539,466	52,045	2,081,787	12,490,720	5	597	19.9	3,487	104,612	1,272,784	0.6	9.8
	75	2,248	89,911	359,644	79,962	3,198,468	12,793,871	5	856	28.5	3,737	112,096	1,363,833	0.4	9.4
15	25	2,221	111,039	888,310	25,526	1,276,296	10,210,367	5	367	12.2	3,478	104,329	1,269,341	1.0	8.0
	50	2,221	111,039	444,155	58,212	2,910,597	11,642,387	5	801	26.7	3,634	109,011	1,326,302	0.5	8.8
	75	2,221	111,039	333,116	89,335	4,466,764	11,874,494	5	1,145	38.2	3,901	117,033	1,423,903	0.3	8.3
20	25	2,554	127,689	766,133	31,312	1,565,617	9,393,701	5	418	13.9	3,745	112,365	1,367,106	0.9	6.9
	50	2,554	127,689	383,066	73,868	3,693,397	11,080,192	5	1,003	33.4	3,682	110,471	1,344,058	0.4	8.2
	75	2,554	127,689	255,378	113,753	5,687,650	11,375,299	5	1,469	49.0	3,872	116,153	1,413,201	0.2	8.0

Table 85: Fox River Project – Number of mine development crews

Equipment	Thickness	Level	Year 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Small	10-m	25-m	-	-	-	1	3	4	4	4	4	4	4	4	4	4	4	4	4	3	2	2	2	-	61
		50-m	-	-	-	1	3	3	3	3	3	2	2	2	2	2	1	1	1	1	1	1	-	-	32
		75-m	-	-	-	1	3	3	2	2	2	2	2	1	1	1	1	1	1	1	-	-	-	-	24
	15-m	25-m	-	-	-	1	3	4	4	3	4	4	3	3	3	3	3	3	2	2	1	-	-	-	46
		50-m	-	-	-	1	3	3	3	2	2	2	2	2	1	1	1	1	1	1	-	-	-	-	26
		75-m	-	-	-	1	3	3	2	2	2	1	1	1	1	1	1	1	-	-	-	-	-	-	20
	20-m	25-m	-	-	-	1	3	4	4	3	3	3	3	3	3	3	3	2	1	-	-	-	-	-	39
		50-m	-	-	-	1	3	3	2	2	2	1	1	1	1	1	1	1	1	-	-	-	-	-	21
		75-m	-	-	-	1	3	3	2	1	1	1	1	1	1	1	1	-	-	-	-	-	-	-	17
Mid-Size	10-m	25-m	-	-	-	1	4	6	4	5	5	5	5	5	5	5	4	3	1	-	-	-	-	-	58
		50-m	-	-	-	1	3	4	4	3	3	3	2	2	2	2	2	2	-	-	-	-	-	-	33
		75-m	-	-	-	1	3	4	3	2	2	2	1	1	1	1	1	1	1	-	-	-	-	-	24
	15-m	25-m	-	-	-	1	4	5	4	5	5	4	4	4	4	4	3	-	-	-	-	-	-	-	47
		50-m	-	-	-	1	3	4	4	3	2	2	1	2	2	2	1	-	-	-	-	-	-	-	27
		75-m	-	-	-	1	3	3	3	2	2	2	1	1	1	1	1	-	-	-	-	-	-	-	21
	20-m	25-m	-	-	-	1	4	5	4	4	4	4	4	4	3	2	-	-	-	-	-	-	-	-	39
		50-m	-	-	-	1	3	4	3	2	2	2	1	1	1	1	1	-	-	-	-	-	-	-	22
		75-m	-	-	-	1	3	3	2	2	1	1	1	1	1	-	-	-	-	-	-	-	-	-	16
Large	10-m	25-m	-	-	-	1	4	5	6	5	6	6	6	6	5	5	4	1	-	-	-	-	-	-	60
		50-m	-	-	-	1	4	4	4	4	3	3	3	2	2	2	1	-	-	-	-	-	-	-	33
		75-m	-	-	-	1	4	4	4	2	2	2	2	1	1	1	1	-	-	-	-	-	-	-	25
	15-m	25-m	-	-	-	1	4	5	5	5	5	5	4	4	4	4	1	-	-	-	-	-	-	-	47
		50-m	-	-	-	1	3	4	4	3	2	2	2	2	2	2	-	-	-	-	-	-	-	-	27
		75-m	-	-	-	1	4	3	3	3	3	2	1	1	1	-	-	-	-	-	-	-	-	-	22
	20-m	25-m	-	-	-	1	4	5	5	5	5	4	4	4	1	-	-	-	-	-	-	-	-	-	38
		50-m	-	-	-	1	3	3	3	3	3	2	2	2	-	-	-	-	-	-	-	-	-	-	22
		75-m	-	-	-	1	3	3	2	2	2	2	2	-	-	-	-	-	-	-	-	-	-	-	17

Table 86: Fox River Project – Cash flow - Small equipment, 10-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	679	679	679	679	679	679	679	679	679	679	679	679	679	679	679	158	10,348
Mine Development	-	-	-	-	-	21	54	56	58	66	28	46	58	66	55	25	101	30	52	34	39	-	787
Total Ore Production	-	-	-	-	-	21	733	735	737	745	707	725	737	745	734	704	781	709	731	713	719	158	11,135
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	93	93	93	93	93	93	93	93	93	93	94	93	93	93	93	93	93
Mill Recovery ($/tonne)	-	-	-	-	-	89	83	83	83	83	83	83	83	83	83	83	84	83	83	83	83	83	
Gross Sales (US $)	-	-	-	-	-	1,920	61,034	61,175	61,368	62,065	58,674	60,286	61,368	62,065	61,130	58,393	65,269	58,860	60,805	59,246	59,714	13,113	926,483
Treatment charge (10%)	-	-	-	-	-	192	6,103	6,118	6,137	6,206	5,867	6,029	6,137	6,206	6,113	5,839	6,527	5,886	6,080	5,925	5,971	1,311	92,648
Revenue (US $)	-	-	-	-	-	1,728	54,931	55,058	55,231	55,858	52,807	54,257	55,231	55,858	55,017	52,553	58,742	52,974	54,724	53,322	53,742	11,802	833,835
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	6,757	6,687	6,733	6,289	6,917	6,766	6,557	6,885	6,296	7,062	6,481	5,082	3,204	3,286	2,559	-	87,561
Mining	-	-	-	-	-	-	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	2,859	667	43,552
Hoisting	-	-	-	-	-	-	2,116	2,090	2,112	2,067	2,110	2,118	2,086	2,124	2,062	2,121	2,097	2,016	1,912	1,903	1,879	896	31,710
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	7,152
Backfill	-	-	-	-	-	-	920	920	920	920	920	920	920	920	920	920	920	920	920	920	920	920	14,719
Processing	-	-	-	-	-	271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	9,271	2,163	141,501
Total Operating Cost	-	-	-	-	-	692	22,343	22,248	22,316	21,827	22,498	22,355	22,114	22,479	21,829	22,653	22,049	20,569	18,588	18,660	17,909	5,067	326,196
Gross Income (US $)	-	-	-	-	-	1,035	32,588	32,810	32,914	34,031	30,309	31,902	33,117	33,379	33,189	29,901	36,693	32,405	36,136	34,662	35,833	6,735	507,639
Depreciation (US $)	-	-	-	-	-	1,035	26,178	26,178	26,602	26,721	25,686	543	1,101	1,101	1,282	1,282	1,282	724	1,163	982	982	2,083	144,924
Taxable Income (US $)	-	-	-	-	-	-	6,410	6,632	6,312	7,310	4,623	31,359	32,016	32,278	31,907	28,619	35,411	31,681	34,974	33,680	34,851	4,652	362,715
Income Tax (@ 40.0%)	-	-	-	-	-	-	2,564	2,653	2,525	2,924	1,849	12,544	12,807	12,911	12,763	11,448	14,165	12,672	13,989	13,472	13,940	1,861	145,086
Net After-Tax Profit (US $)	-	-	-	-	-	-	3,846	3,979	3,787	4,386	2,774	18,815	19,210	19,367	19,144	17,171	21,247	19,008	20,984	20,208	20,911	2,791	217,629
Investment (US $)																							
Mine Development	10,756	10,756	20,358	28,068	21,810	7,436	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	99,185
Mining Equipment	-	-	-	-	-	4,686	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,164
Mine Dev. Equipment	-	-	2,123	594	-	-	-	2,123	594	-	-	-	2,123	594	-	-	-	1,530	594	-	-	-	10,273
Plant	-	-	-	-	12,150	12,150	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	24,300
Total Investment	10,756	10,756	22,482	28,662	33,960	24,273	-	2,123	594	-	-	2,787	2,123	1,498	-	-	-	4,316	594	-	-	-	144,924
After-Tax Cash Flow (US $)	-10,756	-10,756	-22,482	-28,662	-33,960	-23,237	30,024	28,034	29,796	31,107	28,460	16,572	18,187	18,969	20,426	18,453	22,528	15,416	21,553	21,190	21,893	4,874	217,629

Table 87: Fox River Project – Cash flow - Small equipment, 10-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	721	721	721	721	721	721	721	721	721	721	721	721	721	721	721	384	11,194
Mine Development	-	-	-	-	-	17	33	34	15	41	25	9	59	21	29	36	8	20	8	24	14	-	394
Total Ore Production	-	-	-	-	-	17	754	754	736	762	745	730	780	742	750	757	728	741	729	744	734	384	11,588
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	90	90	89	90	89	89	90	89	89	90	89	89	89	89	89	89	89
Mill Recovery ($/tonne)	-	-	-	-	-	88	79	79	79	79	79	79	79	79	79	79	79	79	79	79	79	79	
Gross Sales (US $)	-	-	-	-	-	1,476	59,528	59,545	57,912	60,195	58,738	57,407	61,804	58,441	59,155	59,779	57,262	58,329	57,313	58,651	57,791	30,173	913,500
Treatment charge (10%)	-	-	-	-	-	148	5,953	5,955	5,791	6,020	5,874	5,741	6,180	5,844	5,915	5,978	5,726	5,833	5,731	5,865	5,779	3,017	91,350
Revenue (US $)	-	-	-	-	-	1,329	53,575	53,591	52,121	54,176	52,865	51,667	55,624	52,597	53,239	53,801	51,536	52,496	51,582	52,786	52,012	27,156	822,150
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	5,190	5,078	5,423	3,929	3,608	3,415	3,245	3,511	2,338	1,565	1,578	1,602	1,646	1,564	415	-	44,106
Mining	-	-	-	-	-	-	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	2,904	1,549	45,115
Hoisting	-	-	-	-	-	-	2,093	2,080	2,107	2,050	2,015	1,994	2,014	2,002	1,942	1,918	1,881	1,912	1,887	1,904	1,856	1,271	30,925
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	7,152
Backfill	-	-	-	-	-	-	934	934	934	934	934	934	934	934	934	934	934	934	934	934	934	934	14,937
Processing	-	-	-	-	-	209	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	9,396	5,011	146,164
Total Operating Cost	-	-	-	-	-	629	20,938	20,813	21,185	19,634	19,278	19,063	18,914	19,167	17,934	17,138	17,114	17,169	17,188	17,123	15,926	9,186	288,398
Gross Income (US $)	-	-	-	-	-	699	32,637	32,778	30,937	34,542	33,586	32,603	36,710	33,430	35,305	36,663	34,422	35,327	34,394	35,663	36,086	17,970	533,753
Depreciation (US $)	-	-	-	-	-	699	26,153	26,153	26,578	26,578	25,878	425	982	982	1,163	1,163	1,163	606	1,163	982	982	1,964	143,612
Taxable Income (US $)	-	-	-	-	-	-	6,484	6,625	4,359	7,964	7,708	32,179	35,728	32,448	34,142	35,500	33,259	34,721	33,231	34,681	35,104	16,006	390,140
Income Tax (@ 40.0%)	-	-	-	-	-	-	2,594	2,650	1,744	3,186	3,083	12,871	14,291	12,979	13,657	14,200	13,304	13,889	13,292	13,872	14,042	6,402	156,056
Net After-Tax Profit (US $)	-	-	-	-	-	-	3,891	3,975	2,615	4,779	4,625	19,307	21,437	19,469	20,485	21,300	19,956	20,833	19,938	20,809	21,063	9,603	234,084
Investment (US $)																							
Mine Development	10,756	10,756	20,358	28,068	22,091	6,705	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	98,736
Mining Equipment	-	-	-	-	-	4,701	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,179
Mine Dev. Equipment	-	-	2,123	594	-	-	-	2,123	-	-	-	-	2,123	-	-	-	-	2,123	-	-	-	-	9,086
Plant	-	-	-	-	12,306	12,306	-	-	-	-		-	-	-	-	-	-	-	-	-	-	-	24,611
Total Investment	10,756	10,756	22,482	28,662	34,397	23,712	-	2,123	-	-	-	2,787	2,123	905	-	-	-	4,910	-	-	-	-	143,612
After-Tax Cash Flow (US $)	-10,756	-10,756	-22,482	-28,662	-34,397	-23,012	30,044	28,005	29,193	31,356	30,503	16,945	20,295	19,546	21,648	22,463	21,118	16,528	21,101	21,791	22,045	11,567	234,084

Table 88: Fox River Project – Cash flow - Small equipment, 10-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	719	719	719	719	719	719	719	719	719	719	719	719	719	719	719	128	10,920
Mine Development	-	-	-	-	-	17	17	18	14	20	33	15	33	2	29	33	17	15	-	-	-	-	262
Total Ore Production	-	-	-	-	-	17	736	738	733	739	752	734	753	721	748	753	736	735	719	719	719	128	11,182
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	84	84	84	84	84	84	84	83	84	84	84	84	83	83	83	83	84
Mill Recovery ($/tonne)	-	-	-	-	-	86	72	72	72	72	73	72	73	72	72	73	72	72	72	72	72	72	
Gross Sales (US $)	-	-	-	-	-	1,444	53,195	53,331	52,940	53,466	54,570	53,009	54,639	51,886	54,249	54,639	53,195	53,076	51,750	51,750	51,750	9,200	808,091
Treatment charge (10%)	-	-	-	-	-	144	5,319	5,333	5,294	5,347	5,457	5,301	5,464	5,189	5,425	5,464	5,319	5,308	5,175	5,175	5,175	920	80,809
Revenue (US $)	-	-	-	-	-	1,300	47,875	47,997	47,646	48,120	49,113	47,708	49,175	46,698	48,824	49,175	47,875	47,768	46,575	46,575	46,575	8,280	727,282
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	3,507	3,380	3,491	3,490	3,266	2,148	1,626	1,642	1,477	1,626	1,640	1,439	-	-	-	-	28,730
Mining	-	-	-	-	-	-	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	2,903	516	44,062
Hoisting	-	-	-	-	-	-	1,989	1,994	2,024	2,003	1,989	1,956	1,918	1,876	1,911	1,918	1,896	1,885	1,829	1,829	1,829	845	29,690
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	7,152
Backfill	-	-	-	-	-	-	928	928	928	928	928	928	928	928	928	928	928	928	928	928	928	928	14,854
Processing	-	-	-	-	-	213	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	9,349	1,662	142,107
Total Operating Cost	-	-	-	-	-	633	19,097	18,975	19,115	19,094	18,856	17,705	17,144	17,120	16,989	17,144	17,136	16,925	15,430	15,430	15,430	4,372	266,595
Gross Income (US $)	-	-	-	-	-	666	28,778	29,023	28,531	29,025	30,257	30,004	32,031	29,578	31,835	32,031	30,739	30,843	31,145	31,145	31,145	3,908	460,686
Depreciation (US $)	-	-	-	-	-	666	26,068	26,068	26,374	26,493	25,827	425	982	982	1,163	1,163	1,163	606	857	557	557	1,115	141,066
Taxable Income (US $)	-	-	-	-	-	-	2,710	2,955	2,157	2,532	4,430	29,579	31,049	28,596	30,672	30,868	29,576	30,238	30,288	30,588	30,588	2,793	319,620
Income Tax (@ 40.0%)	-	-	-	-	-	-	1,084	1,182	863	1,013	1,772	11,832	12,420	11,438	12,269	12,347	11,831	12,095	12,115	12,235	12,235	1,117	127,848
Net After-Tax Profit (US $)	-	-	-	-	-	-	1,626	1,773	1,294	1,519	2,658	17,747	18,629	17,158	18,403	18,521	17,746	18,143	18,173	18,353	18,353	1,676	191,772
Investment (US $)																							
Mine Development	10,756	10,756	20,358	28,068	22,038	6,459	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	98,436
Mining Equipment	-	-	-	-	-	4,695	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,173
Mine Dev. Equipment	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	594	-	-	-	-	-	-	-	-	6,963
Plant	-	-	-	-	12,247	12,247	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24,493
Total Investment	10,756	10,756	22,482	28,662	34,285	23,401	-	1,530	594	-	-	2,787	1,530	1,498	-	-	-	2,787	-	-	-	-	141,066
After-Tax Cash Flow (US $)	-10,756	-10,756	-22,482	-28,662	-34,285	-22,734	27,694	26,312	27,075	28,012	28,485	15,385	18,082	16,642	19,566	19,684	18,909	15,962	19,030	18,910	18,910	2,791	191,772

Table 89: Fox River Project – Cash flow - Small equipment, 15-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	703	703	703	703	703	703	703	703	703	703	703	703	703	500	-	-	9,642
Mine Development	-	-	-	-	-	13	67	58	23	56	60	46	59	39	57	43	52	29	43	-	-	-	645
Total Ore Production	-	-	-	-	-	13	771	761	726	759	763	749	762	743	760	746	755	733	746	500	-	-	10,287
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	94	94	94	94	94	94	94	94	94	94	94	94	94	94	-	-	94
Mill Recovery ($/tonne)	-	-	-	-	-	90	85	85	85	85	85	85	85	85	85	85	85	85	85	84	-	-	
Gross Sales (US $)	-	-	-	-	-	1,177	65,407	64,520	61,407	64,346	64,702	63,462	64,642	62,877	64,459	63,175	63,992	61,993	63,227	42,201	-	-	871,589
Treatment charge (10%)	-	-	-	-	-	118	6,541	6,452	6,141	6,435	6,470	6,346	6,464	6,288	6,446	6,318	6,399	6,199	6,323	4,220	-	-	87,159
Revenue (US $)	-	-	-	-	-	1,059	58,866	58,068	55,267	57,912	58,232	57,116	58,178	56,589	58,013	56,858	57,593	55,794	56,905	37,980	-	-	784,430
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	6,076	5,814	6,894	6,932	5,788	5,059	4,851	5,114	4,948	4,999	3,436	3,227	2,386	-	-	-	65,523
Mining	-	-	-	-	-	-	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,885	2,052	-	-	39,560
Hoisting	-	-	-	-	-	-	2,068	2,089	2,164	2,179	2,071	2,061	2,034	2,062	2,032	2,053	1,962	1,930	1,915	1,464	-	-	28,084
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	-	-	6,311
Backfill	-	-	-	-	-	-	798	798	798	798	798	798	798	798	798	798	798	798	798	798	-	-	11,175
Processing	-	-	-	-	-	161	9,445	9,445	9,445	9,445	9,445	9,445	9,445	9,445	9,445	9,445	9,445	9,445	9,445	6,716	-	-	129,658
Total Operating Cost	-	-	-	-	-	581	21,693	21,452	22,607	22,661	21,407	20,668	20,434	20,725	20,529	20,601	18,947	18,706	17,850	11,451	-	-	280,311
Gross Income (US $)	-	-	-	-	-	478	37,173	36,616	32,659	35,251	36,825	36,447	37,744	35,864	37,485	36,257	38,646	37,088	39,055	26,530	-	-	504,119
Depreciation (US $)	-	-	-	-	-	478	26,517	26,517	26,941	27,060	26,582	543	1,101	1,101	1,282	1,282	1,282	724	857	557	-	-	142,823
Taxable Income (US $)	-	-	-	-	-	-	10,656	10,099	5,718	8,191	10,243	35,904	36,644	34,764	36,203	34,975	37,364	36,364	38,198	25,972	-	-	361,296
Income Tax (@ 40.0%)	-	-	-	-	-	-	4,262	4,040	2,287	3,276	4,097	14,362	14,657	13,905	14,481	13,990	14,946	14,546	15,279	10,389	-	-	144,518
Net After-Tax Profit (US $)	-	-	-	-	-	-	6,394	6,060	3,431	4,915	6,146	21,542	21,986	20,858	21,722	20,985	22,419	21,818	22,919	15,583	-	-	216,777
Investment (US $)																							
Mine Development	10,756	10,756	21,644	27,983	21,700	7,589	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	100,429
Mining Equipment	-	-	-	-	-	4,707	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,185
Mine Dev. Equipment	-	-	2,123	594	-	-	-	2,123	594	-	-	-	2,123	594	-	-	-		-	-	-	-	8,150
Plant	-	-	-	-	12,366	12,366	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	24,731
Total Investment	10,756	10,756	23,767	28,576	34,066	24,662	-	2,123	594		-	2,787	2,123	1,498	-	-	-	2,787	-	-	-	-	144,495
After-Tax Cash Flow (US $)	-10,756	-10,756	-23,767	-28,576	-34,066	-24,184	32,910	30,453	29,779	31,975	32,728	19,299	20,964	20,461	23,003	22,267	23,700	19,756	23,776	16,141	-	-	215,105

Table 90: Fox River Project – Cash flow - Small equipment, 15-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	726	726	726	726	726	726	726	726	726	726	726	726	726	726	645	-	10,803
Mine Development	-	-	-	-	-	15	33	12	20	26	37	28	37	34	15	33	18	3	11	-	-	-	322
Total Ore Production	-	-	-	-	-	15	759	738	745	752	762	753	762	760	741	759	744	728	737	726	645	-	11,126
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	91	91	91	91	91	91	91	91	91	91	91	91	91	91	91	-	91
Mill Recovery ($/tonne)	-	-	-	-	-	89	81	81	81	81	81	81	81	81	81	81	81	81	81	81	81	-	
Gross Sales (US $)	-	-	-	-	-	1,337	61,515	59,654	60,301	60,877	61,805	61,000	61,805	61,578	59,889	61,515	60,178	58 787	59,542	58,552	52,046	-	900,381
Treatment charge (10%)	-	-	-	-	-	134	6,152	5,965	6,030	6,088	6,180	6,100	6,180	6,158	5,989	6,152	6,018	5 879	5,954	5,855	5,205	-	90,038
Revenue (US $)	-	-	-	-	-	1,203	55,364	53,689	54,271	54,789	55,624	54,900	55,624	55,420	53,900	55,364	54,160	52 908	53,588	52,697	46,841	-	810,343
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	5,130	3,856	3,389	3,303	3,408	3,280	2,305	1,564	1,611	1,619	1,626	1 497	299	-	-	-	32,886
Mining	-	-	-	-	-	-	2,910	2,910	2,910	2,910	2,910	2,910	2,910	2,910	2,910	2,910	2,910	2.910	2,910	2,910	2,586	-	43,323
Hoisting	-	-	-	-	-	-	2,071	2,030	2,004	1,977	2,020	2,004	1,947	1,923	1,913	1,927	1,905	1.884	1,858	1,839	1,705	-	29,008
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	-	6,731
Backfill	-	-	-	-	-	-	796	796	796	796	796	796	796	796	796	796	796	796	796	796	796	-	11,945
Processing	-	-	-	-	-	187	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	9,424	8,377	-	140,500
Total Operating Cost	-	-	-	-	-	608	20,752	19,437	18,944	18,831	18,979	18,836	17,803	17,038	17,074	17,097	17,082	16.932	15,708	15,390	13,886	-	264,395
Gross Income (US $)	-	-	-	-	-	596	34,612	34,251	35,327	35,958	36,645	36,064	37,821	38,383	36,826	38,267	37,078	35,977	37,880	37,307	32,956	-	545,948
Depreciation (US $)	-	-	-	-	-	596	26,262	26,262	26,568	26,687	26,091	425	982	982	1,163	1,163	1,163	606	857	557	557	-	140,920
Taxable Income (US $)	-	-	-	-	-	-	8,350	7,989	8,759	9,272	10,554	35,639	36,839	37,401	35,663	37,104	35,916	35,371	37,023	36,749	32,399	-	405,028
Income Tax (@ 40.0%)	-	-	-	-	-	-	3,340	3,196	3,504	3,709	4,222	14,256	14,736	14,960	14,265	14,842	14,366	14,148	14,809	14,700	12,959	-	162,011
Net After-Tax Profit (US $)	-	-	-	-	-	-	5,010	4,794	5,255	5,563	6,332	21,384	22,103	22,440	21,398	22,263	21,549	21,223	22,214	22,050	19,439	-	243,017
Investment (US $)																							
Mine Development	10,756	10,756	21,644	27,983	21,458	6,612	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	99,209
Mining Equipment	-	-	-	-	-	4,704	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,182
Mine Dev. Equipment	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	594	-	-	-		-	-	-	-	6,963
Plant	-	-	-	-	12,340	12,340	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	24,680
Total Investment	10,756	10,756	23,767	28,576	33,798	23,656	-	1,530	594	-	-	2,787	1,530	1,498	-	-	-	2,787	-	-	-	-	142,034
After-Tax Cash Flow (US $)	-10,756	-10,756	-23,767	-28,576	-33,798	-23,061	31,272	29,526	31,230	32,250	32,423	19,022	21,556	21,924	22,561	23,426	22,712	19,041	23,071	22,607	19,996	-	241,902

Table 91: Fox River Project – Cash flow - Small equipment, 15-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	742	742	742	742	742	742	742	742	742	742	742	742	742	742	270	-	10,664
Mine Development	-	-	-	-	-	15	33	32	6	14	-	30	34	37	-	22	18		-	-	-	-	242
Total Ore Production	-	-	-	-		15	776	775	748	757	742	773	777	779	742	765	760	742	742	742	270	-	10,906
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	87	87	86	86	86	87	87	87	86	87	87	86	86	86	86	-	87
Mill Recovery ($/tonne)	-	-	-	-	-	87	76	76	75	75	75	76	76	76	75	76	75	75	75	75	75	-	
Gross Sales (US $)	-	-	-	-	-	1,313	58,710	58,600	56,315	57,054	55,800	58,426	58,772	58,994	55,800	57,740	57,346	55,800	55,800	55,800	20,302	-	822,574
Treatment charge (10%)	-	-	-	-	-	131	5,871	5,860	5,631	5,705	5,580	5,843	5,877	5,899	5,580	5,774	5,735	5,580	5,580	5,580	2,030	-	82,257
Revenue (US $)	-	-	-	-	-	1,182	52,839	52,740	50,683	51,349	50,220	52,584	52,895	53,095	50,220	51,966	51,612	50,220	50,220	50,220	18,272	-	740,317
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	3,884	3,352	3,579	2,680	1,562	1,612	1,598	1,626	1,640	1,570	745		-	-	-	-	23,847
Mining	-	-	-	-	-	-	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	2,928	1,065	-	42,061
Hoisting	-	-	-	-	-	-	2,020	2,035	2,061	2,008	1,915	1,953	1,960	1,957	1,911	1,939	1,909	1,867	1,867	1,867	1,081	-	28,350
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	-	6,731
Backfill	-	-	-	-	-	-	807	807	807	807	807	807	807	807	807	807	807	807	807	807	807	-	12,099
Processing	-	-	-	-	-	185	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	9,535	3,469	-	137,143
Total Operating Cost	-	-	-	-	-	606	19,595	19,078	19,330	18,379	17,167	17,255	17,249	17,273	17,241	17,199	16,344	15,558	15,558	15,558	6,843	-	250,231
Gross Income (US $)	-	-	-	-	-	576	33,245	33,662	31,353	32,970	33,053	35,328	35,646	35,822	32,979	34,767	35,268	34,662	34,662	34,662	11,428	-	490,086
Depreciation (US $)	-	-	-	-	-	576	26,221	26,221	26,527	26,646	26,070	425	982	982	1,163	1,163	1,163	606	857	557	557	-	140,716
Taxable Income (US $)	-	-	-	-	-	-	7,023	7,441	4,826	6,324	6,983	34,904	34,664	34,840	31,816	33,604	34,105	34,057	33,805	34,105	10,871	-	349,370
Income Tax (@ 40.0%)	-	-	-	-	-	-	2,809	2,977	1,931	2,530	2,793	13,962	13,866	13,936	12,727	13,442	13,642	13,623	13,522	13,642	4,348	-	139,748
Net After-Tax Profit (US $)	-	-	-	-	-	-	4,214	4,465	2,896	3,795	4,190	20,942	20,798	20,904	19,090	20,162	20,463	20,434	20,283	20,463	6,523	-	209,622
Investment (US $)																							
Mine Development	10,756	10,756	21 588	27,871	21,505	6,240	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	98,717
Mining Equipment	-	-	-	-	-	4,717	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,195
Mine Dev. Equipment	-	-	2,123	594	-	-	-	1,530	594	-		-	1,530	594	-	-	-	-	-	-	-	-	6,963
Plant	-	-	-	-	12,478	12,478	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24,955
Total Investment	10,756	10,756	23,711	28,465	33,982	23,435	-	1,530	594	-		2,787	1,530	1,498	-	-	-	2,787	-	-	-	-	141,830
After-Tax Cash Flow (US $)	-10,756	-10,756	-23,711	-28,465	-33,982	-22,859	30,435	29,156	28,829	30,440	30,260	18,580	20,251	20,388	20,253	21,325	21,626	18,253	21,140	21,020	7,080	-	208,508

Table 92: Fox River Project – Cash flow - Small equipment, 20-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	753	753	753	753	753	753	753	753	753	753	753	728	-	-	-	-	9,007
Mine Development	-	-	-	-	-	14	72	48	39	30	64	45	50	41	66	61	22	-	-	-	-	-	553
Total Ore Production	-	-	-	-	-	14	824	800	792	783	817	798	803	794	819	814	774	728	-	-	-	-	9,559
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	95	95	95	95	95	95	95	95	95	95	95	94	-	-	-	-	95
Mill Recovery ($/tonne)	-	-	-	-	-	90	85	85	85	85	85	85	85	85	85	85	85	85	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,288	70,464	68,273	67,540	66,698	69,789	68,048	68,501	67,721	69,932	69,505	65,942	61,864	-	-	-	-	815,564
Treatment charge (10%)	-	-	-	-	-	129	7,046	6,827	6,754	6,670	6,979	6,805	6,850	6,772	6,993	6,950	6,594	6,186	-	-	-	-	81,556
Revenue (US $)	-	-	-	-	-	1,159	63,417	61,446	60,786	60,028	62,810	61,244	61,651	60,949	62,939	62,554	59,348	55,677	-	-	-	-	734,008
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	6,430	4,933	4,911	5,167	4,841	5,094	5,024	5,078	4,168	3,247	2,201	159	-	-	-	-	51,252
Mining	-	-	-	-	-	-	2,939	2,939	2,939	2,939	2,939	2,939	2,939	2,939	2,939	2,939	2,939	2,841	-	-	-	-	35,175
Hoisting	-	-	-	-	-	-	2,187	2,140	2,113	2,156	2,108	2,142	2,115	2,130	2,083	2,045	1,963	1,847	-	-	-	-	25,029
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	-	-	-	-	5,469
Backfill	-	-	-	-	-	-	744	744	744	744	744	744	744	744	744	744	744	744	-	-	-	-	8,930
Processing	-	-	-	-	-	171	9,841	9,841	9,841	9,841	9,841	9,841	9,841	9,841	9,841	9,841	9,841	9,513	-	-	-	-	117,936
Total Operating Cost	-	-	-	-	-	592	22,562	21,018	20,969	21,268	20,894	21,182	21,085	21,154	20,196	19,237	18,110	15,525	-	-	-	-	243,792
Gross Income (US $)	-	-	-	-	-	568	40,855	40,427	39,816	38,761	41,915	40,062	40,566	39,795	42,743	43,317	41,238	40,152	-	-	-	-	490,216
Depreciation (US $)	-	-	-	-	-	568	26,780	26,780	27,204	27,323	26,755	543	1,101	1,101	1,282	1,282	1,282	724	-	-	-	-	142,724
Taxable Income (US $)	-	-	-	-	-	-	14,075	13,648	12,612	11,438	15,160	39,519	39,465	38,694	41,461	42,036	39,956	39,428	-	-	-	-	347,492
Income Tax (@ 40.0%)	-	-	-	-	-	-	5,630	5,459	5,045	4,575	6,064	15,807	15,786	15,478	16,585	16,814	15,983	15,771	-	-	-	-	138,997
Net After-Tax Profit (US $)	-	-	-	-	-	-	8,445	8,189	7,567	6,863	9,096	23,711	23,679	23,217	24 877	25,221	23,974	23,657	-	-	-	-	208,495
Investment (US $)																			-	-			
Mine Development	10,756	10,756	22,278	27,945	21,358	7,620	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	100,714
Mining Equipment	-	-	-	-	-	4,753	-	-	-	-	-	2,787	-	905	-	-	-	-	-	-	-	-	8,445
Mine Dev. Equipment	-	-	2,123	594	-	-	-	2,123	594	-	-	-	2,123	594	-	-	-	-	-	-	-	-	8,150
Plant	-	-	-	-	12,857	12,857	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	25,715
Total Investment	10,756	10,756	24,401	28,538	34,215	25,231	-	2,123	594	-	-	2,787	2,123	1,498	-	-	-	-	-	-	-	-	143,024
After-Tax Cash Flow (US $)	-10,756	-10,756	-24,401	-28,538	-34,215	-24,663	35,225	32,845	34,178	34,186	35,851	21,468	22,657	22,819	26,158	26,503	25,255	24,381	-	-	-	-	208,196

Table 93: Fox River Project – Cash flow - Small equipment, 20-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	740	740	740	740	740	740	740	740	740	740	740	740	740	740	136	-	10,492
Mine Development	-	-	-	-	-	14	39	39	-	19	12	17	26	31	35	-	24	20	-	-	-	-	276
Total Ore Production	-	-	-	-	-	14	779	779	740	759	752	757	766	771	774	740	764	760	740	740	136	-	10,768
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	92	92	92	92	92	92	92	92	92	92	92	92	92	92	92	-	92
Mill Recovery ($/tonne)	-	-	-	-	-	89	82	82	81	82	82	82	82	82	82	81	82	82	81	81	81	-	
Gross Sales (US $)	-	-	-	-	-	1,213	63,762	63,789	60,285	61,963	61,356	61,821	62,630	63,061	63,357	60,285	62,427	62,051	60,285	60,285	11,052	-	879,620
Treatment charge (10%)	-	-	-	-	-	121	6,376	6,379	6,028	6,196	6,136	6,182	6,263	6,306	6,336	6,028	6,243	6,205	6,028	6,028	1,105	-	87,962
Revenue (US $)	-	-	-	-	-	1,092	57,385	57,410	54,256	55,766	55,220	55,639	56,367	56,755	57,022	54,256	56,185	55,845	54,256	54,256	9,947	-	791,658
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	3,251	3,380	3,799	2,518	1,626	1,633	1,634	1,648	1,626	1,640	1,634	808	-	-	-	-	25,196
Mining	-	-	-	-	-	-	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	2,925	536	-	41,490
Hoisting	-	-	-	-	-	-	1,978	2,020	2,080	2,005	1,921	1,926	1,940	1,956	1,946	1,907	1,937	1,908	1,863	1,863	858	-	28,108
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	-	6,731
Backfill	-	-	-	-	-	-	718	718	718	718	718	718	718	718	718	718	718	718	718	718	718	-	10,770
Processing	-	-	-	-	-	167	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	9,524	1,746	-	135,246
Total Operating Cost	-	-	-	-	-	587	18,818	18,987	19,467	18,111	17,134	17,147	17,162	17,192	17,159	17,134	17,158	16,304	15,451	15,451	4,279	-	247,541
Gross Income (US $)	-	-	-	-	-	504	38,568	38,423	34,789	37,655	38,086	38,492	39,205	39,563	39,862	37,122	39,026	39,541	38,805	38,805	5,668	-	544,118
Depreciation (US $)	-	-	-	-	-	504	26,347	26,347	26,653	26,771	26,267	425	982	982	1,163	1,163	1,163	606	857	557	557	-	141,344
Taxable Income (US $)	-	-	-	-	-	-	12,221	12,076	8,137	10,884	11,819	38,068	38,223	38,581	38,699	35,959	37,863	38,936	37,949	38,248	5,111	-	402,774
Income Tax (@ 40.0%)	-	-	-	-	-	-	4,888	4,831	3,255	4,354	4,728	15,227	15,289	15,432	15,480	14,384	15,145	15,574	15,179	15,299	2,044	-	161,110
Net After-Tax Profit (US $)	-	-	-	-	-	-	7,333	7,246	4,882	6,530	7,091	22,841	22,934	23,149	23,220	21,576	22,718	23,361	22,769	22,949	3,067	-	241,664
Investment (US $)																							
Mine Development	10,756	10,756	22,278	27,945	21,801	5,837	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	99,374
Mining Equipment	-	-	-	-	-	4,716	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,194
Mine Dev. Equipment	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	594	-	-	-	-	-	-	-	-	6,963
Plant	-	-	-	-	12,464	12,464	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24,928
Total Investment	10,756	10,756	24,401	28,538	34,265	23,017	-	1,530	594	-	-	2,787	1,530	1,498	-	-	-	2,787	-	-	-	-	142,458
After-Tax Cash Flow (US $)	-10,756	-10,756	-24,401	-28,538	-34,265	-22,512	33,679	32,063	30,941	33,302	33,358	20,478	22,386	22,633	24,382	22,739	23,881	21,180	23,626	23,506	3,624	-	240,550

Table 94: Fox River Project – Cash flow - Small equipment, 20-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	743	743	743	743	743	743	743	743	743	743	743	743	743	743	111	-	10,514
Mine Development	-	-	-	-	-	11	35	-	31	15	-	-	29	20	26	18	-	-	-	-	-	-	184
Total Ore Production	-	-	-	-	-	11	778	743	774	758	743	743	772	763	769	761	743	743	743	743	111	-	10,698
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	88	87	88	88	87	87	88	88	88	88	87	87	87	87	87	-	87
Mill Recovery ($/tonne)	-	-	-	-	-	87	77	76	77	77	76	76	77	77	77	77	76	76	76	76	76	-	
Gross Sales (US $)	-	-	-	-	-	994	59,742	56,720	59,424	58,059	56,720	56,720	59,225	58,430	59,026	58,258	56,720	56,720	56,720	56,720	8,508	-	818,707
Treatment charge (10%)	-	-	-	-	-	99	5,974	5,672	5,942	5,806	5,672	5,672	5,922	5,843	5,903	5,826	5,672	5,672	5,672	5,672	851	-	81,871
Revenue (US $)	-	-	-	-	-	895	53,768	51,048	53,481	52,253	51,048	51,048	53,302	52,587	53,124	52,432	51,048	51,048	51,048	51,048	7,657	-	736,836
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	3,147	1,595	1,638	1,809	1,899	1,633	1,659	1,626	1,634	675	-	-	-	-	-	-	17,315
Mining	-	-	-	-	-	-	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	2,929	439	-	41,444
Hoisting	-	-	-	-	-	-	1,979	1,896	1,948	1,972	1,977	1,926	1,959	1,935	1,946	1,907	1,868	1,868	1,868	1,868	817	-	27,736
Ventilation	-	-	-	-	-	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	421	-	6,731
Backfill	-	-	-	-	-	-	717	717	717	717	717	717	717	717	717	717	717	717	717	717	717	-	10,757
Processing	-	-	-	-	-	139	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	9,513	1,427	-	134,753
Total Operating Cost	-	-	-	-	-	560	18,706	17,072	17,166	17,361	17,456	17,139	17,198	17,141	17,160	16,162	15,448	15,448	15,448	15,448	3,822	-	238,736
Gross Income (US $)	-	-	-	-	-	335	35,061	33,976	36,315	34,893	33,592	33,909	36,104	35,446	35,964	36,270	35,600	35,600	35,600	35,600	3,836	-	498,100
Depreciation (US $)	-	-	-	-	-	335	26,087	26,087	26,393	26,511	26,177	425	982	982	1,044	1,044	1,044	487	738	557	557	-	139,450
Taxable Income (US $)	-	-	-	-	-	-	8,975	7,890	9,923	8,381	7,415	33,484	35,122	34,464	34,920	35,226	34,556	35,113	34,862	35,042	3,278	-	358,650
Income Tax (@ 40.0%)	-	-	-	-	-	-	3,590	3,156	3,969	3,352	2,966	13,394	14,049	13,786	13,968	14,090	13,822	14,045	13,945	14,017	1,311	-	143,460
Net After-Tax Profit (US $)	-	-	-	-	-	-	5,385	4,734	5,954	5,029	4,449	20,091	21,073	20,679	20,952	21,136	20,733	21,068	20,917	21,025	1,967	-	215,190
Investment (US $)																							
Mine Development	10,756	10,756	22,278	27,945	20,360	6,005	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	98,101
Mining Equipment	-	-	-	-	-	4,715	-	-	-	-	-	2,787	-	905	-	-	-	2,787	-	-	-	-	11,193
Mine Dev. Equipment	-	-	2,123	594	-	-	-	1,530	594	-	-	-	1,530	-	-	-	-	-	-	-	-	-	6,370
Plant	-	-	-	-	12,451	12,451	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	24,902
Total Investment	10,756	10,756	24,401	28,538	32,811	23,170	-	1,530	594	-	-	2,787	1,530	905	-	-	-	2,787	-	-	-	-	140,565
After-Tax Cash Flow (US $)	-10,756	-10,756	-24,401	-28,538	-32,811	-22,835	31,471	29,291	31,753	31,540	30,626	17,728	20,526	20,756	21,996	22,180	21,778	18,768	21,655	21,583	2,524	-	214,075

Table 95: Fox River Project – Cash flow - Mid-Size equipment, 10-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	953	953	953	953	953	953	953	953	953	953	953	222	-	-	-	-	10,701
Mine Development	-	-	-	-	-	46	71	79	79	83	81	79	83	83	76	91	31	-	-	-	-	-	884
Total Ore Production	-	-	-	-	-	46	1,023	1,032	1,032	1,036	1,034	1,032	1,036	1,036	1,029	1,044	984	222	-	-	-	-	11,585
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	90	90	90	90	90	90	90	90	90	90	89	89	-	-	-	-	90
Mill Recovery ($/tonne)	-	-	-	-	-	88	79	79	79	79	79	79	79	79	79	80	79	79	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	4,038	81,172	81,951	81,951	82,282	82,100	81,951	82,282	82,271	81,685	82,976	77,666	17,484	-	-	-	-	919,809
Treatment charge (10%)	-	-	-	-	-	404	8,117	8,195	8,195	8,228	8,210	8,195	8,228	8,227	8,169	8,298	7,767	1,748	-	-	-	-	91,981
Revenue (US $)	-	-	-	-	-	3,634	73,055	73,756	73,756	74,054	73,890	73,756	74,054	74,044	73,517	74,679	69,900	15,736	-	-	-	-	827,828
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	8,454	9,059	8,701	9,001	9,143	8,814	8,993	9,367	8,082	6,130	2,513	-	-	-	-	-	88,257
Mining	-	-	-	-	-	-	3,428	3,428	3,428	3,428	3,428	3,428	3,428	3,428	3,428	3,428	3,428	600	-	-	-	-	38,509
Hoisting	-	-	-	-	-	-	2,642	2,642	2,620	2,639	2,642	2,632	2,636	2,656	2,582	2,469	2,269	691	-	-	-	-	29,420
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511	511	-	-	-	-	6,649
Backfill	-	-	-	-	-	-	1,173	1,173	1,173	1,173	1,173	1,173	1,173	1,173	1,173	1,173	1,173	1,173	-	-	-	-	14,075
Processing	-	-	-	-	-	518	11,585	11,585	11,585	11,585	11,585	11,585	11,585	11,585	11,585	11,585	11,585	2,703	-	-	-	-	130,661
Total Operating Cost	-	-	-	-	-	1,029	27,794	28,399	28,020	28,338	28,483	28,144	28,327	28,721	27,362	25,297	21,480	6,179	-	-	-	-	307,571
Gross Income (US $)	-	-	-	-	-	2,605	45,261	45,357	45,736	45,716	45,407	45,612	45,727	45,323	46,154	49,382	48,420	9,557	-	-	-	-	520,257
Depreciation (US $)	-	-	-	-	-	2,605	31,555	31,555	32,033	32,173	29,708	758	1,533	1,533	1,598	1,598	1,598	823	-	-	-	-	169,069
Taxable Income (US $)	-	-	-	-	-	-	13,707	13,802	13,702	13,542	15,698	44,854	44,194	43,791	44,557	47,784	46,822	8,733	-	-	-	-	351,187
Income Tax (@ 40.0%)	-	-	-	-	-	-	5,483	5,521	5,481	5,417	6,279	17,942	17,678	17,516	17,823	19,114	18,729	3,493	-	-	-	-	140,475
Net After-Tax Profit (US $)	-	-	-	-	-	-	8,224	8,281	8,221	8,125	9,419	26,912	26,516	26,275	26,734	28,671	28,093	5,240	-	-	-	-	210,712
Investment (US $)																							
Mine Development	11,169	11,169	21,757	35,942	26,070	11,635	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	117,741
Mining Equipment	-	-	-	-	-	6,164	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,059
Mine Dev. Equipment	-	-	2,393	700	700	-	-	2,393	700	700	-	-	2,393	-	700	-	-	-	-	-	-	-	10,677
Plant	-	-	-	-	15,039	15,039	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30,078
Total Investment	11,169	11,169	24,150	36,641	41,808	32,838	-	2,393	700	700	-	3,871	2,393	1,025	700	-	-	-	-	-	-	-	169,554
After-Tax Cash Flow (US $)	-11,169	-11,169	-24,150	-36,641	-41,808	-30,233	39,779	37,443	39,555	39,599	39,127	23,600	25,656	26,783	27,632	30,268	29,691	6,063	-	-	-	-	210,228

Table 96: Fox River Project – Cash flow - Mid-Size equipment, 10-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	1,005	536	-	-	-	-	11,588
Mine Development	-	-	-	-	-	19	38	37	38	39	60	46	16	27	78	43	-	-	-	-	-	-	442
Total Ore Production	-	-	-	-	-	19	1,043	1,042	1,043	1,044	1,065	1,051	1,021	1,031	1,082	1,048	1,005	536	-	-	-	-	12,030
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	85	85	85	85	85	85	84	84	85	85	84	84	-	-	-	-	85
Mill Recovery ($/tonne)	-	-	-	-	-	87	73	73	73	73	74	73	73	73	74	73	73	73	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,653	76,388	76,311	76,388	76,473	78,300	77,058	74,496	75,383	79,791	76,819	73,081	38,977	-	-	-	-	881,119
Treatment charge (10%)	-	-	-	-	-	165	7,639	7,631	7,639	7,647	7,830	7,706	7,450	7,538	7,979	7,682	7,308	3,898	-	-	-	-	88,112
Revenue (US $)	-	-	-	-	-	1,488	68,749	68,680	68,749	68,826	70,470	69,352	67,046	67,845	71,812	69,137	65,773	35,079	-	-	-	-	793,007
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	7,351	6,378	5,502	5,440	4,240	3,508	3,584	3,394	3,489	3,257	198	-	-	-	-	-	46,340
Mining	-	-	-	-	-	-	3,486	3,486	3,486	3,486	3,486	3,486	3,486	3,486	3,486	3,486	3,486	1,859	-	-	-	-	40,207
Hoisting	-	-	-	-	-	-	2,624	2,599	2,552	2,539	2,484	2,427	2,439	2,401	2,458	2,396	2,261	1,498	-	-	-	-	28,678
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511	511	-	-	-	-	6,649
Backfill	-	-	-	-	-	-	1,184	1,184	1,184	1,184	1,184	1,184	1,184	1,184	1,184	1,184	1,184	1,184	-	-	-	-	14,206
Processing	-	-	-	-	-	214	11,684	11,684	11,684	11,684	11,684	11,684	11,684	11,684	11,684	11,684	11,684	6,003	-	-	-	-	134,744
Total Operating Cost	-	-	-	-	-	726	26,841	25,843	24,920	24,845	23,590	22,800	22,888	22,660	22,813	22,519	19,325	11,056	-	-	-	-	270,825
Gross Income (US $)	-	-	-	-	-	763	41,909	42,837	43,830	43,981	46,880	46,552	44,158	45,185	48,999	46,618	46,448	24,023	-	-	-	-	522,182
Depreciation (US $)	-	-	-	-	-	763	30,330	30,330	30,808	30,948	30,185	619	1,393	1,253	1,458	1,458	1,458	683	-	-	-	-	161,683
Taxable Income (US $)	-	-	-	-	-	-	11,579	12,508	13,021	13,033	16,695	45,933	42,765	43,932	47,541	45,160	44,990	23,340	-	-	-	-	360,499
Income Tax (@ 40.0%)	-	-	-	-	-	-	4,632	5,003	5,209	5,213	6,678	18,373	17,106	17,573	19,016	18,064	17,996	9,336	-	-	-	-	144,199
Net After-Tax Profit (US $)	-	-	-	-	-	-	6,948	7,505	7,813	7,820	10,017	27,560	25,659	26,359	28,525	27,096	26,994	14,004	-	-	-	-	216,299
Investment (US $)																							
Mine Development	11,169	11,169	21,757	35,204	24,196	8,564	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	112,058
Mining Equipment	-	-	-	-	-	6,176	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,071
Mine Dev. Equipment	-	-	2,393	700	-	-	-	2,393	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	8,578
Plant	-	-	-	-	15,161	15,161	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30,321
Total Investment	11,169	11,169	24,150	35,904	39,357	29,900	-	2,393	700	-	-	3,871	1,693	1,724	-	-	-	-	-	-	-	-	162,028
After-Tax Cash Flow (US $)	-11,169	-11,169	-24,150	-35,904	-39,357	-29,137	37,277	35,441	37,921	38,768	40,202	24,308	25,359	25,888	29,982	28,554	28,452	14,687	-	-	-	-	215,954

Table 97: Fox River Project – Cash flow - Mid-Size equipment, 10-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	984	984	984	984	984	984	984	984	984	984	984	623	-	-	-	-	11,448
Mine Development	-	-	-	-	-	19	19	34	23	38	14	19	13	41	19	7	45	2	-	-	-	-	295
Total Ore Production	-	-	-	-	-	19	1,003	1,018	1,007	1,022	999	1,003	997	1,025	1,003	992	1,029	625	-	-	-	-	11,743
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	77	77	77	77	77	77	77	77	77	76	77	76	-	-	-	-	77
Mill Recovery ($/tonne)	-	-	-	-	-	84	64	65	64	65	64	64	64	65	64	64	65	64	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,602	64,471	65,679	64,791	66,073	64,082	64,471	63,985	66,328	64,471	63,498	66,653	39,979	-	-	-	-	756,085
Treatment charge (10%)	-	-	-	-	-	160	6,447	6,568	6,479	6,607	6,408	6,447	6,398	6,633	6,447	6,350	6,665	3,998	-	-	-	-	75,609
Revenue (US $)	-	-	-	-	-	1,442	58,024	59,111	58,312	59,466	57,674	58,024	57,586	59,696	58,024	57,148	59,988	35,981	-	-	-	-	680,477
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	5,596	4,428	3,555	3,704	2,425	1,764	1,660	1,591	1,806	1,660	1,690	50	-	-	-	-	29,926
Mining	-	-	-	-	-	-	3,463	3,463	3,463	3,463	3,463	3,463	3,463	3,463	3,463	3,463	3,463	2,193	-	-	-	-	40,288
Hoisting	-	-	-	-	-	-	2,483	2,480	2,381	2,430	2,369	2,296	2,288	2,315	2,306	2,283	2,321	1,642	-	-	-	-	27,596
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511	511	-	-	-	-	6,649
Backfill	-	-	-	-	-	-	1,153	1,153	1,153	1,153	1,153	1 153	1,153	1,153	1,153	1,153	1,153	1,153	-	-	-	-	13,842
Processing	-	-	-	-	-	-	11,409	11,409	11,409	11,409	11,409	11,409	11,409	11,409	11,409	11,409	11,409	7,110	-	-	-	-	132,608
Total Operating Cost	-	-	-	-	-	511	24,616	23,445	22,473	22,671	21,331	20,597	20,485	20,443	20,649	20,480	20,548	12,660	-	-	-	-	250,909
Gross Income (US $)	-	-	-	-	-	930	33,409	35,667	35,839	36,795	36,343	37,427	37,101	39,253	37,375	36,668	39,440	23,321	-	-	-	-	429,567
Depreciation (US $)	-	-	-	-	-	930	30,085	30,085	30,424	30,564	29,633	479	1,253	1,253	1,458	1,458	1,458	683	-	-	-	-	159,761
Taxable Income (US $)	-	-	-	-	-	-	3,323	5,582	5,415	6,231	6,710	36,948	35,848	38,000	35,917	35,210	37,982	22,638	-	-	-	-	269,806
Income Tax (@ 40.0%)	-	-	-	-	-	-	1,329	2,233	2,166	2,493	2,684	14,779	14,339	15,200	14,367	14,084	15,193	9,055	-	-	-	-	107,922
Net After-Tax Profit (US $)	-	-	-	-	-	-	1,994	3,349	3,249	3,739	4,026	22,169	21,509	22,800	21,550	21,126	22,789	13,583	-	-	-	-	161,883
Investment (US $)																							
Mine Development	11,169	11,169	21,757	35,204	24,137	8,112	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	111,548
Mining Equipment	-	-	-	-	-	6,143	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,038
Mine Dev. Equipment	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	7,878
Plant	-	-	-	-	14,821	14,821	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	29,642
Total Investment	11,169	11,169	24,150	35,904	38,959	29,076	-	1,693	700	-	-	3,871	1,693	1,724	-	-	-	-	-	-	-	-	160,106
After-Tax Cash Flow (US $)	-11,169	-11,169	-24,150	-35,904	-38,959	-28,146	32,079	31,741	32,973	34,302	33,659	18,777	21,068	22,329	23,008	22,584	24,247	14,266	-	-	-	-	161,539

Table 98: Fox River Project – Cash flow - Mid-Size equipment, 15-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,065	1,065	1,065	1,065	1,065	1,065	1,065	1,065	1,065	308	-		-	-	-	-	9,892
Mine Development	-	-	-	-	-	45	55	87	87	74	78	84	54	89	77	-	-		-	-	-	-	730
Total Ore Production	-	-	-	-	-	45	1,120	1,152	1,152	1,139	1,143	1,149	1,119	1,154	1,142	308	-		-	-	-	-	10,622
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	92	92	92	92	92	92	92	92	92	91	-		-	-	-	-	92
Mill Recovery ($/tonne)	-	-	-	-	-	89	82	82	82	82	82	82	82	82	82	81	-		-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	4,049	91,567	94,367	94,367	93,265	93,570	94,159	91,467	94,575	93,535	25,033	-		-	-	-	-	869,955
Treatment charge (10%)	-	-	-	-	-	405	9,157	9,437	9,437	9,327	9,357	9,416	9,147	9,458	9,353	2,503	-		-	-	-	-	86,995
Revenue (US $)	-	-	-	-	-	3,644	82,411	84,930	84,930	83,939	84,213	84,743	82,320	85,118	84,181	22,530	-		-	-	-	-	782,959
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	8,413	8,930	8,941	7,315	7,208	7,233	7,190	6,596	4,789	-	-	-	-	-	-	-	66,615
Mining	-	-	-	-	-	-	3,553	3,553	3,553	3,553	3,553	3,553	3,553	3,553	3,553	1,026	-	-	-	-	-	-	33,004
Hoisting	-	-	-	-	-	-	2,776	2,823	2,811	2,731	2,721	2,723	2,711	2,656	2,579	1,129	-	-	-	-	-	-	25,659
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	-	-	-	-	-	-	5,626
Backfill	-	-	-	-	-	-	1,069	1,069	1,069	1,069	1,069	1,069	1,069	1,069	1,069	1,069	-	-	-	-	-	-	10,690
Processing	-	-	-	-	-	501	12,349	12,349	12,349	12,349	12,349	12,349	12,349	12,349	12,349	3,392	-	-	-	-	-	-	115,034
Total Operating Cost	-	-	-	-	-	1,013	28,671	29,236	29,234	27,528	27,412	27,439	27,383	26,734	24,850	7,128	-	-	-	-	-	-	256,628
Gross Income (US $)	-	-	-	-	-	2,631	53,739	55,695	55,696	56,410	56,801	57,305	54,937	58,383	59,331	15,402	-	-	-	-	-	-	526,331
Depreciation (US $)	-	-	-	-	-	2,631	31,776	31,776	32,255	32,255	29,764	619	1,393	1,393	1,598	1,458	-	-	-	-	-	-	166,916
Taxable Income (US $)	-	-	-	-	-	-	21,963	23,919	23,441	24,156	27,038	56,686	53,544	56,991	57,734	13,945	-	-	-	-	-	-	359,416
Income Tax (@ 40.0%)	-	-	-	-	-	-	8,785	9,567	9,377	9,662	10,815	22,674	21,418	22,796	23,094	5,578	-	-	-	-	-	-	143,766
Net After-Tax Profit (US $)	-	-	-	-	-	-	13,178	14,351	14,065	14,493	16,223	34,012	32,127	34,194	34,640	8,367	-	-	-	-	-	-	215,650
Investment (US $)																							
Mine Development	11,169	11,169	23,167	35,849	25,784	9,737	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	116,875
Mining Equipment	-	-	-	-	-	6,255	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,150
Mine Dev. Equipment	-	-	2,393	700	700	-	-	2,393	-	700	-	-	2,393	-	-	-	-	-	-	-	-	-	9,278
Plant	-	-	-	-	15,980	15,980	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	31,959
Total Investment	11,169	11,169	25,560	36,548	42,464	31,972	-	2,393	-	700	-	3,871	2,393	1,025	-	-	-	-	-	-	-	-	169,261
After-Tax Cash Flow (US $)	-11,169	-11,169	-25,560	-36,548	-42,464	-29,341	44,954	43,734	46,320	46,049	45,986	30,760	31,126	34,563	36,238	9,824	-	-	-	-	-	-	213,304

Table 99: Fox River Project – Cash flow - Mid-Size equipment, 15-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,014	1,014	1,014	1,014	1,014	1,014	1,014	1,014	1,014	1,014	913	-	-	-	-	-	11,054
Mine Development	-	-	-	-	-	17	38	42	42	37	23	38	63	42	21	-	-	-	-	-	-	-	365
Total Ore Production	-	-	-	-	-	17	1,053	1,056	1,056	1,051	1,037	1,053	1,077	1,056	1,035	1,014	913	-	-	-	-	-	11,419
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	88	88	88	88	87	88	88	88	87	87	87	-	-	-	-	-	88
Mill Recovery ($/tonne)	-	-	-	-	-	88	77	77	77	77	76	77	77	77	76	76	76	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,531	80,667	81,008	80,970	80,533	79,308	80,667	82,843	80,970	79,174	77,301	69,571	-	-	-	-	-	874,542
Treatment charge (10%)	-	-	-	-	-	153	8,067	8,101	8,097	8,053	7,931	8,067	8,284	8,097	7,917	7,730	6,957	-	-	-	-	-	87,454
Revenue (US $)	-	-	-	-	-	1,378	72,600	72,908	72,873	72,479	71,377	72,600	74,559	72,873	71,257	69,571	62,614	-	-	-	-	-	787,088
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	7,681	4,904	3,439	3,688	2,440	3,435	3,445	3,230	1,416	-	-		-	-	-	-	33,678
Mining	-	-	-	-	-	-	3,497	3,497	3,497	3,497	3,497	3,497	3,497	3,497	3,497	3,497	3,147		-	-	-	-	38,112
Hoisting	-	-	-	-	-	-	2,649	2,525	2,418	2,476	2,406	2,416	2,453	2,412	2,337	2,271	2,107		-	-	-	-	26,470
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511		-	-	-	-	6,138
Backfill	-	-	-	-	-	-	1,010	1,010	1,010	1,010	1 010	1,010	1,010	1,010	1,010	1,010	1,010		-	-	-	-	11,114
Processing	-	-	-	-	-	195	11,725	11,725	11,725	11,725	11,725	11,725	11,725	11,725	11,725	11,725	11,297		-	-	-	-	128,741
Total Operating Cost	-	-	-	-	-	706	27,074	24,172	22,600	22,907	21,589	22,595	22,642	22,385	20,497	19,014	18,072		-	-	-	-	244,254
Gross Income (US $)	-	-	-	-	-	671	45,526	48,735	50,273	49,573	49,788	50,005	51,917	50,488	50,760	50,557	44,542		-	-	-	-	542,834
Depreciation (US $)	-	-	-	-	-	671	30,570	30,570	30,909	31,049	30,377	479	1,253	1,253	1,458	1,458	1,458		-	-	-	-	161,503
Taxable Income (US $)	-	-	-	-	-	-	14,956	18,165	19,364	18,524	19,411	49,526	50,664	49,235	49,302	49,099	43,084	-	-	-	-	-	381,331
Income Tax (@ 40.0%)	-	-	-	-	-	-	5,982	7,266	7,746	7,410	7,764	19,811	20,266	19,694	19,721	19,640	17,234	-	-	-	-	-	152,532
Net After-Tax Profit (US $)	-	-	-	-	-	-	8,973	10,899	11,619	11,114	11,646	29,716	30,398	29,541	29,581	29,459	25,850	-	-	-	-	-	228,798
Investment (US $)																							
Mine Development	11,169	11,169	23,167	35,111	24,103	8,438	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	113,156
Mining Equipment	-	-	-	-	-	6,180	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,075
Mine Dev. Equipment	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-	7,878
Plant	-	-	-	-	15,211	15,211	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30,422
Total Investment	11,169	11,169	25,560	35,810	39,314	29,829	-	1,693	700	-	-	3,871	1,693	1,724	-	-	-	-	-	-	-	-	162,532
After-Tax Cash Flow (US $)	-11,169	-11,169	-25,560	-35,810	-39,314	-29,157	39,543	39,776	41,828	42,163	42,024	26,324	29,958	29,070	31,039	30,917	27,308	-	-	-	-	-	227,770

Table 100: Fox River Project – Cash flow - Mid-Size equipment, 15-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,016	1,016	1,016	1,016	1,016	1,016	1,013	1,016	1,016	1,016	829		-	-	-	-	10,983
Mine Development	-	-	-	-	-	17	38	40	42	33	11	28	19	7	38	-	-	-	-	-	-	-	274
Total Ore Production	-	-	-	-	-	17	1,054	1,056	1,057	1,049	1,027	1,044	1,032	1,023	1,053	1,016	829	-	-	-	-	-	11,257
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	81	81	81	81	81	81	81	81	81	81	81	-	-	-	-	-	81
Mill Recovery ($/tonne)	-	-	-	-	-	85	69	69	69	69	69	69	69	69	69	69	69	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,492	73,106	73,240	73,402	72,680	70,785	72,212	71,262	70,424	73,041	69,826	57,025	-	-	-	-	-	778,497
Treatment charge (10%)	-	-	-	-	-	149	7,311	7,324	7,340	7,268	7,079	7,221	7,126	7,042	7,304	6,983	5,702	-	-	-	-	-	77,850
Revenue (US $)	-	-	-	-	-	1,343	65,796	65,916	66,062	65,412	63,707	64,991	64,135	63,382	65,737	62,844	51,322	-	-	-	-	-	700,647
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	5,432	4,057	3,520	3,545	2,370	1,838	1,760	1,669	1,507	-	-	-	-	-	-	-	25,699
Mining	-	-	-	-	-	-	3,498	3,498	3,498	3,498	3,498	3,498	3,496	3,498	3,498	3,498	2,857	-	-	-	-	-	37,836
Hoisting	-	-	-	-	-	-	2,519	2,488	2,422	2,462	2,415	2,381	2,340	2,331	2,360	2,273	1,972	-	-	-	-	-	25,963
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511	-	-	-	-	-	6,138
Backfill	-	-	-	-	-	-	1,004	1,004	1,004	1,004	1,004	1,004	1,004	1,004	1,004	1,004	1,004	-	-	-	-	-	11,048
Processing	-	-	-	-	-	194	11,661	11,661	11,661	11,661	11,661	11,661	11,661	11,661	11,661	11,661	9,176	-	-	-	-	-	125,979
Total Operating Cost	-	-	-	-	-	705	24,625	23,220	22,617	22,682	21,459	20,894	20,773	20,675	20,542	18,948	15,521	-	-	-	-	-	232,662
Gross Income (US $)	-	-	-	-	-	638	41,170	42,696	43,445	42,730	42,247	44,097	43,363	42,707	45,195	43,895	35,801	-	-	-	-	-	467,985
Depreciation (US $)	-	-	-	-	-	638	30,057	30,057	30,396	30,535	29,898	479	1,253	1,253	1,318	1,318	1,318	-	-	-	-	-	158,517
Taxable Income (US $)	-	-	-	-	-	-	11,113	12,639	13,049	12,195	12,350	43,619	42,110	41,454	43,877	42,578	34,484	-	-	-	-	-	309,467
Income Tax (@ 40.0%)	-	-	-	-	-	-	4,445	5,056	5,220	4,878	4,940	17,447	16,844	16,582	17,551	17,031	13,794	-	-	-	-	-	123,787
Net After-Tax Profit (US $)	-	-	-	-	-	-	6,668	7,584	7,829	7,317	7,410	26,171	25,266	24,872	26,326	25,547	20,690	-	-	-	-	-	185,680
Investment (US $)																							
Mine Development	11,169	11,169	23,106	34,988	23,551	6,773	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	110,755
Mining Equipment	-	-	-	-	-	6,173	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,068
Mine Dev. Equipment	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	-	-	-	-	-	-	-	-	-	7,179
Plant	-	-	-	-	15,132	15,132	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	30,264
Total Investment	11,169	11,169	25,499	35,688	38,683	28,078	-	1,693	700	-	-	3,871	1,693	1,025	-	-	-	-	-	-	-	-	159,266
After-Tax Cash Flow (US $)	-11,169	-11,169	-25,499	-35,688	-38,683	-27,440	36,725	35,947	37,525	37,852	37,307	22,779	24,825	25,101	27,644	26,864	22,008	-	-	-	-	-	184,932

Table 101: Fox River Project – Cash flow - Mid-Size equipment, 20-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,066	1,066	1,066	1,066	1,066	1,066	1,066	1,066	675	-	-	-	-	-	-	-	9,201
Mine Development	-	-	-	-	-	44	72	94	50	61	88	89	75	56	-	-	-	-	-	-	-	-	629
Total Ore Production	-	-	-	-	-	44	1,138	1,160	1,115	1,126	1,154	1,155	1,141	1,122	675	-	-	-	-	-	-	-	9,830
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	93	93	93	93	93	93	93	93	93	-	-	-	-	-	-	-	93
Mill Recovery ($/tonne)	-	-	-	-	-	89	83	83	83	83	83	83	83	83	83	-	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	3,891	94,567	96,532	92,567	93,529	96,008	96,092	94,863	93,159	55,809	-	-	-	-	-	-	-	817,018
Treatment charge (10%)	-	-	-	-	-	389	9,457	9,653	9,257	9,353	9,601	9,609	9,486	9,316	5,581	-	-	-	-	-	-	-	81,702
Revenue (US $)	-	-	-	-	-	3,502	85,111	86,878	83,310	84,176	86,408	86,483	85,377	83,843	50,228	-	-	-	-	-	-	-	735,316
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	7,782	6,346	6,874	6,918	7,081	6,574	5,827	3,184	-	-	-	-	-	-	-	-	50,586
Mining	-	-	-	-	-	-	3,554	3,554	3,554	3,554	3,554	3,554	3,554	3,554	2,251	-	-	-	-	-	-	-	30,682
Hoisting	-	-	-	-	-	-	2,796	2,658	2,684	2,711	2,709	2,684	2,628	2,499	1,723	-	-	-	-	-	-	-	23,093
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	-	-	-	-	-	-	-	5,115
Backfill	-	-	-	-	-	-	952	952	952	952	952	952	952	952	952	-	-	-	-	-	-	-	8,564
Processing	-	-	-	-	-	472	12,335	12,335	12,335	12,335	12,335	12,335	12,335	12,335	7,317	-	-	-	-	-	-	-	106,467
Total Operating Cost	-	-	-	-	-	983	27,930	26,356	26,909	26,981	27,142	26,611	25,807	23,035	12,754	-	-	-	-	-	-	-	224,508
Gross Income (US $)	-	-	-	-	-	2,519	57,181	60,522	56,400	57,196	59,265	59,872	59,570	60,808	37,474	-	-	-	-	-	-	-	510,808
Depreciation (US $)	-	-	-	-	-	2,519	31,964	31,964	32,303	32,443	30,064	619	1,393	1,393	1,458	-	-	-	-	-	-	-	166,117
Taxable Income (US $)	-	-	-	-	-	-	25,217	28,558	24,098	24,753	29,202	59,254	58,178	59,416	36,017	-	-	-	-	-	-	-	344,691
Income Tax (@ 40.0%)	-	-	-	-	-	-	10,087	11,423	9,639	9,901	11,681	23,701	23,271	23,766	14,407	-	-	-	-	-	-	-	137,876
Net After-Tax Profit (US $)	-	-	-	-	-	-	15,130	17,135	14,459	14,852	17,521	35,552	34,907	35,649	21,610	-	-	-	-	-	-	-	206,814
Investment (US $)																			-	-			
Mine Development	11,169	11,169	23,863	35,807	25,871	9,972	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	117,851
Mining Equipment	-	-	-	-	-	6,253	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,148
Mine Dev. Equipment	-	-	2,393	700	700	-	-	1,693	700	700	-	-	1,693	-	-	-	-	-	-	-	-	-	8,578
Plant	-	-	-	-	15,962	15,962	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	31,924
Total Investment	11,169	11,169	26,256	36,507	42,533	32,187	-	1,693	700	700	-	3,871	1,693	1,025	-	-	-	-	-	-	-	-	169,501
After-Tax Cash Flow (US $)	-11,169	-11,169	-26,256	-36,507	-42,533	-29,668	47,094	47,406	46,062	46,595	47,585	32,300	34,606	36,017	23,068	-	-	-	-	-	-	-	203,431

Table 102: Fox River Project – Cash flow - Mid-Size equipment, 20-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,056	1,056	1,056	1,056	1,056	1,056	1,056	1,056	1,056	1,056	106		-	-	-	-	10,666
Mine Development	-	-	-	-	-	16	45	44	19	50	41	44	17	39	-	-	-		-	-	-	-	315
Total Ore Production	-	-	-	-	-	16	1,101	1,100	1,075	1,106	1,097	1,100	1,073	1,095	1,056	1,056	106		-	-	-	-	10,981
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	89	89	89	89	89	89	89	89	89	89	89		-	-	-	-	89
Mill Recovery ($/tonne)	-	-	-	-	-	88	78	78	78	78	78	78	78	78	78	78	78		-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,422	86,169	86,096	83,889	86,644	85,854	86,137	83,689	85,622	82,230	82,230	8,223		-	-	-	-	858,206
Treatment charge (10%)	-	-	-	-	-	142	8,617	8,610	8,389	8,664	8,585	8,614	8,369	8,562	8,223	8,223	822	-	-	-	-	-	85,821
Revenue (US $)	-	-	-	-	-	1,280	77,553	77,486	75,501	77,979	77,269	77,523	75,320	77,059	74,007	74,007	7,401	-	-	-	-	-	772,385
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	5,580	4,571	3,664	3,347	1,728	1,747	1,753	1,729	1,300	-	-	-	-	-	-	-	25,419
Mining	-	-	-	-	-	-	3,543	3,543	3,543	3,543	3,543	3,543	3,543	3,543	3,543	3,543	354	-	-	-	-	-	35,786
Hoisting	-	-	-	-	-	-	2,591	2,569	2,514	2,492	2,437	2,440	2,408	2,431	2,374	2,339	803	-	-	-	-	-	25,397
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511	-	-	-	-	-	6,138
Backfill	-	-	-	-	-	-	925	925	925	925	925	925	925	925	925	925	925	-	-	-	-	-	10,172
Processing	-	-	-	-	-	176	12,014	12,014	12,014	12,014	12,014	12,014	12,014	12,014	12,014	12,014	1,153	-	-	-	-	-	121,474
Total Operating Cost	-	-	-	-	-	688	25,165	24,133	23,172	22,833	21,159	21,181	21,155	21,153	20,668	19,333	3,746	-	-	-	-	-	224,386
Gross Income (US $)	-	-	-	-	-	592	52,387	53,353	52,328	55,147	56,110	56,343	54,165	55,906	53,339	54,674	3,655	-	-	-	-	-	547,999
Depreciation (US $)	-	-	-	-	-	592	30,754	30,754	31,092	31,232	30,640	479	1,253	1,253	1,318	1,318	1,318	-	-	-	-	-	162,001
Taxable Income (US $)	-	-	-	-	-	-	21,634	22,599	21,236	23,915	25,470	55,864	52,912	54,653	52,021	53,357	2,337	-	-	-	-	-	385,998
Income Tax (@ 40.0%)	-	-	-	-	-	-	8,653	9,040	8,494	9,566	10,188	22,346	21,165	21,861	20,808	21,343	935	-	-	-	-	-	154,399
Net After-Tax Profit (US $)	-	-	-	-	-	-	12,980	13,559	12,742	14,349	15,282	33,518	31,747	32,792	31,213	32,014	1,402	-	-	-	-	-	231,599
Investment (US $)																							
Mine Development	11,169	11,169	23,863	35,069	24,062	7,993	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	113,325
Mining Equipment	-	-	-	-	-	6,215	-	-	-	-	-	3,871	-	1,025	-	-	-	-	-	-	-	-	11,110
Mine Dev. Equipment	-	-	2,393	700	-	-	-	1,693	700	-	-	-	1,693	-	-	-	-	-	-	-	-	-	7,179
Plant	-	-	-	-	15,568	15,568	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	31,135
Total Investment	11,169	11,169	26,256	35,769	39,629	29,776	-	1,693	700	-	-	3,871	1,693	1,025	-	-	-	-	-	-	-	-	162,749
After-Tax Cash Flow (US $)	-11,169	-11,169	-26,256	-35,769	-39,629	-29,184	43,734	42,620	43,134	45,581	45,922	30,127	31,307	33,020	32,530	33,332	2,720	-	-	-	-	-	230,850

Table 103: Fox River Project – Cash flow - Mid-Size equipment, 20-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,057	1,057	1,057	1,057	1,057	1,057	1,057	1,057	1,057	1,057	159	-	-	-	-	-	10,728
Mine Development	-	-	-	-	-	16	45	44	-	-	30	25	28	22	-	-	-	-	-	-	-	-	210
Total Ore Production	-	-	-	-	-	16	1,102	1,101	1,057	1,057	1,087	1,082	1,085	1,078	1,057	1,057	159	-	-	-	-	-	10,937
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	83	83	83	83	83	83	83	83	83	83	83	-	-	-	-	-	83
Mill Recovery ($/tonne)	-	-	-	-	-	86	72	72	71	71	71	71	71	71	71	71	71	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,390	78,934	78,861	75,084	75,084	77,704	77,240	77,472	76,937	75,084	75,084	11,263	-	-	-	-	-	780,135
Treatment charge (10%)	-	-	-	-	-	139	7,893	7,886	7,508	7,508	7,770	7,724	7,747	7,694	7,508	7,508	1,126	-	-	-	-	-	78,014
Revenue (US $)	-	-	-	-	-	1,251	71,040	70,975	67,575	67,575	69,933	69,516	69,725	69,243	67,575	67,575	10,136	-	-	-	-	-	702,122
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	3,513	3,555	2,082	1,879	1,803	1,757	1,736	844	-	-	-	-	-	-	-	-	17,168
Mining	-	-	-	-	-	-	3,544	3,544	3,544	3,544	3,544	3,544	3,544	3,544	3,544	3,544	532	-	-	-	-	-	35,973
Hoisting	-	-	-	-	-	-	2,471	2,515	2,463	2,418	2,437	2,419	2,422	2,389	2,340	2,340	888	-	-	-	-	-	25,103
Ventilation	-	-	-	-	-	511	511	511	511	511	511	511	511	511	511	511	511		-	-	-	-	6,138
Backfill	-	-	-	-	-	-	917	917	917	917	917	917	917	917	917	917	917		-	-	-	-	10,092
Processing	-	-	-	-	-	175	11,927	11,927	11,927	11,927	11,927	11,927	11,927	11,927	11,927	11,927	1,716		-	-	-	-	121,166
Total Operating Cost	-	-	-	-	-	687	22,884	22,970	21,446	21,198	21,140	21,076	21,059	20,133	19,241	19,241	4,565		-	-	-	-	215,639
Gross Income (US $)	-	-	-	-	-	564	48,156	48,005	46,130	46,377	48,793	48,440	48,666	49,110	48,335	48,335	5,571		-	-	-	-	486,483
Depreciation (US $)	-	-	-	-	-	564	30,369	30,369	30,708	30,848	30,284	479	1,253	914	979	979	979		-	-	-	-	158,725
Taxable Income (US $)	-	-	-	-	-	-	17,786	17,635	15,422	15,529	18,510	47,962	47,414	48,196	47,356	47,356	4,592		-	-	-	-	327,757
Income Tax (@ 40.0%)	-	-	-	-	-	-	7,115	7,054	6,169	6,212	7,404	19,185	18,965	19,278	18,942	18,942	1,837		-	-	-	-	131,103
Net After-Tax Profit (US $)	-	-	-	-	-	-	10,672	10,581	9,253	9,318	11,106	28,777	28,448	28,918	28,413	28,413	2,755		-	-	-	-	196,654
Investment (US $)																							
Mine Development	11,169	11,169	23,863	35,069	23,781	6,579	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	111,629
Mining Equipment	-	-	-	-	-	6,205	-	-	-	-	-	3,871	-	1,025	-	-	-		-	-	-	-	11,100
Mine Dev. Equipment	-	-	2,393	700	-	-	-	1,693	700	-	-	-	-	-	-	-	-		-	-	-	-	5,485
Plant	-	-	-	-	15,460	15,460	-	-	-	-	-	-	-	-	-	-	-		-	-	-	-	30,921
Total Investment	11,169	11,169	26,256	35,769	39,241	28,244	-	1,693	700	-	-	3,871	-	1,025	-	-	-		-	-	-	-	159,135
After-Tax Cash Flow (US $)	-11,169	-11,169	-26,256	-35,769	-39,241	-27,679	41,041	39,257	39,261	40,166	41,389	25,385	29,701	28,807	29,392	29,392	3,734		-	-	-	-	196,244

Table 104: Fox River Project – Cash flow - Large equipment, 10-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	1,155	847	-	-	-	-	-	-	11,243
Mine Development	-	-	-	-	-	26	119	103	127	109	111	117	100	109	122	37	-	-	-	-	-	-	1,079
Total Ore Production	-	-	-	-	-	26	1,274	1,258	1,282	1,264	1,266	1,272	1,256	1,264	1,277	884	-	-	-	-	-	-	12,322
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	84	84	84	84	84	84	84	84	84	84	-	-	-	-	-	-	84
Mill Recovery ($/tonne)	-	-	-	-	-	86	73	73	73	73	73	73	73	73	73	72	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	2,209	92,888	91,533	93,599	92,029	92,261	92,767	91,323	92,029	93,149	63,826	-	-	-	-	-	-	897,631
Treatment charge (10%)	-	-	-	-	-	221	9,289	9,153	9,360	9,203	9,226	9,279	9,132	9,203	9,315	6,383	-	-	-	-	-	-	89,763
Revenue (US $)	-	-	-	-	-	1,988	83,599	82,380	84,239	82,826	83,034	83,508	82,190	82,826	83,834	57,443	-	-	-	-	-	-	807,868
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	11,570	11,340	11,884	11,586	12,123	11,470	9,807	9,763	7,669	2,043	-	-	-	-	-	-	99,255
Mining	-	-	-	-	-	-	3,678	3,678	3,678	3,678	3,678	3,678	3,678	3,678	3,678	2,697	-	-	-	-	-	-	35,797
Hoisting	-	-	-	-	-	-	3,103	3,060	3,079	3,078	3,107	3,068	2,967	2,967	2,847	2,063	-	-	-	-	-	-	29,340
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	599	-	-	-	-	-	-	6,584
Backfill	-	-	-	-	-	-	1,354	1,354	1,354	1,354	1,354	1,354	1,354	1,354	1,354	993	-		-	-	-	-	13,175
Processing	-	-	-	-	-	265	13,218	13,218	13,218	13,218	13,218	13,218	13,218	13,218	13,218	9,178	-		-	-	-	-	128,409
Total Operating Cost	-	-	-	-	-	864	33,522	33,248	33,811	33,512	34 079	33,387	31,622	31,578	29,365	17,572	-		-	-	-	-	312,560
Gross Income (US $)	-	-	-	-	-	1,124	50,077	49,132	50,428	49,313	48,956	50,122	50,568	51,248	54,469	39,871	-		-	-	-	-	495,308
Depreciation (US $)	-	-	-	-	-	1,124	35,893	35,893	36,508	36,709	35,786	1,951	1,951	1,951	1,750	1,549	-		-	-	-	-	191,066
Taxable Income (US $)	-	-	-	-	-	-	14,184	13,239	13,920	12,605	13,170	48,171	48,617	49,297	52,719	38,322	-		-	-	-	-	304,243
Income Tax (@ 40.0%)	-	-	-	-	-	-	5,674	5,296	5,568	5,042	5,268	19,268	19,447	19,719	21,087	15,329	-		-	-	-	-	121,697
Net After-Tax Profit (US $)	-	-	-	-	-	-	8,510	7,943	8,352	7,563	7,902	28,903	29,170	29,578	31,631	22,993	-	-	-	-	-	-	182,546
Investment (US $)																							
Mine Development	11,574	11,574	24,982	44,441	29,031	11,546	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	133,148
Mining Equipment	-	-	-	-	-	7,103	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,775
Mine Dev. Equipment	-	-	3,075	1,004	1,004	-	-	3,075	1,004	1,004	-	-	3,075	-	-	-	-	-	-	-	-	-	13,243
Plant	-	-	-	-	17,065	17,065	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	34,130
Total Investment	11,574	11,574	28,057	45,445	47,100	35,714	-	3,075	1,004	1,004	4,672	-	3,075	-	-	-	-	-	-	-	-	-	102,296
After-Tax Cash Flow (US $)	-11,574	-11,574	-28,057	-45,445	-47,100	-34,590	44,403	40,761	43,856	43,268	39,016	30,854	28,046	31,529	33,381	24,543	-	-	-	-	-	-	181,315

Table 105: Fox River Project – Cash flow - Large equipment, 10-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Slopes	-	-	-	-	-	-	1,255	1,255	1,255	1,255	1,255	1,255	1,255	1,255	1,255	1,193	-	-	-	-	-	-	12,491
Mine Development	-	-	-	-	-	28	46	63	52	50	81	57	47	74	43	-	-	-	-	-	-	-	539
Total Ore Production	-	-	-	-	-	28	1,302	1,318	1,307	1,305	1,336	1,312	1,303	1,329	1,298	1,193	-	-	-	-	-	-	13,030
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	76	77	76	76	77	77	76	77	76	75	-		-	-	-	-	76
Mill Recovery ($/tonne)	-	-	-	-	-	84	64	64	64	64	64	64	64	64	64	63	-		-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	2,365	83,165	84,560	83,630	83,454	86,061	84,047	83,263	85,453	82,864	75,334	-		-	-	-	-	834,197
Treatment charge (10%)	-	-	-	-	-	236	8,317	8,456	8,363	8,345	8,606	8,405	8,326	8,545	8,286	7,533	-		-	-	-	-	83,420
Revenue (US $)	-	-	-	-	-	2,128	74,849	76,104	75,267	75,109	77,455	75,642	74,936	76,908	74,578	67,801	-		-	-	-	-	750,777
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	8,115	8,112	6,490	6,219	6,074	4,787	3,859	3,743	2,605	-	-		-	-	-	-	50,003
Mining	-	-	-	-	-	-	3,788	3,788	3,788	3,788	3,788	3,788	3,788	3,788	3,788	3,599	-		-	-	-	-	37,694
Hoisting	-	-	-	-	-	-	3,017	3,032	2,977	2,933	2,948	2,861	2,796	2,812	2,741	2,514	-		-	-	-	-	28,630
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	599	-		-	-	-	-	6,584
Backfill	-	-	-	-	-	-	1,386	1,386	1,386	1,386	1,386	1,386	1,386	1,386	1,386	1,317	-		-	-	-	-	13,795
Processing	-	-	-	-	-	293	13,514	13,514	13,514	13,514	13,514	13,514	13,514	13,514	13,514	12,382	-		-	-	-	-	134,298
Total Operating Cost	-	-	-	-	-	892	30,418	30,431	28,754	28,438	28,309	26,935	25,942	25,842	24,632	20,411	-		-	-	-	-	271,004
Gross Income (US $)	-	-	-	-	-	1,237	44,431	45,673	46,514	46,671	49,146	48,707	48,995	51,066	49,945	47,390	-	-	-	-	-	-	479,774
Depreciation (US $)	-	-	-	-	-	1,237	35,360	35,360	35,975	36,176	34,940	1,750	1,750	1,549	1,549	1,549	-	-	-	-	-	-	187,197
Taxable Income (US $)	-	-	-	-	-	-	9,071	10,313	10,539	10,494	14,207	46,956	47,244	49,517	48,396	45,841	-	-	-	-	-	-	292,577
Income Tax (@ 40.0%)	-	-	-	-	-	-	3,628	4,125	4,215	4,198	5,683	18,783	18,898	19,807	19,358	18,336	-	-	-	-	-	-	117,031
Net After-Tax Profit (US $)	-	-	-	-	-	-	5,442	6,188	6,323	6,297	8,524	28,174	28,347	29,710	29,038	27,504	-	-	-	-	-	-	175,546
Investment (US $)																							
Mine Development	11,574	11,574	24,982	44,441	28,688	9,468	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	130,728
Mining Equipment	-	-	-	-	-	7,139	-	-	-	-	4,672		-	-	-	-	-	-	-	-	-	-	11,810
Mine Dev. Equipment	-	-	3,075	1,004	-	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	-	-	-	-	-	11,234
Plant	-	-	-	-	17,427	17,427	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	34,855
Total Investment	11,574	11,574	28,057	45,445	46,116	34,035	-	3,075	1,004	-	4,672	-	2,071	1,004	-	-	-	-	-	-	-	-	188,628
After-Tax Cash Flow (US $)	-11,574	-11,574	-28,057	-45,445	-46,116	-32,798	40,803	38,472	41,294	42,473	38,792	29,924	28,026	30,256	30,587	29,054	-	-	-	-	-	-	174,115

Table 106: Fox River Project – Cash flow - Large equipment, 10-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,345	1,345	1,345	1,345	1,345	1,345	1,345	1,345	1,345	688	-	-	-	-	-	-	12,794
Mine Development	-	-	-	-	-	26	47	21	43	45	47	45	24	24	38	-	-	-	-	-	-	-	360
Total Ore Production	-	-	-	-	-	26	1,392	1,366	1,388	1,390	1,392	1,390	1,369	1,369	1,383	688	-	-	-	-	-	-	13,154
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	66	65	66	66	66	66	65	65	66	65	-		-	-	-	-	66
Mill Recovery ($/tonne)	-	-	-	-	-	80	53	52	53	53	53	53	52	52	52	52	-		-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	2,076	73,289	71,198	72,923	73,091	73,289	73,091	71,395	71,395	72,542	35,523	-		-	-	-	-	689,811
Treatment charge (10%)	-	-	-	-	-	208	7,329	7,120	7,292	7,309	7,329	7,309	7,140	7,140	7,254	3,552	-		-	-	-	-	68,981
Revenue (US $)	-	-	-	-	-	1,869	65,960	64,078	65,630	65,782	65,960	65,782	64,256	64,256	65,288	31,971	-		-	-	-	-	620,830
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	7,946	4,115	4,048	4,133	3,895	2,145	1,986	2,003	1,263	-	-		-	-	-	-	31,533
Mining	-	-	-	-	-	-	3,887	3,887	3,887	3,887	3,887	3,887	3,887	3,887	3,887	1,987	-		-	-	-	-	36,973
Hoisting	-	-	-	-	-	-	3,159	2,992	2,943	3,002	2,934	2,881	2,844	2,844	2,834	1,717	-		-	-	-	-	28,148
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	599	-		-	-	-	-	6,584
Backfill	-	-	-	-	-	-	1,438	1,438	1,438	1,438	1,438	1,438	1,438	1,438	1,438	735	-		-	-	-	-	13,675
Processing	-		-	-	-	-	13,975	13,975	13,975	13,975	13,975	13,975	13,975	13,975	13,975	6,900	-		-	-	-	-	132,677
Total Operating Cost	-	-	-	-	-	599	31,003	27,006	26,889	27,033	26,727	24,925	24,728	24,746	23,996	11,937	-		-	-	-	-	249,590
Gross Income (US $)	-	-	-	-	-	1,270	34,956	37,072	38,741	38,749	39,233	40,857	39,527	39,509	41,291	20,034	-		-	-	-	-	371,240
Depreciation (US $)	-	-	-	-	-	1,270	35,613	35,613	36,027	36,228	34,958	1,549	1,549	1,549	1,349	1,349	-		-	-	-	-	187,055
Taxable Income (US $)	-	-	-	-	-	- -	657	1,459	2,714	2,520	4,275	39,308	37,978	37,960	39,943	18,685	-		-	-	-	-	184,185
Income Tax (@ 40.0%)	-	-	-	-	-	- -	263	584	1,085	1,008	1,710	15,723	15,191	15,184	15,977	7,474	-		-	-	-	-	73,674
Net After-Tax Profit (US $)	-	-	-	-	-	- -	394	875	1,628	1,512	2,565	23,585	22,787	22,776	23,966	11,211	-		-	-	-	-	110,511
Investment (US $)																							
Mine Development	11,574	11,574	24,982	44,441	28,794	9,439	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	130,804
Mining Equipment	-	-	-	-	-	7,194	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,866
Mine Dev. Equipment	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	2,071	-	-	-	-	-	-	-	-	-	9,226
Plant	-	-	-	-	17,994	17,994	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	35,988
Total Investment	11,574	11,574	28,057	45,445	46,788	34,627	-	2,071	1,004	-	4,672	-	2,071	-	-	-	-	-	-	-	-	-	187,884
After-Tax Cash Flow (US $)	-11,574	-11,574	-28,057	-45,445	-46,788	-33,356	35,219	34,417	36,651	37,740	32,851	25,134	22,265	24,325	25,314	12,560	-	-	-	-	-	-	109,682

Table 107: Fox River Project – Cash flow - Large equipment, 15-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,115	1,115	1,115	1,115	1,115	1,115	1,115	1,115	1,115	173	-		-	-	-	-	10,210
Mine Development	-	-	-	-	-	45	137	52	100	111	92	116	62	134	40	-	-		-	-	-	-	888
Total Ore Production	-	-	-	-	-	45	1,252	1,167	1,215	1,226	1,207	1,231	1,177	1,249	1,156	173	-		-	-	-	-	11,099
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	88	87	88	88	88	88	87	88	87	87	-	-	-	-	-	-	88
Mill Recovery ($/tonne)	-	-	-	-	-	88	77	76	77	77	77	77	77	77	76	76	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	3,908	96,682	89,230	93,409	94,410	92,701	94,844	90,116	96,431	88,230	13,173	-	-	-	-	-	-	853,134
Treatment charge (10%)	-	-	-	-	-	391	9,668	8,923	9,341	9,441	9,270	9,484	9,012	9,643	8,823	1,317	-	-	-	-	-	-	85,313
Revenue (US $)	-	-	-	-	-	3,517	87,013	80,307	84,068	84,969	83,431	85,360	81,104	86,788	79,407	11,856	-	-	-	-	-	-	767,821
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	10,099	10,026	10,161	9,979	9,981	10,018	8,471	5,637	115	-	-	-	-	-	-	-	74,488
Mining	-	-	-	-	-	-	3,634	3,634	3,634	3,634	3,634	3,634	3,634	3,634	3,634	565	-	-	-	-	-	-	33,270
Hoisting	-	-	-	-	-	-	2,955	2,912	2,944	2,899	2,800	2,813	2,797	2,795	2,526	906	-	-	-	-	-	-	26,347
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	599	-	-	-	-	-	-	6,584
Backfill	-	-	-	-	-	-	1,117	1,117	1,117	1,117	1,117	1,117	1,117	1,117	1,117	174	-	-	-	-	-	-	10,231
Processing	-	-	-	-	-	458	12,864	12,864	12,864	12,864	12,864	12,864	12,864	12,864	12,864	1,782	-	-	-	-	-	-	118,013
Total Operating Cost	-	-	-	-	-	1,057	31,267	31,152	31,319	31,091	30,995	31,044	29,481	26,646	20,855	4,025	-	-	-	-	-	-	268,932
Gross Income (US $)	-	-	-	-	-	2,460	55,747	49,155	52,750	53,878	52,436	54,316	51,623	60,143	58,552	7,830	-	-	-	-	-	-	498,889
Depreciation (US $)	-	-	-	-	-	2,460	35,870	35,870	36,485	36,686	34,225	1,750	1,750	1,750	1,549	1,549	-	-	-	-	-	-	189,946
Taxable Income (US $)	-	-	-	-	-	-	19,877	13,286	16,265	17,192	18,210	52,565	49,873	58,393	57,002	6,281	-	-	-	-	-	-	308,943
Income Tax (@ 40.0%)	-	-	-	-	-	-	7,951	5,314	6,506	6,877	7,284	21,026	19,949	23,357	22,801	2,512	-	-	-	-	-	-	123,577
Net After-Tax Profit (US $)	-	-	-	-	-	-	11,926	7,971	9,759	10,315	10,926	31,539	29,924	35,036	34,201	3,769	-	-	-	-	-	-	185,366
Investment (US $)																							
Mine Development	11,574	11,574	26,742	44,326	28,757	10,975	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	133,948
Mining Equipment	-	-	-	-	-	7,060	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,732
Mine Dev. Equipment	-	-	3,075	1,004	1,004	-	-	3,075	1,004	-	-	-	3,075	-	-	-	-	-	-	-	-	-	12,239
Plant	-	-	-	-	16,629	16,629	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	33,258
Total Investment	11,574	11,574	29,817	45,330	46,390	34,664	-	3,075	1,004	-	4,672	-	3,075	-	-	-	-	-	-	-	-	-	191,176
After-Tax Cash Flow (US $)	-11,574	-11,574	-29,817	-45,330	-46,390	-32,204	47,796	40,766	45,240	47,001	40,480	33,289	28,599	36,786	35,751	5,318	-	-	-	-	-	-	184,136

Table 108: Fox River Project – Cash flow - Large equipment, 15-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,168	1,168	1,168	1,168	1,168	1,168	1,168	1,168	1,168	1,129	-		-	-	-	-	11,642
Mine Development	-	-	-	-	-	29	43	48	42	45	53	60	74	46	5	-	-		-	-	-	-	444
Total Ore Production	-	-	-	-	-	29	1,211	1,216	1,210	1,213	1,222	1,229	1,242	1,214	1,173	1,129	-	-	-	-	-	-	12,087
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	81	81	81	81	81	81	81	81	80	80	-	-	-	-	-	-	81
Mill Recovery ($/tonne)	-	-	-	-	-	85	69	69	69	69	69	69	69	69	69	68	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	2,456	83,669	84,067	83,557	83,801	84,554	85,150	86,280	83,885	80,422	77,329	-	-	-	-	-	-	835,169
Treatment charge (10%)	-	-	-	-	-	246	8,367	8,407	8,356	8,380	8,455	8,515	8,628	8,388	8,042	7,733	-	-	-	-	-	-	83,517
Revenue (US $)	-	-	-	-	-	2,210	75,302	75,661	75,201	75,421	76,099	76,635	77,652	75,496	72,379	69,596	-	-	-	-	-	-	751,652
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	8,320	5,759	3,906	4,121	3,930	3,848	3,795	3,381	115	-	-	-	-	-	-	-	37,177
Mining	-	-	-	-	-	-	3,692	3,692	3,692	3,692	3,692	3,692	3,692	3,692	3,692	3,569	-	-	-	-	-	-	36,799
Hoisting	-	-	-	-	-	-	2,885	2,765	2,652	2,716	2,673	2 073	2,676	2,633	2,483	2,414	-	-	-	-	-	-	26,568
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	599	-	-	-	-	-	-	6,584
Backfill	-	-	-	-	-	-	1,120	1,120	1,120	1,120	1,120	1,120	1,120	1,120	1,120	1,083	-	-	-	-	-	-	11,162
Processing	-	-	-	-	-	30[illegible]	12,890	12,890	12,890	12,890	12,890	12,390	12,890	12,890	12,890	12,017	-	-	-	-	-	-	128,330
Total Operating Cost	-	-	-	-	-	905	29,506	26,824	24,859	25,137	24,904	24,821	24,771	24,314	20,899	19,681	-	-	-	-	-	-	246,620
Gross Income (US $)	-	-	-	-	-	1,305	45,796	48,836	50,342	50,284	51,196	51,814	52,881	51,182	51,481	49,915	-	-	-	-	-	-	505,033
Depreciation (US $)	-	-	-	-	-	1,305	35,154	35,154	35,568	35,769	34,464	1,549	1,549	1,549	1,349	1,349	-	-	-	-	-	-	184,759
Taxable Income (US $)	-	-	-	-	-	-	10,643	13,682	14,774	14,515	16,732	50,265	51,332	49,633	50,132	48,567	-	-	-	-	-	-	320,273
Income Tax (@ 40.0%)	-	-	-	-	-	-	4,257	5,473	5,910	5,806	6,693	20,106	20,533	19,853	20,053	19,427	-	-	-	-	-	-	128,109
Net After-Tax Profit (US $)	-	-	-	-	-	-	6,386	8,209	8,865	8,709	10,039	30,159	30,799	29,780	30,079	29,140	-	-	-	-	-	-	192,164
Investment (US $)																							
Mine Development	11,574	11,574	26,742	44,326	27,950	9,139	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	131,304
Mining Equipment	-	-	-	-	-	7,064	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,735
Mine Dev. Equipment	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	2,071	-	-	-	-	-	-	-	-	-	9,226
Plant	-	-	-	-	16,661	16,661	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	33,322
Total Investment	11,574	11,574	29,817	45,330	44,611	32,863	-	2,071	1,004	-	4,672	-	2,071	-	-	-	-	-	-	-	-	-	185,588
After-Tax Cash Flow (US $)	-11,574	-11,574	-29,817	-45,330	-44,611	-31,558	41,539	41,292	43,429	44,478	39,831	31,708	30,277	31,329	31,428	30,489	-	-	-	-	-	-	191,335

Table 109: Fox River Project – Cash flow - Large equipment, 15-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,404	1,404	1,404	1,404	1,404	1,404	1,404	1,404	639	-	-	-	-	-	-	-	11,874
Mine Development	-	-	-	-	-	22	47	82	52	40	36	-	41	12	-	-	-	-	-	-	-	-	333
Total Ore Production	-	-	-	-	-	22	1,452	1,487	1,456	1,445	1,440	1,404	1,446	1,417	639	-	-	-	-	-	-	-	12,208
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	72	73	72	72	72	71	72	71	71	-	-	-	-	-	-	-	72
Mill Recovery ($/tonne)	-	-	-	-	-	82	59	60	59	59	59	58	59	59	58	-	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,769	85,886	88,742	86,248	85,313	84,948	81,988	85,371	82,994	37,323	-	-	-	-	-	-	-	720,582
Treatment charge (10%)	-	-	-	-	-	177	8,589	8,874	8,625	8,531	8,495	8,199	8,537	8,299	3,732	-	-	-	-	-	-	-	72,058
Revenue (US $)	-	-	-	-	-	1,592	77,298	79,868	77,623	76,781	76,453	73,789	76,834	74,695	33,590	-	-	-	-	-	-	-	648,524
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	6,055	5,672	5,172	3,590	1,981	1,842	1,861	283	-	-	-	-	-	-	-	-	26,456
Mining	-	-	-	-	-	-	3,953	3,953	3,953	3,953	3,953	3,953	3,953	3,953	1,799	-	-	-	-	-	-	-	33,420
Hoisting	-	-	-	-	-	-	3,133	3,127	3,125	3,061	2,949	2,902	2,959	2,867	1,641	-	-	-	-	-	-	-	25,764
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	-	-	-	-	-	-	-	5,985
Backfill	-	-	-	-	-	-	1,262	1,262	1,262	1,262	1,262	1,262	1,262	1,262	575	-	-	-	-	-	-	-	10,675
Processing	-	-	-	-	-	214	14,396	14,396	14,396	14,396	14,396	14,396	14,396	14,396	6,339	-	-	-	-	-	-	-	121,719
Total Operating Cost	-	-	-	-	-	812	29,398	29,008	28,507	26,860	25,139	24,954	25,029	23,360	10,952	-	-	-	-	-	-	-	224,020
Gross Income (US $)	-	-	-	-	-	780	47,900	50,860	49,116	49,922	51,314	48,835	51,805	51,335	22,638	-	-	-	-	-	-	-	424,504
Depreciation (US $)	-	-	-	-	-	780	35,823	35,823	36,438	36,438	35,658	1,549	1,549	934	934	-	-	-	-	-	-	-	185,927
Taxable Income (US $)	-	-	-	-	-	-	12,077	15,037	12,678	13,484	15,656	47,286	50,255	50,400	21,704	-	-	-	-	-	-	-	238,577
Income Tax (@ 40.0%)	-	-	-	-	-	-	4,831	6,015	5,071	5,393	6,262	18,914	20,102	20,160	8,681	-	-	-	-	-	-	-	95,431
Net After-Tax Profit (US $)	-	-	-	-	-	-	7,246	9,022	7,607	8,090	9,393	28,371	30,153	30,240	13,022	-	-	-	-	-	-	-	143,146
Investment (US $)																							
Mine Development	11,574	11,574	26,665	44,172	28,525	8,260	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	130,771
Mining Equipment	-	-	-	-	-	7,245	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,917
Mine Dev. Equipment	-	-	3,075	1,004	-	-	-	3,075	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7,155
Plant	-	-	-	-	18,509	18,509	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	37,019
Total Investment	11,574	11,574	29,740	45,177	47,035	34,014	-	3,075	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	186,862
After-Tax Cash Flow (US $)	-11,574	-11,574	-29,740	-45,177	-47,035	-33,235	43,069	41,770	44,045	44,528	40,380	29,921	31,703	31,175	13,957	-	-	-	-	-	-	-	142,212

Table 110: Fox River Project – Cash flow - Large equipment, 20-m ore, 25-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,348	1,348	1,348	1,348	1,348	1,348	1,303	-	-	-	-	-	-	-	-	-	9,394
Mine Development	-	-	-	-	-	13	149	116	105	120	119	119	25	-	-	-	-	-	-	-	-	-	766
Total Ore Production	-	-	-	-	-	13	1,497	1,465	1,453	1,468	1,468	1,467	1,329	-	-	-	-	-	-	-	-	-	10,160
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	90	90	90	90	90	90	89	-	-	-	-	-	-	-	-	-	90
Mill Recovery ($/tonne)	-	-	-	-	-	88	79	79	79	79	79	79	79	-	-	-	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,163	118,822	115,970	114,925	116,254	116,227	116,181	104,388	-	-	-	-	-	-	-	-	-	803,928
Treatment charge (10%)	-	-	-	-	-	116	11,882	11,597	11,493	11,625	11,623	11,618	10,439	-	-	-	-	-	-	-	-	-	80,393
Revenue (US $)	-	-	-	-	-	1,046	106,940	104,373	103,433	104,628	104,604	104,563	93,949	-	-	-	-	-	-	-	-	-	723,536
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	9,984	9,966	10,040	8,766	7,850	6,940	2,017	-	-	-	-	-	-	-	-	-	55,562
Mining	-	-	-	-	-	-	3,891	3,891	3,891	3,891	3,891	3,891	3,761	-	-	-	-	-	-	-	-	-	27,106
Hoisting	-	-	-	-	-	-	3,299	3,275	3,279	3,217	3,168	3,096	2,778	-	-	-	-	-	-	-	-	-	22,113
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	-	-	-	-	-	-	-	-	-	4,788
Backfill	-	-	-	-	-	-	1,137	1,137	1,137	1,137	1,137	1,137	1,100	-	-	-	-	-	-	-	-	-	7,924
Processing	-	-	-	-	-	128	14,540	14,540	14,540	14,540	14,540	14,540	12,903	-	-	-	-	-	-	-	-	-	100,272
Total Operating Cost	-	-	-	-	-	727	33,450	33,408	33,486	32,150	31,186	30,203	23,156	-	-	-	-	-	-	-	-	-	217,766
Gross Income (US $)	-	-	-	-	-	320	73,490	70,965	69,947	72,478	73,418	74,360	70,792	-	-	-	-	-	-	-	-	-	505,769
Depreciation (US $)	-	-	-	-	-	320	37,320	37,320	37,935	37,935	37,816	1,750	1,750	-	-	-	-	-	-	-	-	-	192,147
Taxable Income (US $)	-	-	-	-	-	-	36,170	33,645	32,011	34,543	35,602	72,609	69,042	-	-	-	-	-	-	-	-	-	313,622
Income Tax (@ 40.0%)	-	-	-	-	-	-	14,468	13,458	12,805	13,817	14,241	29,044	27,617	-	-	-	-	-	-	-	-	-	125,449
Net After-Tax Profit (US $)	-	-	-	-	-	-	21,702	20,187	19,207	20,726	21,361	43,566	41,425	-	-	-	-	-	-	-	-	-	188,173
Investment (US $)																							
Mine Development	11,574	11,574	27,611	45,049	29,872	11,202	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	136,882
Mining Equipment	-	-	-	-	-	7,262	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,934
Mine Dev. Equipment	-	-	3,075	1,004	1,004	-	-	3,075	-	1,004	-	-	-	-	-	-	-	-	-	-	-	-	9,163
Plant	-	-	-	-	18,686	18,686	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	37,372
Total Investment	11,574	11,574	30,686	46,053	49,562	37,151	-	3,075	-	1,004	4,672	-	-	-	-	-	-	-	-	-	-	-	195,352
After-Tax Cash Flow (US $)	-11,574	-11,574	-30,686	-46,053	-49,562	-36,831	59,022	54,432	57,142	57,657	54,505	45,316	43,176	-	-	-	-	-	-	-	-	-	184,969

Table 111: Fox River Project – Cash flow - Large equipment, 20-m ore, 50-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,326	1,326	1,326	1,326	1,326	1,326	1,326	1,326	475	-	-	-	-	-	-	-	11,080
Mine Development	-	-	-	-	-	20	55	53	44	69	84	47	12	-	-	-	-	-	-	-	-	-	383
Total Ore Production	-	-	-	-	-	20	1,381	1,378	1,370	1,394	1,409	1,373	1,338	1,326	475	-	-	-	-	-	-	-	11,463
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	84	84	83	84	84	83	83	83	83	-	-	-	-	-	-	-	83
Mill Recovery ($/tonne)	-	-	-	-	-	86	72	72	72	72	72	72	72	71	71	-	-		-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	1,702	99,441	99,197	98,474	100,576	101,865	98,708	95,726	94,671	33,924	-	-	-	-	-	-	-	824,284
Treatment charge (10%)	-	-	-	-	-	170	9,944	9,920	9,847	10,058	10,187	3,871	9,573	9,467	3,392	-	-	-	-	-	-	-	82,428
Revenue (US $)	-	-	-	-	-	1,532	89,497	89,277	88,627	90,518	91,679	88,838	86,154	85,204	30,531	-	-		-	-	-	-	741,856
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	5,580	4,571	3,664	3,347	1,728	1,747	1,753	1,729	1,300	-	-		-	-	-	-	25,419
Mining	-	-	-	-	-	-	3,843	3,843	3,843	3,843	3,843	3,843	3,843	3,843	1,390	-	-		-	-	-	-	32,133
Hoisting	-	-	-	-	-	-	3,028	3,021	2,987	2,930	2,935	2,898	2,746	2,724	1,381	-	-		-	-	-	-	24,651
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	-	-		-	-	-	-	5,985
Backfill	-	-	-	-	-	-	1,091	1,091	1,091	1,091	1,091	1,091	1,091	1,091	391	-	-		-	-	-	-	9,117
Processing	-	-	-	-	-	200	13,988	13,988	13,988	13,988	13,988	13,988	13,988	13,988	4,811	-	-		-	-	-	-	116,914
Total Operating Cost	-	-	-	-	-	799	28,128	27,111	26,172	25,797	24,183	24,165	24,020	23,972	9,872	-	-		-	-	-	-	214,219
Gross Income (US $)	-	-	-	-	-	733	61,368	62,165	62,455	64,721	67,496	64,673	62,134	61,231	20,659	-	-		-	-	-	-	527,637
Depreciation (US $)	-	-	-	-	-	733	35,496	35,496	36,111	36,111	35,378	1,549	1,549	934	934	-	-		-	-	-	-	184,293
Taxable Income (US $)	-	-	-	-	-	-	25,873	26,670	26,344	28,610	32,118	63,123	60,585	60,297	19,725	-	-	-	-	-	-	-	343,344
Income Tax (@ 40.0%)	-	-	-	-	-	-	10,349	10,668	10,538	11,444	12,847	25,249	24,234	24,119	7,890	-	-	-	-	-	-	-	137,338
Net After-Tax Profit (US $)	-	-	-	-	-	-	15,524	16,002	15,806	17,166	19,271	37,874	36,351	36,178	11,835	-	-	-	-	-	-	-	206,006
Investment (US $)																							
Mine Development	11,574	11,574	27,611	44,274	28,134	7,019	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	130,186
Mining Equipment	-	-	-	-	-	7,196	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,868
Mine Dev. Equipment	-	-	3,075	1,004	-	-	-	3,075	-	-	-	-	-	-	-	-	-	-	-	-	-	-	7,155
Plant	-	-	-	-	18,009	18,009	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	36,018
Total Investment	11,574	11,574	30,686	45,278	46,143	32,224	-	3,075	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	185,227
After-Tax Cash Flow (US $)	-11,574	-11,574	-30,686	-45,278	-46,143	-31,490	51,019	48,422	51,918	53,277	49,977	39,423	37,900	37,113	12,769	-	-	-	-	-	-	-	205,072

Table 112: Fox River Project – Cash flow - Large equipment, 20-m ore, 75-m levels

Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	Total
Ore Production (tonnes)																							
Stopes	-	-	-	-	-	-	1,394	1,394	1,394	1,394	1,394	1,394	1,394	1,394	225	-	-	-	-	-	-	-	11,375
Mine Development	-	-	-	-	-	34	47	46	7	51	51	18	-	-	-	-	-	-	-	-	-	-	255
Total Ore Production	-	-	-	-	-	34	1,441	1,440	1,400	1,445	1,445	1,412	1,394	1,394	225	-	-	-	-	-	-	-	11,631
Revenue Calculation																							
Value of Ore ($/tonne)	-	-	-	-	-	100	75	75	75	76	76	75	75	75	75	-	-	-	-	-	-	-	75
Mill Recovery ($/tonne)	-	-	-	-	-	83	63	63	62	63	63	62	62	62	62	-	-	-	-	-	-	-	
Gross Sales (US $)	-	-	-	-	-	2,855	90,571	90,447	87,168	90,902	90,902	88,141	86,619	86,619	13,955	-	-	-	-	-	-	-	728,178
Treatment charge (10%)	-	-	-	-	-	285	9,057	9,045	8,717	9,090	9,090	8,814	8,662	8,662	1,396	-	-	-	-	-	-	-	72,818
Revenue (US $)	-	-	-	-	-	2,569	81,514	81,402	78,451	81,811	81,811	79,326	77,957	77,957	12,560	-	-	-	-	-	-	-	655,360
Operating Cost (US $)																							
Mine Development	-	-	-	-	-	-	3,513	3,555	2,082	1,879	1,803	1,757	1,736	844	-	-	-	-	-	-	-	-	17,168
Mining	-	-	-	-	-	-	3,919	3,919	3,919	3,919	3,919	3,919	3,919	3,919	637	-	-	-	-	-	-	-	31,987
Hoisting	-	-	-	-	-	-	2,995	2,988	3,074	3,015	2,988	2,868	2,831	2,831	986	-	-	-	-	-	-	-	24,578
Ventilation	-	-	-	-	-	599	599	599	599	599	599	599	599	599	599	-	-	-	-	-	-	-	5,985
Backfill	-	-	-	-	-	-	1,116	1,116	1,116	1,116	1,116	1,116	1,116	1,116	180	-	-	-	-	-	-	-	9,110
Processing	-	-	-	-	-	340	14,290	14,290	14,290	14,290	14,290	14,290	14,290	14,290	2,226	-	-	-	-	-	-	-	116,885
Total Operating Cost	-	-	-	-	-	938	26,432	26,467	25,080	24,817	24,714	24,548	24,490	23,598	4,628	-	-	-	-	-	-	-	205,713
Gross Income (US $)	-	-	-	-	-	1,631	55,083	54,936	53,371	56,994	57,097	54,779	53,467	54,359	7,932	-	-	-	-	-	-	-	449,648
Depreciation (US $)	-	-	-	-	-	1,631	35,261	35,261	35,675	35,876	34,245	1,549	1,549	1,135	934	-	-	-	-	-	-	-	183,117
Taxable Income (US $)	-	-	-	-	-	-	19,822	19,675	17,696	21,118	22,852	53,229	51,917	53,224	6,997	-	-	-	-	-	-	-	266,531
Income Tax (@ 40.0%)	-	-	-	-	-	-	7,929	7,870	7,079	8,447	9,141	21,292	20,767	21,290	2,799	-	-	-	-	-	-	-	106,612
Net After-Tax Profit (US $)	-	-	-	-	-	-	11,893	11,805	10,618	12,671	13,711	31,937	31,150	31,934	4,198	-	-	-	-	-	-	-	159,919
Investment (US $)																							
Mine Development	11,574	11,574	27,611	44,274	26,161	7,039	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	128,233
Mining Equipment	-	-	-	-	-	7,232	-	-	-	-	4,672	-	-	-	-	-	-	-	-	-	-	-	11,904
Mine Dev. Equipment	-	-	3,075	1,004	-	-	-	2,071	1,004	-	-	-	-	-	-	-	-	-	-	-	-	-	7,155
Plant	-	-	-	-	18,379	18,379	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	36,759
Total Investment	11,574	11,574	30,686	45,278	44,540	32,651	-	2,071	1,004	-	4,672	-	-	-	-	-	-	-	-	-	-	-	184,051
After-Tax Cash Flow (US $)	-11,574	-11,574	-30,686	-45,278	-44,540	-31,020	47,154	44,994	45,289	48,547	43,284	33,487	32,700	33,070	5,133	-	-	-	-	-	-	-	158,984